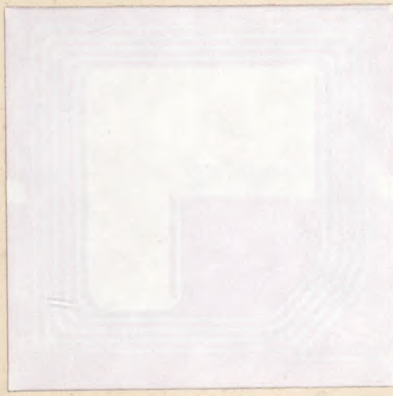

THE MINERAL INDUSTRY

ITS

STATISTICS, TECHNOLOGY AND TRADE

DURING

1906

FOUNDED BY RICHARD P. ROTHWELL

EDITED BY

WALTER RENTON INGALLS

Editor of the Engineering and Mining Journal; Member American Institute of Mining Engineers; Member Institution of Mining and Metallurgy; Member American Chemical Society; Member Society of Chemical Industry, Etc.

VOLUME XV

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CONTRIBUTORS TO VOLUME XV.

ADDICKS, LAWRENCE.
Superintendent, United States Metals Refining Co., Chrome, N. J.
The Chrome Plant of the U. S. Metals Refining Company.

ATWATER, C. G.
Engineer, United Coke and Gas Company, New York.
Ammonia and Ammonium Sulphate.
By-product Coke Ovens.

AUSTIN, L. S.
Professor of Metallurgy Michigan College of Mines, Houghton Mich.
Progress in the Metallurgy of Copper.

BAIN, H. FOSTER.
Director, Illinois Geological Survey, Urbana, Ill.
Petroleum in Illinois.

BARTLETT, C. O.
President, C. O. Bartlett & Snow Company, Cleveland, Oh io
Manufacture of Calcined Plaster.

BEASON, L. H.
Correspondent of the *Engineering and Mining Journal*, Salt Lake City, Utah
Copper Mining in Utah.
Gold and Silver Mining in Utah.
Lead Mining in Utah

BELL, ROBERT N.
State Mine Inspector, Boise, Idaho
Coal Mining in Idaho.
Copper Mining in Idaho.
Gold and Silver Mining in Idaho.
Lead Mining in Idaho.
Phosphate Mining in Idaho.
Sapphire Mining in Idaho.

BETTS, ANSON G.
Consulting Chemist and Metallurgist, Troy, N. Y.
Manufacture of Hydrofluoric Acid.
Sodium Transmission Lines.

BOHN, C. A.
Mining Engineer, Mexico City, Mexico.
Coal Mining in Mexico.
Copper Mining in Mexico.
Gold and Silver Mining in Mexico
Lead Mining in Mexico.

COLCORD, F. F.
Chief Chemist, American Smelting and Refining Company, Maurer, N. J.
Sampling and Assaying.

COLLINS, GEORGE E.
Consulting Mining Engineer, Denver, Colo.
Glod and Silver Mining in Colorado.

COURTIS, WM. M.
Consulting Mining Engineer, Detroit, Mich.
Potassium Salts in the United States.

CUSHING, GEORGE H.
Correspondent, *Engineering and Mining Journal*, Cleveland, O.
The Cleveland Coal Market in 1906.
The Cleveland Iron Market in 1906.

DORR, J. V. N.
Consulting Mining Engineer, Lead, S. D.
Gold and Silver Mining in South Dakota.

DOUGLAS, JAMES.
President, Copper Queen Consolidated Mining Company, New York City.
Copper Mining in Arizona in 1906.

FOHS, F. JULIUS.
Assistant Geologist, Kentucky Geological Survey, Lexington, Kentucky.
Fluorspar.

FRIEDMAN, L. W.
Correspondent, *Engineering and Mining Journal*, Birmingham, Ala.
Coal Mining in Alabama in 1906.
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FULTON, CHARLES H.
President, South Dakota School of Mines, Rapid City, South Dakota.
Cyanidation in the United States and Mexico.

GEORGE, HAROLD C.
Instructor, School of Mines and Mining, Western University of Pennsylvania, Allegheny, Penn.
Petroleum.

GIBSON, THOMAS W.
Deputy Minister of Mines of Ontario, Toronto, Ont.
The Cobalt District.

GLIDDEN, JOHN T.
Editorial Staff of *Engineering and Mining Journal*.
Antimony.
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GREENAWALT, WILLIAM A.
Mining Engineer, Denver, Colo.
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HAWORTH, ERASMUS.
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HOBART, FREDERICK.
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HOFMAN, H. O.
Professor of Metallurgy, Massachusetts Institute of Technology, Boston, Mass.
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HUTCHINS, J. P.
Consulting Mining Engineer, New York City.
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HUTCHINSON, W. SPENCER.
Consulting Mining Engineer, Boston, Mass.
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INGALLS, W. R.
Editor of the *Engineering and Mining Journal.*
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JACOBS, E.
Editor, *British Columbia Mining Record,* Victoria, B. C.
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JAMES, ALFRED.
Vice-President, Institution of Mining and Metallurgy, and Consulting Engineer, London, England.
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JUDD, E. K.
Mining Engineer with Stillwell & Gladding, New York City.
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Consulting Mining Engineer, Socorro, N. M.
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KUNZ, GEORGE FREDERICK.
Gem Expert of Tiffany & Company, New York.
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LESLEY, ROBERT W.
Vice-President, American Society for Testing Materials, Philadelphia, Penn.
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LEWIS, JAMES B.
General Manager, Anchor Tin Mining Company, Lottah, Tasmania.
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LOCKE, C. E.
Assistant Professor of Mining, Massachusetts Institute of Technology, Boston, Mass.
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LOTKA, ALFRED J.
Chemist with General Chemical Company, Laurel Hill, Queens, New York.
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LUTY, S. F.
Correspondent, *Engineering and Mining Journal,* Pittsburg, Penn.
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MANCE, F. S.
Dept. of Mines, Sydney, N. S. W., Australia.
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MEEKS, REGINALD.
Editorial Staff, *Engineering and Mining Journal.*
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MOORE, E. W.
General Manager, Galena Iron Works, Platteville, Wis.
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MORRISON, E.
Correspondent, *Engineering and Mining Journal*, Chicago, Ill.
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MUNROE, CHARLES E.
Professor of Chemistry, George Washington University, Washington, D. C.
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NEWLAND, D. H.
Assistant State Geologist, Albany, N. Y.
Emery in New York.
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OTAGAWA, M.
Assistant General Manager, Furukawa Mining Company, Tokio, Japan.
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PARSONS, FLOYD W.
Associate Editor, *Engineering and Mining Journal.*
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PREUS, WILHELM.
Mining Engineer, Estacion Vacar, Prov. de Cardoba, Spain.
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PURDUE, A. H.
Professor of Geology and Mining, University of Arkansas, Fayetteville, Ark.
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RICHARDS, JOSEPH W.
Professor of Metallurgy, Lehigh University, So. Bethlehem, Penn.
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RICHARDS, R. H.
Professor of Mining, Massachusetts Institute of Technology, Boston, Mass.
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RUHM, H. D.
Mining Engineer, Mt. Pleasant, Tenn.
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SHOCKLEY, W. H.
Mining Engineer, Tonopah, Nev.
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ST. CHARLES, B. E.
Correspondent of the *Engineering and Mining Journal*, Butte, Mont.
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STOUGHTON, BRADLEY.
Adjunct Professor of Metallurgy, School of Mines, Columbia University, New York City.
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TARR, RAYMOND P.
Mining Engineer, Northwestern Improvement Co., Tacoma, Wash.
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TONE, F. J.
Works Manager, Carborundum Company, Niagara Falls, N. Y.
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VAN MATER, J. A.
General Manager, Bertha Mineral Co., Pulaski, Va.
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VAN ZWALUWENBURG, A.
Editorial Staff, *Engineering and Mining Journal.*
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WALKER, EDWARD.
Associate Editor, *Engineering and Mining Journal.*
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WATSON, THOMAS L.
State Geologist, Blacksburg, Va.
Coal Mining in Virginia.

WHEELER, H. A.
Consulting Mining Engineer, St. Louis, Mo.
Lead Mining in Southeastern Missouri.

WILKINSON, W. FISCHER.
Consulting Mining Engineer, London, England.
Gold Mining in the Transvaal.
Gold Mining in Rhodesia.
Gold Mining in West Africa.

WOODBRIDGE, DWIGHT E.
Mining Engineer, Duluth, Minn.
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YALE, C. G.
Associate Editor, *Engineering and Mining Journal*, San Francisco, Cal.
Gold and Silver Mining in California.
Quicksilver Mining in California.

ZOOK, JESSE A.
Statistician, Joplin, Mo.
Zinc and Lead Mining in the Joplin District.

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OCHER AND IRON OXIDE PIGMENTS

TALC AND SOAPSTONE.

TANTALUM.

TIN.

TUNGSTEN.

URANIUM.

VANADIUM.

VALUES OF FOREIGN COINS

COUNTRY	Standard	Monetary Unit	Value in Terms of U.S. Gold Dollar	Coins
Argentine Republic	Gold	Peso	$0.965	God: argentine ($4.824) and ½ argentine. Silver: peso and divisions.
Austria-Hungary	Gold	Crown	.203	Gold: 10 and 20 crowns. Silver: 1 and 5 crowns.
Belgium	Gold	Franc	.193	Gold: 10 and 20 francs. Silver: 5 francs.
Bolivia	Silver	Boliviano	.422	Silver: boliviano and divisions.
Brazil	Gold	Milreis	.546	Gold: 5, 10 and 20 milreis. Silver: ½, 1, and 2 milreis.
British Possessions, N. A. (except Newf'nd).	Gold	Dollar	1.000	
Central Amer. States—				
Costa Rica	Gold	Colon	.465	Gold: 2, 5, 1 and 20 colons ($9.307). Silver: 5, 10 25, and 5, centimos.
British Honduras	Gold	Dollar	1.000	
Guatemala, Honduras, Nicaragua, Salvador	Silver	Peso	.422	Silver: peso and divisions.
Chile	Gold	Peso	.365	Gold: escudo ($1.825), doubloon ($3.650), and condor ($7.300). Silver: peso and divisions.
China	Silver	Tael: Amoy	.691	
		Tael: Canton	.689	
		Tael: Chefoo	.661	
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		Tael: Tientsen	.670	
		Dollar: Hongkong	.455	
		Dollar: British	.455	
		Dollar: Mexican Chopped	.458	
Colombia	Gold	Dollar	1.000	Gold: condor ($9.647) and double condor. Silver peso.
Denmark	Gold	Crown	.268	Gold: 10 and 20 crowns.
Ecuador	Gold	Sucre	.487	Gold: 10 sucres ($4.8665). Silver: sucre and divisions.
Egypt	Gold	Pound (100 piastres)	443.9	Gold: pound (100 piasters), 5, 10, 20, and 50 piasters Silver: 1, 2, 5, 10, and 20 piasters.
Finland	Gold	Mark	.193	Gold: 20 marks ($3.859), 10 marks ($1.93).
France	Gold	Franc	.193	Gold: 5, 10, 20, 50, and 100, francs. Silver: 5 francs
German Empire	Gold	Mark	.238	Gold: 5, 10, and 20 marks.
Great Britain	Gold	Pound sterling	4.866½	Gold: sovereign (pound sterling) and ½ sovereign.
Greece	Gold	Drachma	.193	Gold: 5, 10, 20, 50, and 100 drachmas. Silver: 5 drachmas.
Haiti	Gold	Gourde	.965	Gold: 1, 2, 5, and 10 gourdes. Silver: gourde and divisions.
India	Gold	Pound sterling*	4.866½	Gold: sovereign (pound sterling). Silver: rupee and divisions.
Italy	Gold	Lira	.193	Gold: 5, 10, 20, 50, and 100 lira. Silver: 5 lira.

NOTE.—The coins of silver-standard countries are valued by their pure silver contents, at the average market price of silver for the three months preceding July 1, 1905.

*The sovereign is the standard coin of India, but the rupee ($0.3244⅓) is the money of account, current at 15 to the sovereign.

COUNTRY	Standard	Monetary Unit	Value in Terms of U. S. Gold Dollar	Coins
Japan	Gold	Yen	.498	Gold: 5, 10, and 20 yen. Silver: 10, 20, and 50 sen.
Liberia	Gold	Dollar	1.000	
Mexico	Gold	Peso†	.498	Gold: 5 and 10 pesos. Silver: dollar‡ (or peso) and divisions.
Netherlands	Gold	Florin	.402	Gold: 10 florins. Silver: ½, 1, and 2½ florins.
Newfoundland	Gold	Dollar	1.014	Gold: 2 dollars ($2.027).
Norway	Gold	Crown	.268	Gold: 10 and 20 crowns.
Panama	Gold	Balboa	1.000	Gold: 1, 2½, 5, 10, and 20 balboas. Silver: peso and divisions.
Persia	Silver	Kran	.078	Gold: ½, 1 and 2 tomans ($3.409). Silver: ¼, ½, 1, 2, and 5 krans.
Peru	Gold	Sol	.487	Gold: libra ($4.8665). Silver: sol and divisions.
Philippine Islands	Gold	Peso	.500	Silver: peso, 10, 20, and 50 centavos.
Portugal	Gold	Milreis	1.080	Gold: 1, 2, 5, and 10 milreis.
Russia	Gold	Ruble	.515	Gold: 5, 7½, 10, and 15 rubles. Silver: 5, 10, 15, 20, 25, 50, and 100 copecks.
Spain	Gold	Peseta	.193	Gold: 25 pesetas. Silver: 5 pesetas.
Sweden	Gold	Crown	.268	Gold: 10 and 20 crowns.
Switzerland	Gold	Franc	.193	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
Turkey	Gold	Piaster	.044	Gold: 25, 50, 100, 250, and 500 piasters.
Uruguay	Gold	Peso	1.034	Gold: peso. Silver: peso and divisions.
Venezuela	Gold	Bolivar	.193	Gold: 5, 10, 20, 50, and 100 bolivars. Silver: 5 bolivars.

† Seventy-five centigrams fine gold.
‡ Value in Mexico, $0.498.

INTRODUCTION.

In the preparation of the statistics for this volume, the figures previously reported for 1905, and in some cases for earlier years, have been revised in the light of later and more minute investigation, in accordance with our regular practice; therefore it is important for all who have occasion to refer to them to observe the caution to *use always the figures in the latest volume of* The Mineral Industry. There are no statistical reports of this nature which are absolutely correct, owing to the practical impossibility of obtaining accurate reports from all the producers in some extensive and greatly subdivided industries, the absence of records on the part of many producers, which prevents them from making returns, the unwillingness of a few to give their figures, and the confusion as to the stage in which many products are to be reported. The last difficulty is especially likely to lead to errors in values, some producers estimating the worth of their product at the pit's mouth, and others reporting it in a more or less advanced state of completion, including thus not only the cost of carriage, but also the cost of manipulation. These difficulties appear not only in our own statistics, but also in the statistics reported by various governments. In our own work, however, we make a practice of going backward and correcting figures previously reported, whenever mistakes are discovered by subsequent investigation. In estimating values, we are disposed to use actual market prices rather than the values reported by the producers themselves, which are apt to be misleading for the reasons mentioned above.

For many of the statistics relating to the mineral production of the United States in 1906 and previous years, we are indebted to the United States Geological Survey, and for the production of gold and silver in the United States to George E. Roberts, Director of the Mint. Acknowledgment is due, also, to various State geological surveys and statistical bureaus for information incorporated in this volume. In the text and footnotes to the various tables, we have generally credited such information to the proper sources, but this acknowledgment may stand for any unintentional oversights. The same acknowledgment is due with respect to the foreign statistics, which we state always as officially reported by the respective governments, when such reports are available.

In publishing this volume of The Mineral Industry at a comparatively early date in the year following that to which it especially relates, the editorial work upon it having been completed at the end of May, it has been impossible to collect statistics for all the substances of mineral

production in the United States, but the omissions are generally in the cases of substances of minor importance. The statistics for foreign countries are given in all cases for the latest year available. In the preparation of THE MINERAL INDUSTRY there is always a temptation to postpone the conclusion of the editorial work pending the receipt of missing figures, but it has been considered that May 31 is a reasonable date at which to draw the line, with a view to combining the maximum of completeness with the maximum of promptness.

Some of the statistics reported in this volume are preliminary, and subject to revision. It is our belief, however, that statistics of reasonable commercial accuracy, promptly published, are of greater value to technology and trade than are statistics, corrected to the last unit, which are published a year or two late. However, it has been necessary to use these approximations in only a few instances, and it is hoped that most of the figures which are to be found in this book will prove to require only slight, if any, revision.

PRODUCTION OF ORES AND MINERALS IN THE UNITED STATES.

Products.	Measures.	1905		1906	
		Quantity.	Value.	Quantity.	Value.
Antimony ore	Sh. T.	*Nil.*	....	295	$44,250
Asbestos	Sh. T.	3,100	$126,300	1,695	20,565
Asphaltum (*u*)	Sh. T.	115,267	758,153	116,653	1,066,019
Barytes	Sh. T.	53,252	196,041	63,486	252,719
Bruxite	Lg. T.	47,991	203,960	78,331	352,490
Chlome ore	Lg. T.	40	600	180	1,800
Coal, anthracite	Sh. T.	78,731,523	178,788,244	72,209,566	166,307,002
Coal, bituminous	Sh. T.	308,344,613	354,543,505	341,629,113	400,550,951
Diatomacious earth	Sh. T.	10,977	64,637	....	....
Emery	Sh. T.	2,315	19,667	2,147	22,780
Feldspar (*u*)	Sh. T.	35,419	226,157	72,656	401,531
Flint (*u*)	Sh. T.	51,145	104,109	66,697	243,012
Fluorspar	Sh. T.	39,600	232,452	34,683	201,481
Fuller's earth	Sh. T.	25,745	186,816	28,000	237,950
Garnet	Sh. T.	3,694	114,695	5,404	179,548
Graphite, amorphous	Sh. T.	21,953	80,639	....	....
Graphite, crystalline	lb.	4,260,656	170,426	4,894,483	170,866
Gypsum (*u*)	Sh. T.	1,043,202	821,967	1,540,585	3,837,975
Iron ore	Lg. T.	44,578,456	94,768,122	49,237,129	107,091,574
Limestone flux	Lg. T.	14,098,000	6,739,200	15,486,139	7,339,125
Magnesite	Sh. T.	3,933	224,980	(*e*)4,000	240,000
Manganese ore (*d*)	Lg. T.	863,663	1,681,472	....	....
Mica, sheet (*u*)	lb.	851,000	185,900	1,423,100	252,248
Mica, scrap (*u*)	Sh. T.	856	15,255	1,489	22,742
Monazite (*u*)	lb.	1,352,418	163,908	846,175	152,312
Petroleum, crude	Bbl. (*i*)	139,889,210	118,905,828	131,771,505	80,277,279
Phosphate rock	Lg. T.	1,933,286	9,713,296	2,052,742	12,342,741
Pumice	Sh. T.	1,832	5,540	....	....
Pyrites	Lg. T.	224,980	752,936	225,045	767,866
Quartz, crystalline	Sh. T.	19,039	88,118	....	....
Salt (*u*)	Bbl. (*k*)	25,966,122	6,095,922	28,172,380	6,658,350
Sand, glass	Sh. T.	1,030,334	1,083,730	....	....
Slate, roofing	Squares (*f*)	1,241,227	4,574,550	....	5,668,346
Soda, natural	Sh. T.	(*e*)12,000	18,000	....	....
Sulphur	Lg. T.	215,000	4,742,900	285,000	6,056,250
Talc, common	Sh. T.	40,134	637,062	....	....
Talc, fibrous	Sh. T.	67,000	469,000	64,200	541,600
Tungsten ore (*u*)	Sh. T.	834	257,493	1,096	442,784
Whetstones and oilstones (*u*)	....	....	244,346	....	268,070
Zinc ore	Sh. T.	795,698	15,596,457	905,175	17,250,420
Total enumerated	....	....	$803,602,383	....	$819,262,646

PRODUCTION OF SECONDARY MINERALS AND CHEMICALS IN THE UNITED STATES.

Product.	Measures.	1905		1906	
		Quantity.	Value.	Quantity.	Value.
Alundum	℔.	3,612,000	$252,840	4,331,233	$303,186
Ammonium Sulphate	Sh. T.	65,296	4,108,424	75,000	4,674,750
Arsenic	℔.	1,545,400	50,225	1,663,000	83,150
Borax	Sh. T.	46,334	1,019,158	....	....
Bromine	℔.	899,434	139,432	1,229,000	184,350
Carborundum	℔.	5,596,280	599,628	6,225,280	622.528
Cement, nat. hyd. (*u*)	Bbl. (*g*)	4,473,049	2,413,052	3,935,151	2,362,140
Cement, portland (*u*)	Bbl. (*h*)	35,246,812	33,245,867	46,610,822	51,240,652
Cement, slag (*u*)	Bbl. (*t*)	382,447	272,614	481,224	412,912
Coke	Sh. T.	28,490,611	72,299,694	32,690,362	86,887,392
Copper sulphate (*c*)	℔'	53,718,996	2,417,355	50,925,932	3,157,408
Copperas	Sh. T.	21,103	147,721	22,839	228,390
Crushed steel	℔.	812,000	56,840	837,000	58,590
Graphite, artificial	℔.	4,595,500	313,979	4,868,000	312,764
Lead, white	Sh. T.	122,398	12,068,443	123,640	15,234,297
Lead, sublime, white	Sh. T.	6,977	690,770	7,988	798,880
Lead, red	Sh. T.	16,269	1,919,767	13,693	1,874,448
Lead, orange mineral	Sh. T.	1,000	120,000	2,927	421,488
Litharge	Sh. T.	12,643	1,422,616	13,816	1,890,050
Mineral wool	Sh. T.	6,164	69,560	5,357	55,550
Zinc-oxide (*m*)	Sh. T.	72,603	5,772,240	77,800	6,257,361
Total enumerated	....	....	$139,400,225	....	$177,060,286

PRODUCTION OF METALS IN THE UNITED STATES.

Products.	Measures.	1905		1906	
		Quantity.	Value.	Quantity.	Value.
Aluminum	℔.	11,350,000	$3,632,000	14,350,000	$5,166,000
Antimony	℔.	5,912,000	614,848	5,856,000	1,272,509
Copper	℔.	875,141,741	137,404,200	917,620,000	180,000,339
Ferromanganese (*q*)	Lg. T.	289,983	17,639,666	300,500	24,040,000
Gold (fine)	Troy oz.	4,266,120	88,180,700	4,648,385	96,101,400
Iron (pig)	Lg. T.	22,702,397	377,540,862	25,006,691	453,871,441
Lead	Sh. T.	319,744	30,100,700	345,529	39,093,151
Nickel (*s*)	Sh. T.	6,500	5,005,000	7,150	6,360,640
Platinum	Troy oz.	318	5,320	(*e*)400	8,800
Quicksilver	Flasks (*o*)	30,705	1,217,652	28,293	1,157,184
Silver (fine)	Troy oz.	56,101,600	33,858,438	56,183,500	37,525,521
Zinc	Sh. T.	201,748	23,733,635	225,494	27,961,256
Total Metals	....	....	$718,933,021	....	$872,558,241
Total ores and minerals	....	....	803,602,383	....	819,262,646
Secondary products	....	....	139,400,225	....	177,060,286
Grand Total enumerated	....	....	$1,661,935,629	....	$1,868,881,173

Additional details will be found under the respective captions farther on in this volume. (*c*) Includes sulphate made from metallic copper. (*d*) Includes manganiferous iron ore. (*e*) Estimated. (*f*) One square covers 100 square feet. (*g*) Barrels of 265 ℔. (*h*) Barrels of 380 ℔. (*i*) Barrels of 42 gallons. (*k*) Includes salt used in manufacture of alkali; the barrel of salt weighs 280 ℔. (*m*) Includes a small quantity made from spelter. (*o*) Flasks of 75 ℔. (*q*) Includes spiegeleisen, although the value is given as for ferromanganese. (*s*) Includes nickel from Canadian ores smelted in the United States. (*t*) Barrels of 330 ℔. (*u*) Figures reported by the United States Geological Survey.

As in Vol. XIV, we have divided the large table, which previously followed this introduction, into three sections, one representing the production of ores and minerals in the United States; the second representing the production of metals of domestic origin; and the third representing the manufacture of chemical products, or other articles of commerce that are derived from the ores or minerals directly mined. It is conceived that

this arrangement is more logical, and more valuable than one in which all the substances are heterogeneously assembled. Values are reported merely as an indication of the relative magnitude of the various industries, from the commercial standpoint. As previously noted, comparatively little weight is to be placed upon value figures. For example, as a matter of form, all the copper produced in the United States is computed on the basis of the average price for Lake copper, in conformity with previously existing custom, although, as it appears in the detailed statistics, by far the major portion of the copper produced in the United States is sold as electrolytic copper, which fetches a slightly lower price than Lake; there are similar differences with regard to spelter, and other metals.

Summary of the Production.

Aluminum.—The production of aluminum in the United States in 1906 is estimated at 14,350,000 ℔., valued at $5,166,000, against 11,350,000 ℔., valued at $3,632,000, in 1905. These statistics are estimated upon the basis of known furnace capacity.

Alundum.—In 1906 there were 4,331,233 ℔. produced, valued at $303,186, against 3,612,000 ℔., valued at $252,840 in 1905. This abrasive substance is made only at Niagara Falls, N. Y., by the Norton Emery Wheel Company of Worcester, Mass. It is used in the manufacture of artificial corundum grinding wheels.

Ammonia and Ammonium Sulphate.—The ammonia production during 1906 is estimated at 75,000 short tons, as against 65,296 tons produced in 1905. This figure includes production of all forms of ammonia expressed in equivalent sulphate of ammonia. The production of ammonium sulphate in 1906 was 25,000 short tons, worth $1,555,750, against 19,332 tons, worth $1,216,279, in 1905.

Antimony.—In 1906 there were produced 295 short tons of antimony ore, valued at $44,250. This was the first production of domestic ore for a number of years. Of antimony metal, 2778 tons were produced in 1906, as compared with 2936 tons in 1905. These figures include the metal in imported ores, as well as that in hard lead, which latter represents by far the greater proportion of the total production.

Arsenic.—The production of arsenious acid in 1906 was 1,663,000 ℔., valued at $83,150, against 1,545,400 ℔., worth $50,225, in 1905. The Washoe smelter at Anaconda, Mont., and the Everett smelter on the Pacific Coast are the chief producers of this substance.

Asbestos.—In 1906 the output of asbestos was 1695 short tons, valued at $20,565, against 3100 tons, valued at $126,300, in 1905. The decrease in production was due to less activity in Arizona and in Virginia. The greatest part of domestic demand for asbestos is still supplied by imports from Canada.

Barytes.—The output of barytes in 1906 amounted to 63,486 short tons ($252,719), as compared with 53,252 tons ($196,041) in 1905. Missouri continues to be the largest individual producer.

Bauxite.—The output of bauxite in the United States in 1906 is estimated at 78,331 long tons ($352,490), as compared with 47,991 long tons ($203,960) in 1905. The productive areas of the United States are confined to Arkansas, Georgia and Alabama. The chief feature of 1906 was the greatly increased production in Arkansas.

Bismuth.—There were no shipments of domestic bismuth ore reported in 1906.

Borax.—The output of borax in 1905 was 46,334 short tons, worth $1,019,158, as compared with 45,647 tons, worth $698,810, in 1904. Practically all of the crude borax is produced in California, the statistics of which State for 1906 are not yet available.

Bromine.—In 1906 there were produced 1,229,000 ℔. of bromine ($184,350), against 899,434 ℔. ($139,492) in 1905. Michigan, which reported a decrease in 1905, regained its former place as chief producer in 1906. The other producing States are Ohio, Pennsylvania and West Virginia. A considerable part of the total bromine produced is recovered in the form of bromides.

Carborundum.—The output of carborundum in the United States in 1906 was 6,225,280 ℔. ($622,528), against 5,596,280 ℔. ($559,628) in 1905. The only producer of this substance is the Carborundum Company, of Niagara Falls, N. Y. The material is used both as an abrasive and as a refractory lining for furnaces.

Cement.—The production of cement of all kinds in the United States in 1906 amounted to 51,027,321 bbl., valued at $54,015,713. This output was divided as follows: Portland, 46,610,822 bbl. ($51,240,652); natural, 3,935,151 bbl. ($2,362,140); slag, 481,224 bbl. ($412,921). For comparison the 1905 production is given: Portland, 35,246,812 bbl. ($33,245,867); natural, 4,473,049 ($2,413,052); slag, 382,447 bbl. ($272,-614). The increase of the portland cement industry was enormous and will probably be as great in 1907. The increase in number of cement plants in the middle West and on the Pacific coast was a marked feature of the year.

Chromite.—In 1906 the production of chromite was 180 long tons, valued at $1800, against 40 tons ($600) in 1905. California is the only State producing chromite, the rest of the requirements of the country being supplied by imports from New Caledonia and Turkey. The material is used as a source of chromium salts and also as a refractory lining for metallurgical furnaces.

Coal.—The total production of coal in the United States for 1906 was 413,838,679 short tons, valued at $566,857,953, or an average of $1.37 per

ton. The corresponding output in 1905 was 387,076,136 short tons, valued at $533,331,749, or $1.38 per ton. In 1906 the anthracite production was 72,209,566 short tons ($166,307,002); bituminous, 341,629,-113 short tons ($400,550,951). In 1905 the total output was divided into anthracite, 78,731,523 short tons ($178,788,244), and bituminous 308,344,613 short tons ($354,543,505). All of the States producing anthracite coal reported a decrease in output in 1906 as compared with 1905, but these same States made marked advancement in their production of bituminous coal during 1906.

Coke.—The output of coke in 1906 was 32,690,362 short tons, as against 28,490,611 short tons in 1905. The total values of these outputs are, respectively, $86,887,392 and $72,299,694, giving an average of $2.66 per ton in 1906 and $2.54 per ton in 1905. The great increase in the importance of the coke industry is directly traceable to the prosperity existing in 1906 in the iron and steel trade, upon which the coke industry depends chiefly.

Copper.—The production of copper of domestic origin in 1906 was 917,620,000 lb. as against 875,241 741 lb. in 1905. Colorado, Montana and Utah showed small decreases and a considerable reduction was recorded in Wyoming. Montana is still the largest producer, but if the rate of increase in the Arizona output continues the latter State may soon attain first rank.

Copperas.—In 1906 the output of copperas in the United States was 22,839 short tons, worth $228,390 as compared with 21,103 short tons ($147,721) in 1905. The greatest part of the copperas produced is obtained as a by-product of iron and steel sheet and wire plants. In 1906 the United States Steel Corporation furnished upward of 90 per cent. of all the copperas made in this country.

Cryolite.—There is no production of cryolite in the United States, the entire consumption being supplied by imports from Greenland. The imports of 1906 amounted to 1505 long tons, worth $29,683 as against 1600 long tons, worth $22,482 in 1905.

Crushed Steel.—In 1906 the production of crushed steel was 837,000 lb., ($58,590), against 812,000 lb. ($56,840) in 1905. Crushed steel is used as an abrasive. It is produced by only one concern in the United States.

Emery.—The production of emery in the United States in 1906 was 2147 short tons, valued at $22,780, as compared with 2315 tons ($19,678) in 1905. Practically all of the emery is produced in the region around Peekskill, N. Y., with a smaller amount from Chester, Mass.

Feldspar.—The United States in 1906 produced 72,656 short tons of feldspar worth $401,531 against 35,419 tons ($226,157) in 1905. Penn-

sylvania, Maryland, Connecticut, Maine and New York are the chief producing States, ranking in the order named.

Ferromanganese and Spiegeleisen.—The output of ferromanganese in 1906 was 55,520 long tons against 62,186 in 1905. The production of spiegeleisen was 244,980 long tons in 1906 against 227,797 in 1905. The total values of these products for 1906 and 1905 respectively, are $24,040,000 and $17,639,666.

Fluorspar.—The output of fluorspar in 1906 was 34,683 short tons, valued at $201,481, which is to be compared with a production of 39,600 tons, valued at $232,452, in 1905. The decrease in production was largely due to an accumulation of stocks in the previous year among the Illinois producers. The actual amount marketed in 1906 was in excess of that sold in 1905.

Fuller's Earth.—The 1906 output of fuller's earth was 28,000 short tons, valued at $237,950 against 25,745 short tons, worth $186,816 in 1905. The chief use of fuller's earth is in clarifying oils and fats. The chief producing districts are in the Southern States.

Garnet.—The production of garnet in 1906 amounted to 5404 short tons with a value of $179,548 as compared with 3694 tons, valued at $114,625 in 1905. By far the greatest part of this production comes from New York. The increasing use of artificial abrasives apparently does not greatly restrict the production of garnet.

Gold.—In 1906 the output of the United States was 4,648,385 oz. fine worth $96,101,400, as against 4,266,120 ($88,180,700) oz. fine in 1905. Of the increase, Alaska is to be credited with over 80 per cent., the remarkable rate of increasing production in this territory having been well sustained in 1906. Nevada also made a heavy gain, coming near to doubling its 1905 output. Other States made slightly decreased outputs or increased but little.

Graphite.—The production of crystalline graphite in the United States during 1906 amounted to 4,894,483 ℔., worth $170,866, as against 4,260,656 ℔. ($170,426) in 1905 .New York continues to be the largest producer of crystalline graphite. The output of artificial graphite in 1906 was 4,868,000 ℔. worth $312,764 against 4,595,500 ℔. ($313,979) in 1905. The demand for artificial graphite is growing rapidly, and it will be increased still more in the future as a result of the successful production of soft artificial graphite, which comes into direct competition with natural graphite as a lubricant, and other uses for which ordinary artificial graphite can find no application.

Gypsum.—The output of gypsum in the United States in 1906 was 1,540,585 short tons, valued at $3,837,975, against 1,043,202 tons ($821,967) in 1905. The United States Gypsum Company is the largest operator, controlling 35 plants, which are distributed through 12 States.

Iron.—The production of pig iron in 1906 amounted to 25,006,691 long tons valued at $453,871,441, against 22,702,397 tons valued at $337,540,862 in 1905. The amount of bessemer pig remained about the same in 1906 as in 1905 but foundry and forge pig decreased. Basic pig production increased considerably, showing the growing importance of the basic open-hearth process.

Iron Ore.—In 1906 the United States produced 49,237,129 long tons of iron ore, valued at $107,091,574. The three producing districts contributed their shares as follows: Lake Superior, 38,522,129 tons; Southern States, 7,450,000 tons; other States, chiefly New York and Central States, 3,265,000 tons. The total output in 1905 was 44,578,456 long tons, having a value of $94,768,122.

Lead.—The production of lead of domestic origin in the United States in 1906 amounted to 345,529 ($39,093,151) short tons. The production in 1905 was 319,744 ($30,100,700) tons.

Lead Products.—The domestic output of lead products during 1906 was as follows: White lead, 123,640 short tons ($15,234,297) against 122,398 tons ($12,068,443) in 1905; red lead, 13,693 short tons ($1,874,448) against 16,269 tons ($1,919,767) in 1905; litharge, 13,816 short tons ($1,890,050 against 12,643 tons ($1,422,616) in 1905, orange mineral, 2927 short tons ($421,488) against 1000 tons ($120,000) in 1905; sublimed white lead, 7988 short tons ($798,880) against 6977 tons ($697,700) in 1905.

Limestone.—The consumption of limestone in 1906 for metallurgical purposes was 15,486,139 short tons, having a value of $7,339,125, which is to be compared with a consumption in 1905 amounting to 14,098,000 short tons, valued at $6,739,200. Iron smelting demanded the largest percentage of the above amounts.

Lithia.—The 1906 output of lithia minerals is not yet available. Twenty-one tons, worth $252, were produced in 1905, as compared with 574 tons worth $5155, in 1904. The production increased largely in 1907 under the mistaken impression that there was a large demand for lithia, and the retrenchment necessary from this over-production is marked by the lessened activity in 1905, which probably extended into 1906.

Mica.—In 1906 the output of sheet mica amounted to 1,423,100 lb. ($252,248), against 851,000 lb. ($185,900) produced in 1905. The output of scrap mica in 1906 was 1489 short tons valued at $22,742 as compared with 856 tons produced in 1905 and worth $15,265. The domestic production of mica is but a small part of the consumption and considerable quantities are imported from Canada and India.

Mineral Wool.—The production of mineral wool in 1906 was 5357 short tons, worth $55,550, as compared with 6164 tons, worth $69,560,

in 1905. The chief producers are the U. S. Mineral Wool Company of New York and the Columbia Mineral Wool Company of Chicago.

Monazite.—The output of monazite in 1906 was 846,175 lb., valued at $152,312, against 1,352,418 lb., valued at $163,908, in 1905. The chief production continues to come from the Carolinas.

Nickel.—The output of metallic nickel, almost entirely obtained from Canada, was 7150 short tons, valued at $6,360,640, in 1906, against 6500 tons valued at $5,005,000, in 1905.

Petroleum.—The production of petroleum in 1906 was 131,771,505 bbl. (of 42 gal.), valued at $80,277,279, while the 1905 production was 139,889,210 bbl., valued at $118,905,828. A decline in production was recorded in the Appalachian field and a rapid decline in the Texas and Louisiana fields, but the Mid-continental field showed a remarkable increase in production over 1905.

Phosphate Rock.—The 1906 output of phosphate rock was 2,052,742 long tons, valued at $12,342,741, as against 1,933,286 tons, valued at $9,713,296, in 1905.

Platinum.—The figures for the 1906 production are not yet available, but the estimated output is 400 oz. troy. In 1905 the production was 318 oz. troy. The increase results from awakened attention to the black sand concentrates from Pacific coast placer workings. Imports from Russia supply the major part of the domestic demand.

Potassium Salts.—Domestic consumption is supplied entirely by imports from Germany, which in 1906 amounted to 321,220,320 lb. valued at $6,139,597, as against 307,054,130 lb., valued at $5,522,155, in 1905.

Pyrite.—The production of pyrite was 225,045 long tons in 1906 and 224,980 tons in 1905. These amounts are valued, respectively, at $767,-866 and $752,936. The chief producing State is Virginia, but the production in Alabama is becoming of increasing importance.

Quicksilver.—The 1906 output of quicksilver was 28,293 flasks of 75 lb., worth $1,157,184, as compared with 30,705 flasks, worth $1,217,652, in 1905. These values are calculated on the basis of the average value of quicksilver at New York for the years in question. California is still the chief source of this metal, but small amounts are produced in Texas and Utah.

Salt.—In 1906 there were produced 28,172,380 bbl. of salt of 280 lb. each, valued at $6,658,350. The output in 1905 was 25,966,112 bbl., valued at $6,095,922. Michigan and New York together produce over half of the total output, and in 1905 Michigan outranked New York for the first time.

Silver.—The production of silver in 1906 was 56,183,500 oz. fine, worth $37,525,521, while in 1905 the production was 56,101,600 oz. fine, valued at $33,858,438.

Sulphur.—The production of sulphur during 1906 is estimated at 285,000 long tons, valued at $6,056,250, against 215,000 tons, valued at $4,742,900, in 1905. Louisiana is the chief source of the domestic sulphur in the United States, the small amount which is produced in the West being consumed locally

Talc and Soapstone.—The production of fibrous talc in 1906 amounted to 64,200 short tons, valued at $541,600, against 67,000 tons ($469,000), in 1905. New York is the principal source of the fibrous talc produced in this country.

Tungsten Ore.—The production of tungsten ore in 1906 was 1096 short tons, valued at $442,748, as compared with 834 tons ($257,493), in 1905. Boulder county, Colo., supplies over 85 per cent. of the domestic production in the United States.

Zinc.—The output of virgin spelter in 1906 was 225,494 short tons, against 201,748 tons in 1905. During 1906 there was a large increase of smelting capacity in the United States, due to enlargement of existing works. Only one new plant went into operation, although several are in course of construction.

Zinc Ore.—The output of zinc ore amounted to 905,175 short tons in 1906, valued at $17,250,420, as compared with 795,698 tons, worth $15,596,457, in 1905.

Zinc Oxide.—In 1906 the production of zinc oxide, including zinc-lead pigments, was 77,800 short tons, valued at $6,257,361, as compared with 72,603 tons, valued at $5,772,240, in 1905. There is a small production of zinc oxide by the combustion of spelter, but the great bulk of the product is made directly from ore.

ALUMINUM.

The only producer of aluminum in the United States in 1906 was the Pittsburgh Reduction Company, which on January 1, 1907, changed its name to Aluminum Company of America. The organization, stockholders and board of directors in the company remain exactly the same, and the change was made only that the name might better indicate the nature of the business of the company. As this corporation maintains its policy of refusing to publish its statistics of production, the production of aluminum in the United States can only be approximated. Our estimate of the production for 1906 is 14,350,000 ℔. against 11,350,000 ℔. in 1905. The estimated production of the Canadian branch of this company for 1906 is 6,000,000 ℔. against 3,100,000 ℔. in 1905. These estimates are based on power known to be used and an average output figure per unit of power.

PRODUCTION, IMPORTS AND CONSUMPTION OF ALUMINUM IN THE UNITED STATES.

Year.	Production.			Imports.			Exports.	Consumption
				Crude.		Mfrs.		
	Pounds.	Value.	Per lb.	Pounds.	Value.	Value.	Value.	Value.
1897	4,000,000	$1,400,000	$0.35	1,822	$1,082	$3,647	(a)	$1,404,729
1898	5,200,000	1,690,000	0.33	60	30	13,840	$238,997	1,474,268
1899	6,500,000	2,112,500	0.33	53,622	9,425	7,828	291,515	1,838,238
1900	7,150,000	2,288,000	0.32	256,559	44,455	5,989	281,821	2,056,623
1901	7,150,000	2,238,000	0.31	564,803	104,168	5,580	183,579	2,164,169
1902	7,300,000	2,284,590	0.31	745,217	215,032	3,819	116,052	2,387,389
1903	7,500,000	2,325,000	0.31	498,655	139,298	4,273	157,187	2,311,384
1904	7,700,000	2,233,000	0.29	515,416	128,350	478	166,876	2,494,952
1905	11,350,000	3,632,000	0.32	530,429	106,108	33	290,777	3,015,364
1906	14,350,000	5,166,000	0.36	770,713	154,292	1,866	364,251	4,957,907

(a) Not reported.

Market Conditions.

Aluminum is now a strong competitor with copper and other metals, and doubtless the increase in the price of copper has been instrumental, on this account, in the rise in price of aluminum which occurred during 1906. However, the fact that the demand is far in excess of production is the chief reason for the present high price of aluminum. The amount of aluminum consumed is limited only by the number of furnaces which are now in operation, and by the capacity of the dynamos which operate the furnaces.

The increase in production of 1906 over that of 1905 was due to the partial completion of new equipment and plant whose construction was undertaken during 1905, and the increase in production in 1907 and 1908 will be very marked for the same reason. By the end of 1908, indeed, the production of aluminum in the United States, if present plans are carried out, will make a significant comparison with the production of copper, taking into consideration the relative bulks of the two metals.

PRICE OF ALUMINUM AT NEW YORK.

Grade.	Dec., 1905.	July, 1906.	Dec., 1906.
99% pure	35	36	38
90% pure	33	34	37
No. 12 casting alloy	35	36	37
No. 21 casting alloy	33	34	35¼
No. 31 casting alloy	30	31	33¼

The above prices were for ton lots or over; the prices for small lots were 2 to 3c. per lb. higher.

The Aluminum Industry in Europe.

The European producers of aluminum are only four in number. Their production for 1906 is estimated as follows:

	Metric tons	Pounds.
British Aluminium Works, Scotland	2,500	5,500,000
Société Electro-Metallurgique Française	2,500	5,500,000
Aluminium Industrie Aktien Gesellschaft	4,000	8,800,000
Société des Produits Chimiques d'Alais	2,000	4,400,000
Total	11,000	24,200,000

WORLD'S PRODUCTION OF ALUMINUM.
(In metric tons.)

Year.	Great Britain.	France.	Switzerland.	North America.	Totals.
1897	(a)310	470	800	1,815	3,195
1898	310	565	810	2,359	4,034
1899	559	763	1,300	2,949	5,571
1900	569	1,026	2,500	3,244	7,339
1901	560	1,200	2,500	3,244	7,504
1902	600	1,355	2,500	3,312	7,767
1903	(b)650	1,570	(b)2,500	3,403	8,123
1904	(b)650	1,650	(b)3,000	3,494	8,794
1905	2 250	4,425	3,675	6,560	16,810
1906	2,500	4,500	4,000	7,325	18,325

(a) C. L. Neve Foster, British Mineral Statistics for 1897. (b) Statistics of Metallgesellschaft, Frankfurt am Main.

Bibliography.

Aluminum is treated in every volume of The Mineral Industry, and each article contains detailed discussions of the sources of the raw material for aluminum production, the present state of the art of reduction, improvements in the technology of aluminum, and enumerations of the various applications of aluminum in the arts. The features of special

interest contained in these articles are as follows: In Vol. I, the history of the early investigations which resulted in the discovery of aluminum and its reduction from its compounds. In Vol. VI, an account of various alloys of aluminum and a very thorough review of all the uses and applications of aluminum to various manufacturing industries. In Vol. XII, a description of a new aluminum process, patented by G. Gin of Paris, the first mention of the thermit process of welding, and a résumé of progress in the manufacture of aluminum alloys. In Vol. XIII, a brief discussion of the use of aluminum for electric power transmission and methods for the quick technical analysis of commercial aluminum. In Vol. XIV, a review of the aluminum industry in Europe, the progress in the metallurgy of aluminum during 1905, giving the present American practice of aluminum reduction, with a description of the Betts aluminum process, a review of the various soldering compounds on the market, and a description of the condition of the art of aluminothermics, giving details of welding by the use of thermit.

Below are some of the recent articles which have appeared in the technical press and which deal with the more important phases of the aluminum industry, together with a list of recent patents.

"Recherches récentes sur la fabrication de l'Aluminum." (*L'Echo des Mines*, Jan. 10, 1907). A brief review of all recent experiments which aimed at new means of producing aluminum.

"Laterite in Mysore." E. W. Wetherell. (Mysore Geol. Dept., Memoirs Vol. III, Part I; 27 pp.) Describes the origin of this aluminum ore and discusses its occurrence in India.

"The Use of Chills in Making Aluminum Castings." Amos Brown. (*Brass World*, Dec., 1906; 2½ pp.) Describes the use of the particular methods for accomplishing the difficult casting of aluminum.

"Casting Aluminum in Metal Molds." (*Brass World*, Aug., 1906.) Shows the result of certain massive castings in metal, but as a rule, sand is preferred for this class of work.

"Alloys of Aluminum and Copper." (*Engineering*, Jan. 25, and Feb. 1, 1907; 6 pp.) An abstract from the 8th Report of the Alloys Research Committee dealing with various methods of preparing copper-aluminum alloys, and giving the results of many different tests of the physical and chemical properties of the alloys.

"Melting Aluminum and Its Alloys." W. J. May. (*Mechan. World*, Oct. 5, 1906.) Mentions the difficulties met with in melting aluminum and its alloys, and states how they may be overcome.

"Aluminum-Bismuth and Aluminum Tin-Alloys." A. G. C. Gwyer. (*Zeit. anorg. Chemie*, Vol. 49, 1906; pp. 311–319). Discussion of the physi-

cal properties of these alloys, showing how tin influences the solubility of bismuth in aluminum.

"The 'Schoop' Process of Aluminum Soldering." (*Electric Accumulator*, Jan., 1907; 1 p.) Discusses this new method of aluminum soldering, which is based upon the fact that aluminum becomes soft at certain temperatures, in which condition it may be joined by hammering to another piece of aluminum heated to the same temperature. Tests upon joints welded in this fashion are tabulated with the article.

"Ueber Aluminiumzellen." M. Buttner. (*Zeit. f. Elektrochem.*, Nov. 2, 1906; 10 pp.) A study on the use of aluminum for storage batteries.

"Aluminio en Vez de Cobre en las Instalaciones Electricas." C. T. de Tolentino. (*Revista Minera*, Jan. 16, 1907; 1 p.) A short article, comparing the relative advantages of copper and aluminum as conductors of electricity, and giving a summary of the saving to be made by using aluminum on a line 30,000 m. long.

"The Iodimetric Determination of Basic Alumina and of Free Acid in Aluminum Sulphate and Alums." S. E. Moody. (*Am. Journ. of Sci.*, Dec., 1906; 4½ pp.) Discusses a method of analysis for alumina in the sulphates.

Reduction Process. E. Maemecke, Berlin, Germany. In the Goldschmidt aluminothermic process for the reduction of metals from their oxides, the substitution of a compound of silicon and magnesium as the reducing agent instead of aluminum. Brit. pat. 3089 of 1906.

Aluminothermic Process. K. A. Kuhne, Dresden, Germany. Improvements in the aluminothermic process for reducing metals from their oxides, consisting in the addition of chlorate of potash to the mixture of aluminum and oxide. Brit. pat. 20,884 of 1905.

Aluminothermic Process. H. Goldschmidt, Essen, Germany. Improvements in the inventor's aluminothermic process for producing heat and reducing iron and other metals by the reaction of aluminum on iron and other oxides by the addition of calcium to the aluminum. Brit. pat. 926 of 1906.

Aluminum Alloy. A. Chambaud, Paris. An aluminum alloy which is more readily workable than pure metal, consisting of 99.02 Al. 0.31 Fe, 0.01 Zn, 0.04 Mg. and 0.61 Cu. Brit. pat. 9750 of 1906.

Aluminum Solder. M. Gruber, Berlin, Germany. A solder for aluminum, consisting of 10 parts copper, 25 parts zinc, 60 parts tin, 3 parts cadmium and 2 parts aluminum. Brit. pat. 12,599 of 1905.

Aluminum Bronze. C. Claessen, Berlin, Germany. Improved method of toughening aluminum bronzes by heating and chilling. Brit. pat. 19,282 of 1905.

The Metallurgy of Aluminium in 1906.[1]

By Joseph W. Richards.

The year 1906 witnessed great activity in all details of the aluminium industry. Four firms in Europe and one in America practically control the industry, being owners of a large part of the beds of bauxite, and also of the best water powers conveniently accessible. The unusually high prices of copper, tin, nickel and zinc, as well as the other metals, during 1906, stimulated the demand for aluminium to such an extent that much more could have been sold than the companies were able to produce; this condition held strongly all the year, and was, in fact, most marked at the end of the year.

Plants.

United States.—The Pittsburgh Reduction Company (now the Aluminum Company of America) was the only producer. It owns large bauxite deposits in Georgia, Alabama and Arkansas, obtaining, however, most of its ore from Saline county in the latter State. It has done a large amount of stripping and development work at its mines at Bauxite, Arkansas, also is just completing there a new crushing, grinding and drying plant, and has built a railroad called the Bauxite & Northern, connecting all of its mines in that region with the Chicago, Rock Island & Pacific Railroad and the Missouri Pacific and Iron Mountain systems. This company has realized that reserve stores of bauxite are as essential to the welfare of a large aluminium company as reserves of iron ore are to a great steel corporation, and it has accordingly spent large sums of money in purchasing bauxite lands both in the eastern district—Georgia and Alabama—and in Arkansas.

Concerning the production of pure alumina, this company has enlarged to great dimensions its chemical plant at East St. Louis. The process used is the same as heretofore, but the capacity of the plant has been increased several times. The power for the plant is furnished by condensing turbine engines; the evaporating plant is the most complete and largest of its kind ever built.

The carbons used in the reduction are now manufactured entirely by this company, being baked in electrically-heated furnaces patented by Chas. M. Hall. The old plant for making carbons, at the upper Niagara works, has been practically torn down and rebuilt to three times its previous capacity; the buildings are of steel, with travelling cranes and every up-to-date conveying appliance; the new plant has an equipment and capacity equal to that of any other carbon electrode plant in the world.

The rolling and sheet mills of the company have been correspondingly

[1] In deference to the wishes of Prof. Richards, his preferred spelling "aluminium" has been retained in his article although the regular style of *The Mineral Industry* is "aluminum."—Editor.

enlarged; a new mill for this purpose is in course of construction at Niagara Falls. This is of re-inforced concrete, and when finished will be one of the largest and most complete sheet rolling mills in America.

The work thus done by this company within the last three years, in plants and processes entirely outside of the reduction of the metal, has been on a scale which, remembering the former infancy of the aluminium industry, may be properly characterized as "stupendous." The investments thus made in these accessory enterprises have amounted to several millions of dollars.

The reduction plants of this company, at Niagara Falls, Massena, N. Y., and Shawinegan Falls, Quebec, Canada, are all in process of being greatly enlarged. At Niagara Falls, the lower plant, using power supplied from the canal of the Niagara Falls Power and Manufacturing Company, which has heretofore been reported as of 12,000 h.p. capacity, has been increased by the building of a very large plant to use 45,000 h.p., consisting of five units of 9000 h.p. each. Two of these units (18,000 h.p.) will be in operation by May, 1907, and the whole plant in June, 1907. At Shawinegan Falls, the company is quadrupling its already large capacity and expects the new plant to be finished ready for operation in April, 1907. This plant is nominally controlled by the Northern Aluminium Company, which is a subsidiary company of the American company, and manufactures aluminium chiefly for export; it is expected that this increased capacity will exceed the demands for export, but the company is intent upon providing reserve facilities equal to all possible demands of the near future. At Massena, the company has purchased the entire plant of the St. Lawrence River Power Company, with its canal and power house of 40,000 h.p. capacity, and is actively preparing to dredge out the canal to double this capacity. One of the largest of modern dipper dredges and the most powerful elevator dredge ever built have been installed ready to commence operations as soon as the winter is over. The complete dredging plant has cost over a million dollars. A new power house for this enlarged capacity will be started in 1907, and eight large water wheels, to absorb the capacity of the first canal, have been purchased and will be placed in position in the old power house within a few months. The output of this plant in 1907 will be from two to three times the output of 1906.

The difficulty of hurrying hydraulic construction work, and the very great difficulty in getting materials moved and work done on contract time, has delayed the plans of this company, as it did that of business in general in 1906, and the construction and installation work which it was expected to have completed by the end of 1906 will be completed between that date and June 1, 1907. At the latter date it is confidently expected that the output of American aluminium will be several times as great as the average output in 1906. The company has been as sensible of the shortage

of aluminium in the market during 1906 as has its customers or would-be customers, and hopes by these liberal expenditures of several millions of dollars to keep ahead of all demands in the future.

Great Britain.—The British Aluminium Company is undertaking large extensions of its plant in Scotland.[1] It has commenced to construct a dam near Kinloch-levan, which will form a reservoir with a capacity of 20,000 million gallons, supplied by a water-shed of 55 square miles, which has the largest rainfall in Great Britain. This reservoir will be 7.5 miles long by 0.5 mile wide, obliterates three lakes, is 1000 ft. above sea-level and only five miles from the coast. A concrete conduit will convey water 3.5 miles, where it will connect with a short pipe line delivering the water to turbines at about 900 ft. head. The dam is of concrete, 0.5 mile long, 80 ft. high in the center, and will cost $2,000,000; the rest of the water power scheme will cost $500,000. The horse-power expected from this expenditure is 20,000, but it should be at least 50,000, to correspond with the cost of the water works. The Foyers works will be enlarged from 6000 to 15,000 h.p., by the hydraulic works there nearing completion. The same company has acquired water power at Sarpsfos, Norway, but how much has not been stated. A plant is also contemplated by it in Wales. U. S. Consul Guenther at Frankfort estimates the output of this company in 1906 as 2250 metric tons.

Germany.—The Aluminium Industrie Aktien Gesellschaft, operated mostly by German capital, has works at Rheinfelden, Baden (near Basel), at Neuhausen, Switzerland (at the Rhine Falls), and at Lend Gastein in Upper Austria. This company is financially very successful, having paid 18 per cent. dividends in 1904 and 22 per cent. in 1905, on a capital of $3,000,000; in 1906 it floated $5,000,000 additional capital stock, which was taken by a syndicate of Berlin bankers at 150 per cent. premium. It employs 88 officials and 661 workmen, and produced 3675 tons in 1905.[2] It operated however, 24,000 h.p. in 1906,[3] and must have produced with this about 5500 metric tons. Active enlargements of this company's plants are in progress, such that it expects to be operating 100,000 h.p. toward the close of 1907, and 120,000 h.p. in 1908. This would give its probable production for 1907 as somewhere about 13,000 tons, and its capacity in 1908 as 27,000 tons. This company also runs an alumina plant at Deutsch Lissa, Silesia, where 200 workmen are employed.

France.—The Société Electrométallurgique Française, at Froges, La Praz and Les Sordrettes (St. Michel) has at its command 20,000 h.p.[4], but is said to be producing only 2300 tons annually, which would correspond to about half that power. Enlargements of 12,000 h.p. at Vénéon

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[1]*Electrochem. and Met. Ind.*, Sept., 1906. From the London *Electrician.*
[2]U. S. Consular Report. Consul Guenther, Frankfort.
[3]*Metal Industry*, January, 1907.
[4]R. Pitaval *Journal de l'Electrolyse*, Oct. 15 and Nov. 1, 1906.

(Isère) and 10,000 h.p. at Bissorte (Savoy) are said to be contemplated.

The Société des Produits Chimiques d'Alais et Carmague has 11,000 h.p. at Calypso and 2500 at St. Félix, and is said to be contemplating 12,000 additional at St. Jean (Savoy). Consul Guenther reports its output as 2000 tons, which would correspond fairly well with its reported available power.

A new plant of 30,000 h.p. is said to be proposed for Chippis, in the Valais, French Switzerland. Another plant is spoken of at Aube, in the Pyréneés, of 25,000 h.p.; and one in the southwest of France of 10,000, but they are at present only propositions.

Italy.—For some time it has been known that bauxite exists in Italy, though probably not of high grade. A company has been recently incorporated, called the Aluminium Society, which expects to start operations early in 1907 in the Pescara Valley, where 30,000 h.p. is available, and 14,000 h.p. is already installed for various electro-chemical purposes. The aluminium plant is reported to cover 40,000 sq.m., and to be connected by cableway with its bauxite mines at Lecce di Marsi. The details so far made public are hazy and somewhat confused.

Canada.—It is rumored[1] that a $10,000,000 company is being organized in Canada, to utilize Canadian water powers for manufacturing aluminium.

India.—Recent discoveries in the central provinces of bauxite carrying 51 to 68 per cent. of alumina, and of deposits in Bengal and near Bombay carrying 43 to 65 per cent., have revived hopes that eventually an aluminium industry to supply the local market might come into existence in India. Alfred Chatterton, who has done so much in the Industrial School at Madras to teach the native metal workers to manufacture aluminium ware, writes of the present situation, in the Madras *Mail*, as follows:[2] "At the present time India absorbs less than 100 tons of aluminium yearly; it could be easily increased to 1000 tons. The Indian army uses it for canteens, water bottles, and cooking pots. Most Europeans in India use aluminium cooking utensils. The natives use so much acetic, tartaric and citric acids in their cooking, in combination with salt, that their food rapidly corrodes aluminium. Since 1898, a good market has been started, and if works were established, with the cheap bauxite, power and labor, aluminium could be produced very cheaply, to supply an established and easily enlarged market."

General Remarks.—While but few countries are now producing aluminium, yet the prospects are very likely that countries with very abundant and cheap water powers, like Norway, or with cheap labor and water powers, like India and Japan, will not long stay out of the ranks of the pro-

[1] *Metal Industry*, December, 1906.
[2] *Ibid*, October, 1906.

ducers. The total production in the world, in 1906, was probably not far short of 19,000 metric tons, valued at $12,500,000, but 1907 will probably double that output, and 1908 double that of 1907. These figures are admittedly guesses; they are given because they are thought to be intelligent ones.

Aluminium Salts.

Carbonate.—The existence of true aluminium carbonate has been doubted by some chemists. A. Gawalowski[1] has studied the question and has prepared a carbonate at ordinary atmospheric pressure which is insoluble in water and contains 8 to 9 per cent. of carbon dioxide and 40 to 45 per cent. of alumina. Prepared at eight atmospheres pressure, a polycarbonate is obtained, completely soluble to a clear solution in water. On reducing the pressure carbon dioxide escapes and at atmospheric pressure the carbonate first mentioned precipitates. The filtrate from this still contains some carbonate in solution; on warming it to 25 or 30 deg. C. or keeping it in the air, it deposits a carbonate containing 2 to 3 per cent. of carbon dioxide and 50 per cent. of alumina. The bearing of these facts on the possible *modus operandi* of the hot-spring deposition of bauxite is at once evident. At a depth of 320 ft. a hydrostatic pressure of eight atmospheres would exist, under which alumina would dissolve completely in a carbonated water, and could thus be leached out of aluminous rocks and subsequently deposited at the surface practically as bauxite, where the pressure is relieved.

Processes of Reduction.

Hall Process.—Francis R. Pyne,[2] working in the metallurgical laboratory of Lehigh University, has investigated the melting points of mixtures consisting of cryolite and alumina, thus reproducing one of the mixtures covered by the Hall patents (now expired). In commercial practice, more aluminium fluoride than is contained in cryolite is used in the composition of the bath. Mr. Pyne's results were as follows:

MELTING POINTS OF CRYOLITE—ALUMINA MIXTURES.

Cryolite.	Alumina.	Melting Point	Cryolite.	Alumina.	Melting Point
100	....	1,000°C.	93	7	982
97	3	974	92	8	992
96	4	960	90	10	980
95	5	915	85	15	994
94	6	960	80	20	1,015

The minimum melting point is with 5 per cent. alumina, at 915 deg.; another minimum occurs with 10 per cent. alumina, at 980 deg.; between these is a maximum, about 992 deg. at 8 per cent. alumina, corresponding

[1]*Chemisches Central Blatt*, 1906, i, 640.
[2]*Trans.* Am. Electrochem. Soc., Vol. X; *Electrochem. and Met. Ind.*, November, 1906, p. 435.

fairly well with the proportions of the possible compound 5 (AlF_3. 3NaF). $Al_2 O_3$. The bath with 5 per cent. alumina is indicated as probably the best for electrolytic reduction.

The original Hall patents, covering the electrolysis of a melted bath consisting of double fluoride of aluminium and a more positive metal, as the solvent, and alumina dissolved therein, expired April 2, 1906, and as such are now the property of the public. The working of these, or of any melted electrolyte for obtaining aluminium, without the use of external heat, operating solely by the internally-generated electrical heat of the decomposing current, is still covered by the Bradley patents, according to the decision of the courts, and these latter do not expire until February, 1909. The public is therefore free to use the Hall process, as formerly covered by the Hall patents, but in doing so must not infringe on the manner of working covered by the Bradley patents.

Gin Process.—Gustav Gin[1] seeks to revive the old Deville-Castner sodium-reduction process in a modified form. He would first make aluminium chloride by mixing bauxite with 25 per cent. of its weight of anthracite dust and 10 per cent. of tar, compressing into balls, heating in a retort to redness and passing chlorine, thereby volatilizing aluminium chloride, contaminated by ferric chloride. The latter is removed by pouring liquid zinc into the sublimation chamber, which is then closed and slightly heated, resulting in reducing the iron chloride to the metallic state. The sublimation chamber is then heated to 500 deg. C., and the aluminium chloride vapor allowed to escape through a heated tube where it comes in contact with melted zinc-sodium alloy, resulting in the reaction:

$$4\ Al\ Cl_3 + 3\ Zn\ Na_2 = 4\ Al + 3\ (Zn\ Cl_2.\ 2NaCl)$$

Both zinc and sodium are said to act as reducing agents under these conditions, giving aluminium containing about 1 per cent. of zinc. This is purified by casting it into the channel of an electric furnace on a refractory hearth, the end terminals being large blocks of aluminium. A channel 16 sq. cm. in cross section by 25 m. total length containing 100 kg. of aluminium needs 10,000 amperes at about 25 volts tension, and is heated to whiteness, so that the zinc volatilizes completely in a few minutes. The zinc-sodium alloy is produced electrolytically from the slag produced in the reduction, by adding to it NaCl and KCl and electrolysing it until all the zinc in it and most of the sodium is deposited.

My chief comment on the above is that a person not compelled to use this process could take the same current which produces the $Zn\ Na_2$ alloy and with it prepare the same 4 Al. *direct* from a Hall bath, without any of the costly operations of making aluminium chloride, reducing it or refining the aluminium from zinc.

[1]*Revue Industrielle*, July 14, 1906.

Casting.

Pin Holes.—E. S. Sperry[1] gives some valuable observations on this subject. These annoying defects are usually just under the surface of a casting, and are in evidence as soon as the casting is machined. They are due to the metal absorbing gases while in the furnace, and giving them out as it sets. No harm is done by the material of a graphite crucible, which may be used with safety, but to get absence of gas in the metal the top of the crucible should project out of the fire and above the top of the furnace, like a zinc-melting pot. The metal absorbs gases in the furnace rapidly if exposed to them, and much faster as the temperature becomes high. Overheating must in all cases be avoided. Mr. Sperry formulates the following general rules:

(1) Use only ingot metal, and put the largest pieces in the bottom of the crucible. (2) Do not allow the ingots or any scrap metal to project above the top of the crucible. (3) Do not fill up with coke around the crucible up to the top. (4) Take the crucible from the fire before the metal gets too hot. (5) The metal must never be heated above the correct temperature for pouring. (6) Have the molds ready before the metal is put into the crucible, so that any possible delays are avoided after the metal is melted. (7) Do not pour too cold, since air may be entangled in the pasty metal and form large blow-holes. Wet sand may also cause similar blowholes.

For casting plates for rolling, Mr. Sperry recommends using a mold having a side runner connected with the mold proper throughout its hight by a $\frac{3}{16}$-in. slot. Dross remains in the runner, and only clean metal enters the mold. Runners and gates are best cut off by band saws.

Amos Brown[2] recommends using chills at such parts of an aluminium casting as are ordinarily provided with risers. The latter are often a detriment to a fine casting, causing shrinkage cracks and being difficult to cut off. Pieces of sheet copper or brass, rather thick and flat, about $\frac{1}{16}$ in. thick, may be used as effective substitutes, being placed so as to cover the part of the casting which it is intended to chill into shape. They thus can be made to cause heavy portions of the casting to cool off as quickly as the lighter parts which are not chilled, and thus preserve the shape of the heavier parts, avoid unequal cooling and shrinkage strains. The chills should be cleaned from scale or grease by emery cloth. The proper heat for casting the aluminium is a very low red.

Plating.

A. Giroux claims to have obtained satisfactory results, principally by giving great attention to the preliminary preparation. The articles are first cleaned in potassium cyanide solution until the greasy spots are loos-

[1] *Brass World*, January, 1906.
[2] *Brass World*, December, 1906.

ened; then cleaned by brushing with powdered pumice until a clean surface is obtained. They are then to be well washed and dipped into caustic potash solution until gas bubbles are evolved, and are then immersed in cold water. They are then dipped into a solution containing 0.2 per cent. of corrosive sublimate and 2 per cent. of potassium cyanide. They are then dipped once into the caustic potash solution, and placed at once in the plating bath. The plating must be done slowly and evenly, not at any time showing evolution of gas; only in this way can an adherent plating be obtained.

Soldering.

A. W. King in U. S. pat. 811,725, Feb. 6, 1906, coats the surfaces of aluminium to be soldered with an alloy containing 3 tin, 7 lead, 5 zinc, and 5 to 8 aluminium. The surfaces are cleaned, scraped and heated, then the solder rubbed in until the surfaces are coated. The surfaces thus prepared can then be soldered with ordinary half-tin half-lead solder.

B. C. Senton (U. S. pat. 809,472, Jan. 9, 1906) patents the solder containing 12 parts tin, 2 zinc and 1 aluminium, which ingredients are melted together using one per cent. of common salt as a flux. Antimony may be added to make it harder.

Of the hundred (more or less) solders which have been recommended or patented since Mourey first essayed the problem in 1855, none has entered into commercial use to anything like the extent of the solder of Joseph Richards (U. S. pat. 478,238, July 5, 1892.) This solder contains 29 parts of tin, 11 of zinc, 1 of aluminium and 1 of phosphor tin. It has been manufactured and sold regularly for the last 15 years, the sales in Europe in 1906 being larger than that of any preceding year. Its use is described in Richards' "Aluminium," p. 467.

Welding.

The art of welding aluminium surfaces together, while heating by the hydrogen blowpipe flame, has increased in its applications considerably since it was first exhibited at the Paris Exposition of 1900. It is used considerably in working aluminium in the United States. M. U. Schoop has made a specialty of this art, and visited the United States early in 1907 for the purpose of introducing his latest improvements. He uses an hydrogen-air blowpipe, and a flux, the composition of which is kept secret, which is said to dissolve all grease, oxide, alumina, etc., and enable perfect welds to be easily made. Tests made at the Conservatoire des Arts et Métiers have shown the welds to be equal in strength to pure aluminium. Tubes of any size can be made by spiral welding of sheet.

M. Posno of Paris has also made advances in this art, and has founded the Société Française de Brasure de l'Aluminium to operate his methods.

Uses.

Deoxidation.—Aluminium is largely supplanting phosphide of copper as a deoxidizer in brass and bronze, in which it acts by reducing the oxides of copper, zinc or tin with which the metal may be contaminated. A small excess of aluminium does not injure the metal so much as a small excess of phosphorus. Care must be taken, however, not to cast the alloy immediately after using the deoxidizer, since the alumina formed must be given an opportunity to rise out of the metal and enter the slag. If this is not done, the quality of the metal may be injured by the intermingled alumina. With pure copper used for electrical purposes, silicon is found superior as a deoxidizer to aluminium, because the silica formed is less infusible, tends to unite with copper oxide to a fusible slag, and thus gets out of the melted metal quicker and more completely, leaving it with higher electric conductivity.

Dr. Hans Goldschmidt, the inventor of the process of reducing metallic oxides by powdered aluminium, has recently patented improvements in his method of obtaining fluid iron at high temperature for welding purposes. In place of aluminium as the sole reducing agent acting upon iron oxide, producing the difficultly fusible alumina, he uses a granulated alloy of calcium and aluminium, or a mixture of these two metals in granular form. This alloy gives a very high thermal effect, higher even than aluminium alone, while the heat of formation of the aluminate of lime slag is also utilized, and the slag is much more fusible than alumina alone.

Supposing the alloy of calcium and aluminium to contain 42.5 per cent. of calcium, corresponding to $Ca\ Al_2$, the reaction on Fe_2O_3 would probably be:

$$5\ Fe_2O_3 + 3CaAl_2 = 3(FeO.\ CaO.\ Al_2O_3) + 7\ Fe.$$

In this reaction 70 per cent. of the iron in the oxide used would be reduced to the metallic state, while the resulting slag would be very fluid, and 1.4 parts of fluid iron at high temperature would be obtained per one part of calcium-aluminium alloy used.

Coinage.—It is again rumored that the United States Mint will again try the suitability of aluminium for coinage of cents—a subject which I strongly advocated in 1895.[1]

Electrical Conductors.—The Niagara, Lockport & Ontario Power Company now transmits power 160 miles to Syracuse and 200 miles to Cleveland. The equipment to Syracuse contains 12,000 miles of heavy wire, in three cables of 19 strands each. The voltage is 60,000.[2]

Paint.—The metallic paint of silver white color made of powdered aluminium has deservedly become quite popular for outdoor work. The analysis of a European paint powder, made by Kohn Abrest, showed:

[1] *Aluminium World*, Jan., Feb., March, 1895.
[2] *Metal Industry*, Feb., 1906.

metallic aluminium (trace of iron) 91.20 per cent.; aluminium oxide (alumina) 5.80 per cent.; silica, 1.30 per cent.; insoluble silicon, 0.40 per cent.; carbon, 0.23 per cent.; moisture, 1.07 per cent.; total 100. The large amount of alumina is due to the incipient oxidation of the powder. These powders are often adulterated by zinc and tin dust; mica and similar "fillers" may be likewise present, distinguishable by the large residue insoluble in acid. The powder is manufactured by forcing gas or air into the melted metal while it is setting, accompanied by vigorous mechanical stirring. This granulates the metal, forming a film of crystalline metal partly oxidized, which is easily pulverized. The powdering of this spongy or granulated metal is done in stamp mills or disintegrators, afterward sieving to different sizes and finally polishing in polishing mills.

The powder was originally suspended in various forms of spirit varnish, but it is now used in thin oil varnishes, such as would be made by warming, mixing and stirring well together the following: turpentine 1.5 gal.; palest copal varnish, 0.5 gal.; palest terebine, 4 oz.; magnesium carbonate, 4 oz. The magnesia is allowed to settle and the clear liquid is drawn off. The metallic powder is stirred into this liquid, about 2 lb. of powder to a gallon of liquid. The paint shows up better if applied to a warm surface, and a colorless lacquer on interior work or white copal varnish on outside work protects it well.

Racing Shoes.—The Bryden Horse Shoe Company of Catasauqua, Pa., makes aluminium racing shoes weighing 2 oz. each and selling for $1.25 the set.[1] They have a rim of steel set in edgewise, being cold rolled, slotted, and the steel strip inserted by machinery. Racing men say that, "An ounce off a horses's foot is equal to a pound off his back."

Miscellaneous.—G. R. Gibbons[2] classifies the various uses of aluminium as those based on its low specific gravity, its chemical characteristics, its color, its electric conductivity, and those based on combinations of these properties. He catalogues the new uses as follows: *Light durable plates*—sticking up or transfer plates in lithography, for spreading gelatine in collotype work, die plates for supporting dies of embossing machines, cauls for veneering wood, ivory, etc., press plates for book binders, trays for clipped fur in making hats, shipping receipt binders, stub binders, darbies and hawks for plasterers; *parts of musical instruments*—snare and kettle drum shells, xylophone bars, small bells; in pianos and organs, the molding, tops, bushings, action parts, piano-player fingers, air tubes, organ slides, blowers and valve rings; *machinery*—typewriter and adding machine key levers, platencores, frames, graphophone parts; *miscellaneous*—kodak or camera cases, frames for dress-suit cases, travelling bags, trunks and sample cases, eye-glass cases, cost ledger backs, spirit

[1]*Metal Industry*, August, 1906.
[2]*Metal Industry*, January and February, 1906.

level frames, pans for stearin manufacture, parts of scales, mailing tubes, fishing reels, blocks for forming hats, artificial limbs, gold-washing pans, bowling balls, bench marks, saddle trees, embalming table covers, dowels for stone work, refrigerator linings, beer coolers, dipping baskets (especially for use in nitric acid), aniline evaporating pans, apparatus used in manufacture of gun-cotton, cream of tartar, rubber (mandrels, forms, labels, pans in vulcanizing process), lasts for rubber boots, engine lagging, show-case molding, buckles, telephone diaphragms, mouth-pieces and receiver caps, sounding bars of telephone instruments, electrical meter discs, caps, frames and cases; incubator rheostats, shade roller parts, railway car linings and racks.

Alloys.

Production.—J. N. Pring[1] produces aluminium alloys by reducing metallic oxides by aluminium carbide. The latter is made by heating aluminium in a carbon crucible in a Moissan arc-type furnace. Al_4C_3 results, chiefly from the action of CO on aluminium. This, heated with CuO produces Cu. and CO_2 at low heats:

$$12CuO + Al_4C_3 = 12Cu + 2Al_2O_3 + 3CO_2$$

but at high heats produces:

$$3CuO + Al_4C_3 = Al_4Cu + 3CO.$$
$$Fe_2O_3 + Al_4C_3 = Al_4Fe_2 + 3CO.$$

Any excess of oxide present oxidizes aluminium out of the alloy.

Bronzes.—The strongest is said to be made[2] of 89 parts copper, 10.5 parts aluminium and 0.5 parts silicon, the latter added as silicon copper for the purpose of hardening the alloy. This alloy is not ductile or workable, but must be cast into shape.

According to E. S. Sperry[3] the strongest bronze for bolts, such as are used for anchoring gun carriages, is 9 per cent. aluminium to 91 copper, containing no zinc, tin or other metal, melted with common salt as flux, cast in a 4x4 in. billet, 40 in. long, and rolled in one heat to 2 in. diameter. Such a bolt will show 60,000 ℔. tensile strength per square inch, with 20 per cent. elongation in 8 in. and 20 per cent. reduction of area. Its chief advantage over steel is its non-rusting quality.

E. S. Sperry[4] states that 8 per cent. aluminium bronze will not stand high reductions in rolling. If not frequently annealed it may actually split in rolling into transverse sections. Even if annealed, but reduced too strongly at one pass, it is apt to become overstrained and fatigued, and in course of time, say a year, is apt to develop very pronounced season-cracks. A 4 per cent. bronze is extremely ductile, and draws out splendidly into tubes which develop no season-cracks.

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[1] *Trans.* Chemical Society, LXXXVII, 8530–40; *Electrochem. and Met. Ind.*, January, 1906, p. 27.
[2] *Metal Industry*, May, 1906.
[3] *Brass World*, August, 1906.
[4] *Brass World*, February, 1906, April, 1906.

German-Silver.—E. S. Sperry recommends[1] as a very white alloy, and the stiffest in common use, the mixture of 57 parts copper, 20 nickel, 20 zinc and 3 aluminium. The copper and nickel are first melted together under (borax 2 ℔. to 100 ℔.), the aluminium then added, the metal cooled considerably by gates, etc., until the zinc can be added without loss. If the ingots thus made are remelted, very sound castings may be obtained. Only the best metals should be put into this alloy, viz., Bertha zinc, Lake copper, best grain nickel and No. 1 aluminium. The castings should have large gates, to prevent cracking in the mold, and the alloy poured at a low heat.

Silver.—G. I. Petrenko has made a complete fusion-point diagram of the alloys of aluminium and silver.[2] He finds the two compounds $Al\ Ag_2$ and $Al\ Ag_3$, occupying maxima on the fusing-point curve. Adding aluminium to silver, the melting point falls to a eutectic (minimum) at 567 deg. C., then rises with two breaks at 771 and 721 deg. corresponding respectively to $Al\ Ag_3$ (7.72 per cent. Al.) and $Al\ Ag_2$ (11.15 per cent. Al.) Between these two proportions, the alloys take a good polish. With less than 7.72 per cent. aluminium they show great stability and resistance to atmospheric influences.

Magnesium.—Barnett describes[3] light alloys in which the aluminium is stiffened and hardened by small amounts of magnesium and other metals. A casting alloy is described containing 1.76 per cent. copper, 1.60 magnesium, 1.16 nickel and smaller quantities of antimony and iron; a soft alloy for rolling into sheet contains 0.21 per cent. copper, 1.58 magnesium, 0.72 lead, 3.15 tin and 0.3 iron. These alloys show 8.5 to 10 tons tensile strength in ordinary castings and turn up well. The description lacks definiteness, contains some improbable statements, and does not offer alloys of very good mechanical qualities, after all.

Silicon.—E. Vigouroux states that pure silicon and pure aluminium refuse to combine,[4] but in the presence of a third metal double silicides or alumino-silicides are formed. These are definite crystalline substances with metallic luster, dense, hard and brittle; some are attacked by dilute acids, but most of them resist all acids save hydrofluoric; none is affected by caustic alkali solutions. They may be formed either by heating the three substances together, or by the thermite method of reduction, using mixtures of silica, metallic oxide and an excess of aluminium; or by acting on aluminium by a mixture of the metal, its oxide or its sulphide, and potassium silico-fluoride. Clay vessels must be avoided in handling metals which can form alumino-silicides.

Tellurium.—Cabell Whitehead[5] prepares aluminium telluride by melting

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[1]*Brass World*, March, 1906.
[2]*Zeit. anorg. Chem.*, 1905. *Electrochem. and Met. Ind.*, February 1906, p. 73.
[3]London *Engineering*, February, 1906.
[4]*Comptes Rendus*, 1905, CXLI, 951–3. *Journ. Frank. Inst.*, April, 1906, 318.
[5]*Bull.* George Washington University, V, No. 4 (1906).

aluminium and throwing in small pieces of tellurium from time to time. (The powdered metals, heated together, unite with great violence). The melted aluminium telluride separates perfectly from the excess of aluminium in the crucible.

The telluride decomposes water completely, even in the cold, and a constant stream of hydrogen telluride gas may be obtained by regulating the water supply so that it falls upon the telluride a drop at a time:

$$Al_2Te_3 + 6H_2O = Al_2(OH)_6 + 3H_2Te.$$

Tin and Bismuth.—A. G. C. Gwyer has studied the alloys of aluminium with these metals.[1] With tin, a eutectic is found at 0.48 per cent. aluminium, melting at 229 deg., i.e., 3 deg. below the melting point of pure tin. With bismuth 3.7 per cent. reduces the melting point of aluminium from 657 down to 652 deg., apparently forming a saturated solution of bismuth in aluminium; at the same temperature, the excess of bismuth contains 8 per cent. of aluminium, which amount reduces to zero as the temperature of the mixture is reduced towards the melting point of bismuth. There appears to be no chemical combination even at 1200 deg.

Thorium.—O. Hönigschmid obtains a crystalline alloy of thorium and aluminium by heating the two metals together in a vacuum at about 1000 deg. C., or also by reducing thorium oxide or potassium-thorium fluoride by aluminium.[2] The alloy crystallizes from the aluminium in which it is dissolved in hexagonal needles. It has the color and appearance of aluminium, but is not attacked by aqueous solutions of the alkalies. Analysis shows it to have the formula Th Al_3.

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[1] *Zeit. anorg. Chem.*, 1906, XLIX, 311-9.
[2] *Monatsheft für Chemie*, 1906, XXVII, 205-12. *Journ. Soc. Chem. Ind.*, May 31, 1906.

ALUNDUM.

The manufacture of alundum was begun by the Norton Emery Wheel Company, of Worcester, Mass., in 1904. It is an artificial product formed in the electric furnace from bauxite, and is used as an abrasive. Its chemical composition is exactly the same as that of natural corundum. The production of alundum has been as follows: In 1904, 4,020,000 ℔. valued at $281,400; in 1905, 3,612,000 ℔. valued at $252,840; in 1906, 4,331,233 ℔. valued at $303,186.

Before the invention of the electric furnace, artificial abrasives suitable for grinding purposes were unknown, and manufacturers necessarily depended upon natural products, chiefly corundum, emery, and garnet. Briefly, the process of making alundum consists in taking the mineral bauxite (oxide of aluminum), purifying it and melting in an electric furnace into a large homogeneous mass. Upon cooling this molten fluid solidifies and crystallizes in solid masses of alundum of great purity and uniformity.

The bauxite is heated in large preliminary furnaces to drive off its combined water and is then melted directly in electric furnaces of special design. There are 11 electric furnaces installed at the company's plant at Niagara Falls, N. Y., each furnace being capable of producing three tons of alundum per 24 hours. The temperature at which the bauxite melts into a homogeneous mass is estimated at between 6000 and 7000 deg. F.

After the large masses of molten bauxite have cooled in the furnace, the fusion is broken up by crushers, and passed through rolls to reduce the product to various sizes of grain, which are finally graded by passing through sieves of different mesh in preparation for manufacture into grinding wheels and blocks, polishing stones, etc.

Alundum is much harder than the correspondingly natural product, corundum, represented by the sapphire or ruby, and alundum powder is used for cutting and drilling rubies and sapphires for watch jewels, but its chief use is in the manufacture of "artificial emery" grinding wheels, as they are called.

AMMONIA AND AMMONIUM SULPHATE.

By C. G. Atwater.

The following table gives the ammonium sulphate equivalent of the ammonia produced in various forms (as aqua, anhydrous, sulphate, crude and concentrated liquor, etc.), in the United States for each year since 1898.

TABLE I.—UNITED STATES AMMONIA PRODUCTION, EXPRESSED IN SULPHATE EQUIVALENT.
(Tons of 2000 lb.)

Year	Tons.	Year.	Tons.	Year.	Tons.
1898......	17,000	1901......	(a)29,279	1904......	54,664
1899......	(a)19,500	1902......	36,124	1905......	65,296
1900......	(a)27,600	1903......	41,873	1906......	a75,000

(a) Estimated

These figures are based on the carefully collected data published in "Mineral Resources of the United States" for 1898, 1902, 1903, 1904 and 1905, the intervening years being estimated from the best data obtainable, and an allowance being made throughout for the ammonia produced from other sources than the distillation of bituminous coal. The increase in these figures is principally due to the introduction of the by-product coke oven, which in 1898 produced somewhat over 20 per cent. of the total output, and in 1905 produced more than 60 per cent. The actual ammonium sulphate available for domestic consumption, including that produced as such at plants in this country, and that imported, together with the average market quotations, are shown in Table II.

TABLE II.—AMMONIUM SULPHATE AVAILABLE FOR DOMESTIC CONSUMPTION, INCLUDING U. S. PRODUCT AND IMPORTATIONS, AND AVERAGE MARKET PRICE. (a)
(In tons of 2000 lb.)

	1898	1899	1900	1901	1902	1903	1904	1905	1906
Production, tons......	....	....	....	....	5,638	5,963	14,113	19,332	25,000
Imports, tons.........	8,207	6,976	8,411	14,486	18,146	16,777	16,667	15,288	9,182
Total available for domestic consumption	....	....	....	....	23,784	22,740	30,780	34,620	34,182
Average market price..	$59.09	$50.29	$57.40	$55.16	$59.90	$62.10	$61.71	$62.92	$62.33

(a) The production is given for years ending December 31, while the imports are for the fiscal year ending June 30; this fact will account for some discrepancies.

Data as to the actual production of sulphate for the years 1898 to 1901 inclusive are unfortunately not available. The figures for the ensuing years are from "Mineral Resources." The figures for the average market

price of sulphate are computed from the quotations published by the *Engineering and Mining Journal* at regular intervals throughout the year.

As compared with the small United States production of ammonia, and especially the meager sulphate output, the figures for England and Germany are of interest. This is particularly so if we consider the vast coal consumption of the United States, forming the principal available native source of ammonia, on the one hand, and the present need of ammonia (nitrogen) as a fertilizer in this, the greatest of agricultural countries, on the other hand. The production of the United Kingdom, including that used in the manufacture of soda by the ammonia process, and for other chemical processes, for the last six years, is shown in Table III.

TABLE III.—AMMONIUM SULPHATE AND SULPHATE EQUIVALENT PRODUCED IN THE UNITED KINGDOM. (a)
(Tons of 2240 lb.)

Year.	1901	1902	1903	1904	1905	1906
Gas works.........	143,000	150,000	150,000	150,000	156,000	162,000
Iron works.........	16,500	18,500	19,000	19,500	20,000	20,000
Shale works........	40,000	37,000	37,500	42,500	46,000	46,000
Coke, carbonizing and producer gas...	18,000	23,500	27,500	33,500	46,500	55,500
Total...........	217,500	229,000	234,000	245,500	268,500	283,500

(a) From Bradbury & Hirsch's Review, 1906.

These figures show an output in 1906 equivalent to 317,520 short tons, or four times as much as the United States, a country of twice as many inhabitants. And it is noteworthy that the illuminating gas works supply nearly 60 per cent. of this, whereas they produce less than 40 per cent. of our output. Of the sulphate produced in 1906 the United Kingdom exported 201,500 gross tons, consuming but 82,000 gross tons. Trade conditions are said to have been good, the average price per ton f.o.b. Hull for the year being £12 0s. 9d. as against £12 10s 0d. for the year previous.

The German production of ammonium sulphate and sulphate-equivalent in gas works and by-product coke ovens since 1900 is given in Table IV.

TABLE IV.—PRODUCTION OF AMMONIUM SULPHATE AND SULPHATE EQUIVALENT FROM GAS WORKS AND BY-PRODUCT COKE OVENS IN GERMANY. (a)
(In metric tons, 2204.6 lb.)

Year.	1900	1901	1902	1903	1904	1905	1906
Gas works...........	16,500	17,000	18,000	20,000	21,000	22,000	....
By-product coke ovens	88,500	113,000	117,000	120,000	152,000	168,000	....
Total.............	105,000	130,000	135,000	140,000	173,000	190,000	235,000 (b)

(a) Dr. N. Caro, *Zeit. f. angew. Chem.*, Sept. 14, 1906.
(b) Deutsche Ammoniak-Verkaufs-Vereinigung, 1906.

According to these figures Germany has in six years achieved an increase from 105,000 to 190,000 metric tons, due practically all to the by-pro-

duct coke ovens, and has followed this by an increase in 1906 to 235,000 tons from the same source. This places Germany measurably near to England's yearly production. Fortunately for the latter country the tremendous German increase has been practically taken care of by the home agricultural demand, so that while 38,000 tons were exported, 32,454 tons were imported during the year.

The Japanese demand for sulphate as a fertilizer has largely increased since 1901, and still continues, though in 1906 it fell off a little from the previous year. Japan imported from England the following quantities in gross tons:

1901	1902	1903	1904	1905	1906
1,290	2,429	3,612	14,981	33,861	33,237

It will be noticed that the above figures approach closely the figures for the domestic consumption of the United States in 1905 and 1906.

The estimated ammonium sulphate (and sulphate equivalent) production of the world since 1902 is given in Table V.

TABLE V.—ESTIMATED WORLD'S PRODUCTION OF AMMONIUM SULPHATE.
(In metric tons, 2204.6 lb.)

Country.	1902	1903	1904	1905	1906
England...........	225,300	230,200	241,500	264,400	279,000
Germany..........	135,000	140,000	173,000	190,000	235,000
United States......	33,000	38,000	50,000	60,000	68,000
France (*b*).........	40,000	52,000	43,000	47,300	49,100
Belgium and Holland (*b*)..........	38,000	35,000	(*a*)39,000	24,200	30,000
European Other Countries (*b*).....	45,000	45,000	48,000	55,000	55,000
Total............	516,300	540,200	594,500	640,900	716,100

(*a*) Including Norway, Sweden and Denmark.
(*b*) Estimates from *L'Engrais*.

ANTIMONY.

By John T. Glidden.

In 1906 the United States again became a producer of antimony ore, this being the first production since 1901. The domestic production was not sufficient to reduce the requirement of foreign antimony, but under the stimulus of high prices antimony mining in this country is likely to show a significant increase. The production of refined antimony in the United States is still practically confined to Mathison & Co. of New York, who operate a smelter at Chelsea, Staten Island, New York, and also control the Chapman smelter at San Francisco. A small antimony reduction plant was started up in 1906 at Oakland, California, by C. Solomon, Jr., this being a new producer.

Shipments of antimony ore from the producing mines of Idaho, Utah, Washington and Nevada were made to Oakland and San Francisco during 1906; these shipments averaged 60 per cent. antimony and brought from $2.25 to $2.50 per unit f.o.b. The total shipments of antimony ore in 1906, as reported to us, were 295 tons; there was also some production of low grade ore, which was not shipped.

The chief source of the antimony consumed in the United States is hard, or antimonial lead, which is produced by the silver-lead smelters. Many lead ores contain a little antimony, which is reduced along with the lead and appears to the extent of 0.10 to 2 per cent. in the base bullion. In refining the latter a portion of this antimony is recovered as a by-product—antimonial lead—which ordinarily is made to contain about 20 per cent. antimony. Formerly, this grade of lead sold at a discount, but since antimony has appreciated so highly in value it has commanded a premium. In fact, antimonial lead sells now as a lead-antimony alloy and fetches nearly the full antimony value for its content of that metal, the market price for the alloy being only ¼@½c. per ℔. less than the computed price of its constituents.[1] Antimonial lead is employed largely for making alloys, for which purpose the Hoyt Metal Company, of St. Louis, a subsidiary of the United Lead Company, is the largest consumer in the United States. It is also used for making sheet lead for chemical purposes.

It is expected that the Betts electrolytic process of lead refining will afford a means for economically obtaining metallic antimony from work-

[1]With lead at 6c, per ℔. and antimony at 25c., the value of antimonial lead containing 20 per cent. antimony would be 98,—0.5=9.3c. per lb.

lead, the antimony remaining in the anode slime, and a year ago the Consolidated Mining and Smelting Company, of Trail, B. C., was planning to produce antimony from this source, but we are unaware that it has yet done so. The fact is that a satisfactory way of extracting antimony from this slime has not yet been devised. In the meanwhile, the great rise in the value of antimonial lead has reduced the possible advantage of electrolytic refining in recovering this by-product.

The general conditions of the antimony industry at the end of 1906 are admirably presented by F. T. Havard in an article which follows this general review. The statistics of the United States and the principal foreign countries are given in the accompanying tables.

ANTIMONY STATISTICS OF THE UNITED STATES.
(In short tons.)

Year.	Imports.		Exports.		Production.			Consumption.
	Metal or Regulus.	Ore.	Metal or Regulus.	Ore.	In Hard Lead.	From Domestic Ore.	From Imported Ore. (*a*)	
	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons
1897	573	2,751	....	....	2,217	245	1,100	4,135
1898	1,013	1,863	13	17	2,118	250	738	4,106
1899	1,580	1,991	*Nil.*	*Nil.*	1,586	234	796	4,196
1900	1,816	3,018	21	*Nil.*	2,476	151	1,207	5,638
1901	1,837	866	*Nil.*	25	2,235	50	336	4,458
1902	2,871	840	37	104	2,904	*Nil.*	294	6,032
1903	2,563	1,337	40	*Nil.*	2,552	*Nil.*	535	5,610
1904	2,028	1,245	16	214	2,515	*Nil.*	412	4,939
1905	2,869	988	*Nil.*	*Nil.*	2,561	*Nil.*	395	5,825
1906	3,950	1,124	12	*Nil.*	2,358	150	450	6,866

(*a*) Estimated at 40% extraction from net imports of ore.

AVERAGE MONTHLY PRICES OF ANTIMONY IN NEW YORK
(Cents per pound.)

	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year
1901.													
Cookson's	10.00	10.00	10.00	10.31	10.25	10.25	10.25	10.25	10.12	10.09	10.00	10.00	10.12
Hallett's	9.12	9.22	8.90	8.94	8.75	8.75	8.75	8.43	8.50	8.47	8.37	8.31	8.74
Others	9.25	8.85	8.77	8.73	8.63	8.63	8.63	8.50	8.37	8.34	8.25	8.00	8.55
1902.													
Cookson's	10.00	10.00	9.87	9.87	9.87	9.87	9.75	9.75	9.69	9.44	9.25	9.20	9.71
Hallett's	8.17	8.04	8.06	8.06	8.17	8.25	8.25	8.15	7.92	7.72	7.44	7.25	7.96
Others	7.86	7.75	7.75	7.75	7.90	8.00	8.00	7.90	7.65	7.37	7.22	6.92	7.67
1903.	8.25	8.25	8.25	8.25	8.00	7.50	7.44	7.15	7.00	7.00	6.56	6.75	7.53
Hallett's	7.00	7.00	6.87	6.87	6.75	6.69	6.50	6.40	6.34	6.25	6.25	6.35	6.69
Others	6.75	6.62	6.50	6.50	6.50	6.44	6.25	6.19	6.00	6.00	6.00	5.95	6.31
1904.													
Cookson's	6.938	7.594	7.875	7.875	7.531	7.200	7.188	7.188	6.913	6.984	7.592	8.388	7.439
Hallett's	6.250	6.781	6.825	6.750	6.578	6.438	6.485	6.688	6.537	6.578	7.328	8.160	6.783
Others	5.688	6.203	6.475	6.406	6.203	5.961	5.969	6.062	6.015	6.172	7.204	8.088	6.371
1905.													
Cookson's	8.375	8.375	8.375	8.219	8.406	11.025	12.625	14.500	13.700	13.000	12.500	14.000	11.100
Others	8.063	8.063	7.638	8.125	8.406	10.175	11.875	13.500	12.900	12.000	11.250	12.750	10.400
1906.													
Cookson's	15.0	16.0	17.5	21.31	25.25	26.0	25.25	25.0	24.5	25.2	26.14	26.25	22.78
Hallett's	14.0	15.0	16.5	20.81	24.38	25.0	24.25	24.0	24.0	24.81	25.25	25.24	21.94
Others	13.5	14.25	16.15	20.25	23.31	24.0	23.19	22.75	22.25	23.63	24.50	24.70	21.73

The New York Antimony Market in 1906.

In 1906 there was a remarkable rise in the market value of antimony. Supplies of antimony ore were again rather inadequate and the exhaustion of the more important antimony mines in China made itself keenly felt. The consumption of the metal increased considerably, and the different European governments were large buyers of antimony for ammunition purposes.

The production of antimonial lead, which is gained as a by-product in lead smelting, did not come up to the demands, and low-grade antimonial lead had to be manufactured out of lead and antimony. This low-grade antimonial lead is used to a large extent in storage batteries and as sheet lead in acid chambers.

At the beginning of 1906, antimony was selling at about 14@15c., depending upon the brands, and remained stationary for a few months. During March, however, the demand became so large and supplies so scarce that prices were driven up at a furious rate so that at the beginning of June Cookson's and Hallett's antimony commanded a price of 25@26c., while outside brands could be secured for about ½c. cheaper. After the heavy demand had been satisfied the price settled down. There was, however, at all times a scarcity of spot supplies for the reason that the production at the French and Italian works was frequently interrupted by strikes, and shipments which were contracted for a specified period were, in most cases, not made until several months later.

The market remained neglected until the middle of October, when large contracts were again being entered into by dealers and speculators in this market. Prices then quoted ranged between 24½c. and 25c. for ordinary, 25½c. for Hallett's and 26c. for Cookson's brands. At the end of the year the market was again dull and neglected. This condition can invariably be observed in the antimony market after a spasm of great activity.

Antimony Mining in the United States.

One of the most important results of the increased value of antimony in 1906 was the renewed attention to the deposits of antimony ore in Utah, Idaho, Washington and Nevada.

Hitherto, the great drawback to successful antimony mining has been the difficulty of enriching the low-grade ores to the market standard. The natural sulphide, to be considered first-class ore, must contain 50 per cent. of antimony or more, which implies a rather high degree of concentration, the pure sulphide containing 71.8 per cent. antimony. Ores with less than 50 per cent. antimony are marketed at a serious discount, while the value is moreover affected by the presence of injurious impurities, such as arsenic, lead, and pyrites.

Low-grade sulphide ore can be concentrated by liquation, either in pots or on the hearth of a reverberatory furnace, the molten sulphide draining out of the ore, and flowing to a suitable receptacle. This process is still employed by producers in Japan and Borneo, who are very distant from the market, although even in those parts of the world it is not practiced so much as formerly.

The concentration of antimony sulphide by liquation is a wasteful process. Naturally it has been frequently proposed to substitute mechanical dressing for it. However, the latter operation is attended with difficulty, because of the brittleness of the mineral, stibnite, which is usually disseminated in fine needle-like crystals in a hard quartz gangue, wherefore, in crushing it is largely reduced to fines, which, of course, are the hardest to dress. Moreover, the needle-like crystals into which the mineral breaks make it difficult to size the crushed ore properly by screens. The mechanical dressing of antimony ores has been neglected for these reasons.

In this connection it would probably be worth while to consider the applicability of some of the new processes of ore concentration, such as flotation and electrostatic separation, to the solution of this problem. As everyone knows, there have been great advances in the art of mechanical concentration during the last 10 years, and especially in the development of a new class of processes which can accomplish successful results with certain ores that cannot be worked advantageously even by the most improved methods of jigging and tabling.

Idaho.—In Shoshone county, 90 tons of antimony ore, assaying 60 per cent. antimony, were produced in 1906. The antimony belt of the Cœur d'Alene district is situated along the line of the Northern Pacific Railroad, about midway between Wallace, Idaho, and Thompson Falls, Mont. In only six of the located claims has any definite knowledge of the ore bodies been obtained, and the development is small. It is known that the ore-bearing ledges pitch about 15 deg. and assays of the ore show 12 to 60 per cent. of antimony, 8 to 30 oz. silver, and 90c. to $17 gold per ton.

After selling several carloads of ore from this district in the East at a small profit, it was decided to treat the product on the spot and a furnace was erected which produced an oxide assaying 80 per cent. antimony and over. The furnace was small, producing only 1000 to 1500 lb. of oxide in 24 hours. The oxide was collected in sheet-iron flues and in chambers situated on the hillside. It is reported that these lots of oxide were sold at a good profit before operations were suspended. The plant is idle now on account of dissensions among the owners.

Nevada.—The Antimony King Company owns a mine near Austin, and one at Battle Mountain, which are seven and 10 miles respectively from the nearest railroad. Exploitation of these deposits began in Oc-

tober, 1906, and continued until December, when snow stopped work. In this short time, however, a fair amount of ore, averaging 60 per cent. antimony, was produced and shipped. A party of Japanese are reported to have worked an antimony mine in Lander county.

Utah.—In this State a deposit of antimony ore occurs in the valley of Coyote creek, a tributary of the Sevier river, Garfield county, and is operated by the Utah Antimony Company, whose post office address is Coyote, Garfield county. Stibnite occurs here disseminated through a stratum of yellow, gypsum-bearing sandstone, the mineralized portion of which varies in thickness from four to 60 ft. The antimony mineral forms sometimes fairly regular sheets from half an inch to 8 ft. in thickness and sometimes lenticular masses from one to two feet in thickness at the center and perhaps 20 ft. in diameter.

During 1906 small shipments of hand sorted ore, averaging 60 per cent. antimony, were made from this property to Oakland, Cal. Tests in a small way upon the reduction process which the company proposes to use gave satisfactory results and a plant is to be erected.

Another discovery near Logan, Cache county, was reported during 1906, but no development, was done upon it.

Washington.—The State Smelting and Refining Company, of Seattle, is operating deposits of antimony ore which are believed to be extensive and of fair grade, assays showing from 30 to 35 per cent. of antimony. Several successful tests with a furnace for smelting the ore were made.

At Loomis, Okanogan county, the Lucky Knock mine was operated during 1906. This mine is owned by Messrs. Ingham & Witcher; they produced 40 tons of ore averaging 60 per cent. antimony and shipped it to San Francisco. On account of high transportation charges ore below 40 per cent. cannot be shipped. The company is holding a fair amount of ore running from 25 to 35 per cent. antimony to be concentrated provided development work shows a sufficient quantity of ore to make the installation of a plant profitable. On account of the severe winter of 1906, no ore was mined from the surface and tunnel work only was done.

In the Methow district, the Antimony Queen Mining Company operated the Dixie Queen mine on Gold Creek, in 1906, and produced ore which averaged 50 per cent. antimony.

Antimony Mining in Foreign Countries.

Australia.—The high price for ore stimulated mining in this country. The chief producing center is Hillgrove, N. S. W. Production has increased enormously, jumping from 394 metric tons of ore in 1905 to 2,451 tons in 1906. This increase came entirely from the Hillgrove district. From Woodville and Northcote, in the Hodgkinson field, Queensland, fair supplies of ore were also drawn.

Canada.—The production of antimony ore in 1906 was 562 long tons, which assayed 40 per cent. antimony and 1.5 oz. gold per ton. All of this ore came from the mines of the Dominion Antimony Company, at West Gore, Hants county, Nova Scotia. In addition to the ore of the above grade, the mine is now also shipping ore containing about 20 per cent. antimony and 0.5 oz. gold per ton. In October, 1906, a controlling interest in this property was acquired by the Metallurgical Company of America, which has since been engaged in developing the mine. Plans are now being prepared for the erection of a concentrating mill.

There was no production of antimony ore in New Brunswick during 1906. A company was recently formed, called the Canadian Antimony Company, Limited. This company is reported to have acquired an antimony property at Lake George, New Brunswick, where a somewhat extensive plant was erected some time ago, but was abandoned a short while afterward. The mine is favorably spoken of.

China.—The antimony industry in China is of considerable magnitude. According to B. Giles, acting British Consul at Changsha, a considerable portion of the antimony ore exported from that place goes to the firm of Carlowitz & Co., which has smelting works at Wuchang. A large quantity of ore is also exported to foreign countries, especially to the United Kingdom. There are two antimony smelting establishments at Changsha. The carrying of the antimony is largely in the hands of junks and the customs figures show only a small percentage of the total export. For the year ended December, 1905, the exports are said to have been 37,611 cwt. of regulus and 39,741 cwt. of ore.

Italy.—In Corsica, the Meria mine was reopened and produced 128 tons of ore in 1904. Other mines on this island have resumed work, but it will be some time before any important quantity of ore can be produced.

Portugal.—This kingdom, formerly a producer of some importance, has been losing ground. There are 15 principal producing mines, situated in the districts of Braganza and Douro, which ship their ores to England. A number of less important mines brings the total up to about 35. The reason for the prevailing stagnation in Portugal has been unfavorable political conditions, dependence on foreign capital, and the impossibility of utilizing deposits of medium richness.

As mined, the ore is divided into four classes, containing respectively 65, 54, 45 and 30 to 45 per cent. of antimony sulphide. The first three grades are shipped to England, but the fourth grade will not stand transportation, as long as its silver-gold content is not paid for by smelters. Even in the leanest grade of ores the gold is not less than 5 to 10 grams per ton and the silver not less than 30 grams per ton.

South Africa.—A promising deposit of antimony sulphide and oxide is reported to have been found on the Komatie river, 30 miles south of

Barberton. Other deposits are known to exist in South Africa, but this is the first to make a shipment, having exported in 1906, 15 tons of ore with an average content of 55 per cent. antimony. The cost of transportation to England was $30.75 per ton. There are two parallel lodes on this property, each of which is said to average about 50 per cent. mineral. Two shafts on lode No. 1 are down 50 ft., but the other lode, No. 2, was developed but little up to September, 1906.

PRODUCTION OF ANTIMONY ORE IN FOREIGN COUNTRIES. (a)
(In metric tons.)

	1895	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905
Austria	695	905	864	679	410	201	126	18	41	103	1,673
Bolivia	......	......	......	......	1,213	1,174	190	126	45	......	......
Canada (f)	*Nil.*	*Nil.*	*Nil.*	1,219	6	190	10	82	30	145	(g) 85
France and Algeria	5,703	6,333	5,466	4,571	7,592	7,963	9,867	9,715	12,380	9,065	12,543
Hungary	1,240	1,361	1,800	2,201	1,965	2,373	323	748	205	1,080	......
Italy	2,241	5,086	2,150	1,931	3,791	7,609	8,818	6,116	6,927	5,712	5,083
Japan (b)	1,061	828	348	1,006	712	81	119	88	153	......	......
Mexico (c)	600	3,231	5,873	5,932	10,382	2,313	5,103	1,279	1,856	1,722	.....
N. S. Wales (d)	486	135	172	84	332	252	90	57	13	111	394
New Zealand	55	21	10	......	......	5	30	......	......	......	......
Portugal	753	595	417	245	59	38	126	68	83	31	......
Queensland	......	......	......	......	41	......	......	......	......	*Nil.*	24
Spain	44	54	354	130	50	30	10	67	42	245	77
Turkey	1,322	100	400	(e)	1,173	267	224	(e)	(e)	(e)	(e)
United States	982	136	454	(e)	544	300	100	*Nil.*	*Nil.*	*Nil.*	*Nil.*

(a) From official reports of the respective countries. It will be observed that this table omits the production of China, for which no statistics are available. (b) Mostly crude antimony. (c) Export figures, except for 1903, which represents production. (d) Metal and ore. (e) Not reported. (f) Export figures except for 1898. (g) Includes regulus and salts of antimony.

PRODUCTION OF ANTIMONY METAL IN FOREIGN COUNTRIES.
(In metric tons.)

	1895	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905
Austria	296	422	425	343	271	153	114	24	14	36	90
France	779	969	1,033	1,226	1,499	1,573	1,786	1,725	2,748	2,116	2,396
Hungary (a)	465	500	523	855	940	846	706	683	732	1,007	......
Italy	423	538	404	380	581	1,174	1,721	1,574	905	836	......
Japan	641	517	823	235	229	349	429	528	434	......	......

(a) Regulus.

Metallurgy of Antimony.

This subject is treated so thoroughly in the following article by Mr. Havard that it is unnecessary here to do more than summarize some of the literature of the year. Reference may be made to an excellent series of articles on this subject in the Report of the Department of Mines of Nova Scotia for the year ending September 30, 1906. These articles all come under the head of auriferous antimony ores and are the reports of several investigators who give accounts of the various experiments which they carried on with the view of discovering an efficient means of treating these ores. They contain a large amount of exceedingly interesting and valuable information on the metallurgy of antimony ores, and deal also with attempts at smelting Wilfley table concentrates.

Gold-Antimony Ores.—Of late, special attention has been given to the extraction of gold from antimonial ores. A process invented by F. T. Mason, is as follows: The ore, crushed through a six-mesh screen, is subjected to alternate oxidizing and reducing action in a reverberatory furnace, whereby the antimony is largely volatilized and is collected in condensing chambers. When heavy fumes stop coming over, fine coal is added to the charge and between each rabbling, the ore is heaped up on the hearth in order to help volatilize the remaining antimony. The roasting is continued six hours, during which the charge loses 16 per cent. in weight. All of the gold remains in the ore, while 73 per cent. of the antimony is volatilized. Tests made to extract the gold in the residue by amalgamation and cyaniding showed that 22 per cent. of the gold could be recovered by amalgamation and 53 per cent. by cyanide. Other tests resulted in total gold extraction between 50 and 80 per cent.

Another process is being developed by James S. McArthur of Glasgow. The finely crushed ore is leached with some alkali, such as caustic soda, of not more than 4 per cent. strength, at a temperature of not over 50 deg. C. The solution is filtered and precipitated by carbon dioxide, assisted by cooling. The precipitated antimony is collected in a filter press, and the filtrate after having been causticised by lime is ready to be used over again. By a combination of this process with roasting and chloridizing the insoluble residue, followed by amalgamating and cyaniding, it is claimed that 95 per cent. of the gold can be extracted. With caustic soda at £12 per ton, the estimated cost of treatment per ton is 15s. 6d., divided as follows: crushing, 2s. 6d.; leaching with soda, 7s. 6d.; roasting, 2s. 6d.; amalgamating and cyaniding, 3s. 0d; total, 15s. 6d. Other solvents of the ore which may be used are sodium or calcium sulphides. Sulphurous acid may be used as a precipitant of the antimony.

It was stated in Vol. XIV that the McArthur process had been applied successfully by the Dominion Antimony Company for the treatment of the auriferous stibnite of West Gore, Hants county, Nova Scotia, but this statement seems to have referred only to experimental trials, and so far as we are aware the process has not yet come into commercial use.

H. L. Herrenschmidt (French pat. 349,980, June 11, 1904) states that mixed ore containing gold and antimony, according to one process, is placed in an autoclave with substances capable of giving (by reaction in presence of water on boiling under pressure) sodium sulphide or calcium sulphide. Or, the ore is boiled in the open with the same reagent, with or without previous roasting. If, however, the ore is roasted, it may be treated either with sodium sulphide or with sodium hydroxide. The antimonial solution obtained may be precipitated by sulphurous acid to

recover the antimony; the gold in the ore thus purified is recovered by cyaniding or other suitable method.

The treatment of auriferous antimony ore for extraction of both the gold and the antimony is no longer the troublesome problem that it used to be. Indeed, it appears to have been solved beautifully by the French process of antimony smelting (see the following article by Mr. Havard). In this process the antimony is volatilized, the gold remaining in the residue, which is readily salable with the lead smelters. Under the conditions of operation the volatilization of the antimony is nearly complete, and little or no gold is dragged off with it.

Analytical Methods.

The gravimetric process of determining antimony is so tedious, and the care necessary to insure a pure precipitate and to avoid volatilization while heating is so great, that volumetric processes are very desirable. Rothwell[1] suggests the following modification of Györy's process, which depends on the oxidation of antimonious to antimonic chloride by potassium bromate in hydrochloric acid solution, according to the equation:

$$KBrO_3+3SbCl_3+6HCl=3SbCl_5+KBr+3H_2O$$

The alloy sample is cast into thin plates in order to avoid segregation during cooling. This sample is cut with a hack saw and the filings sized on a 30-mesh screen. The undersize on 30-mesh is used for the analysis sample. Tests show that the rejected filings coarser than 30 mesh are of the same composition as the undersize and may safely be rejected. One gram of ore or filings are weighed into a beaker and 25 c.c. of concentrated HCl, and 5 c.c. of a saturated solution of bromine in hydrochloric acid are added. The covered beaker is set on a hot plate but is not heated so strongly as to drive off the bromine before solution is effected.

Antimony oxides and precipitates of mixed oxides, which do not readily dissolve in this manner, may be fused with eight times their weight of caustic soda in a silver crucible at a dull red heat until the mass turns green. Dissolve the fusion in as little water as possible in a 500 c.c. beaker, acidify with HCl and evaporate down to 10 c.c., when 20 c.c. more of HCl are added.

To reduce the antimony in preparation for titration, sodium sulphite, or sulphurous acid, is used, and the solution is boiled down to insure volatilization of arsenic. If more than 2 or 3 per cent. of arsenic are present, 20 c.c. of strong HCl and 5 c.c. of H_2SO_3 are added and the liquid boiled down again. Add to the thus concentrated solution, 20 c.c. HCl (1.2.) and 40 c.c. hot water and boil one minute to remove traces of H_2SO_3. Now run in with constant stirring a standardized solution of $KBrO_3$ to

[1] *Journ. Soc. Chem. Ind.*, Dec. 31, 1906.

within a few cubic centimeters of the correct amount. If lead chloride begins to crystallize the solution must be boiled again, but otherwise two drops of methyl orange solution are added and bromate run in cautiously again until the color of the indicator is destroyed. The solution should never fall below 60 deg. C. in temperature and must be stirred well in order that a local excess of bromate should never form, for its strength will be lost before attacking the antimony. To calculate the result, subtract from the number of cubic centimeters of bromate required by one gram of sample, the number of cubic centimeters of bromate required by a blank assay; multiply this remainder by 10 and divide the product by the number of cubic centimeters of bromate required to oxidize 50 c.c. of the standard arsenic solution; the result is the per cent. of antimony in the one gram sample.

The bromate solution is standardized as follows: Take 0.8236 gram of pure powdered arsenious oxide and dry carefully at 100 deg. C. Place in a 500 c.c. flask, add 5 c.c. of a 10 per cent. solution of NaOH and shake and warm until dissolved. Add 5 c.c. HCl (1.2.) and make up the solution to 500 c.c. Take 50 c.c. of this solution (equivalent to 0.1 gram of antimony) and add 20 c.c. HCl (1.2). Bring to a boil and titrate with $\frac{N}{20}$ solution of bromate as described above. This standard solution of arsenious oxide if carefully kept will last a fortnight without oxidation.

Iron and copper interfere somewhat with this method, iron tending to make results high but not in so great a ratio as does copper. One fairly good way of correcting for the copper when small in amount is to subtract 0.012 per cent. of antimony for every 0.1 per cent. copper, determining the latter metal colorimetrically. Under ordinarily favorable conditions, this whole procedure for antimony requires only an hour and a half for a determination after the weighing.

Bibliography.

The subject of antimony is treated in every volume of The Mineral Industry. Vol. I contains an article of 12 pages by Prof. J. F. Kemp, in which the characteristics of antimony minerals are touched upon and a very complete account of the metallurgy of antimony is given, including all the essential features of reducing this metal from its ores, refining and casting. Vol. II contains accounts of deposits of ores of antimony, and their distribution throughout the United States and Europe. The uses of antimony as a constituent of various alloys are enumerated. A brief description of antimony mining and smelting as carried out in Japan is also included in this volume. In Volume VII there is a brief description of a process proposed by N. W. Edwards, of London, for treating gold and silver bearing antimony ores, by leaching with a 7 per cent. solution of calcium sulphide. Two methods for the volumetric determination

of antimony are described. Vol. VIII sums up the result of an investigation by Prof. S. Behrens, to determine the properties of babbitt metal. There is also a discussion of the methods of antimony ore analysis suggested by Thomas Brown. In Vol. IX the electrolytic deposition of antimony from its ores by the use of a porous diaphragm is briefly mentioned. Vol. X describes a new continuous method of antimony smelting, patented by Thomas C. Sanderson, of Chelsea, Staten Island, N. Y. The procedure for determining antimony and arsenic in their precipitated sulphides, as proposed by L. D. Skinner and R. H. Hawley, is given in detail. Vol. XII contains a modification of Györy's method for estimating antimony in lead. Vol. XV discusses the production and market for antimony and takes up the progress in the metallurgy of this metal, giving the result of the investigations of A. G. Betts in the electrolysis of antimony solutions. The McArthur process of extracting gold from antimony-gold ores in Nova Scotia is likewise mentioned. The specifications for bearing metal enforced by the German railroads in regard to the composition of babbitt metal are given. The article concludes with a description of the procedure to be used in the electrolytic determination of antimony and tin as reported in the investigations of Arthur Fischer.

The following articles relating to antimony appeared in the technical and scientific press during 1906:

"Antimonial Lead" (*Brass World,* Dec., 1906). Discusses the suitability of this material for artistic castings.

"Alloys of Antimony and Tin." F. E. Gallagher (*Journ. of Phys. Chem.*, Vol. X, 1906; pp. 93-98). Results of a microscopical examination of these alloys, the conclusion being that they do not form any chemical compound, but rather are solid solutions.

"Auriferous Antimonial Ore—Methods of Treatment." A. Selwyn-Brown (*Aust. Min. Stand.*, May 30, 1906). Refers to new methods for the treatment of this class of ore.

"Ueber die technische und ökonomische Ausbeutung von Antimon- und Wolframerz in Portugal." E. Ackermann (*Chem. Zeit.*, July 11, 1906). Mining, industrial and metallurgical notes respecting the production of antimony ore and wolfram ore in Portugal.

"The Volumetric Estimation of Antimony," James Darroch (*Min. and Sci. Press,* Jan. 19, 1907). Gives the procedure of a new method for determining antimony by volumetric means similar to the iodide method, but substituting stannous chloride for sodium thiosulphate.

"Arsenic and Antimony—Discussion of determinations for arsenic and antimony and a proposed scheme for the same." L. B. Skinner (*West. Chem. and Met.*, Dec., 1906). Review of various standard methods of determining these elements, with comments upon their weak points and an outline of a proposed new method of arsenic determination.

"Determination of Antimony and Tin in Babbitt, Type Metal, or other Alloys." W. H. Low (*Journ. Amer. Chem. Soc.*, Jan., 1907). Gives a discussion of various procedures which are in use for determining these two metals in alloys and describes a new procedure which is to overcome the defects of the older methods.

"Separation de l'arsenic dans l'antimoine." H. L. Herrenschmidt (*Rev. des Produits chimiques*, July 1, 1906). A method of separating arsenic from antimony either as oxide or in solution.

"Antimony in Babbitt and Type Metals." H. Yockey (*Journ. Amer. Chem. Soc.*, Oct., 1906). A proposed method for the determination of antimony and its alloys.

The following patents, relating to antimony, have been issued:

Brit. pat. 16,490 of 1905. Antimony pigment. L. Brunet, Brionde, France. Producing pigments of antimony and arsenic sulphides by acting on the alkaline sulpho-antimoniates, or sulpho-arseniates, with sulphurous acid produced by roasting sulphides, the by-product being an alkaline hyposulphite.

Brit. pat. 16,490A of 1905. Antimonial pigment. L. Brunet. Producing a white antimony pigment by first making sulphides of barium and lime and then mixing them with oxides of antimony.

Brit. pat. 17,123 of 1905. Alloy. A. E. Hobson, New Haven, Conn. A new alloy to be used as a substitute for britannia metal, being lighter and tougher than the latter, consisting of 109 ℔. of tin, 3 ℔. of copper, 9 ℔. of antimony and 3 oz. of manganese.

Brit. pat. 10,513 of 1906. Bearing metal. Siemens & Halkse, Berlin. An improved alloy for use in bearings, consisting of equal parts of zinc and cadmium, with about 10 per cent. of antimony.

The Antimony Industry in Europe.

By F. T. Havard.

The extraordinary rise in the price of antimony has been variously attributed, in reports of dealers, importers, smelters and consumers, to increased demand in the machine industries; to the partial replacement of tin, now scarce and expensive, by antimony in alloys, babbitt, and bearing metal; to a change in the quality of the ore coming on the market from high-grade to low-grade, as workings in the mines reached greater depths; to the general increase in prices of all metals as compared with the constant value of gold; to speculative and other causes. These assigned causes, though containing many elements of truth, are not sufficient to account for the very unusual rate at which the price of the metal has risen from its former normal rate of about £40 to £110 and £120 per ton. This sudden rise is due more probably to a drying up of some former

sources of metal supply, which indeed has come to pass in the following way.

If we study the table of antimony ore producers given in The Mineral Industry for 1905, we find that the greater quantity is supplied by mines in France and Italy. This ore is treated at the works of various metallurgical companies in the two countries. Of these companies the most prosperous and the wealthiest is the Société Anonyme Franco-Italienne, whose head office is at Genoa. Now, whereas formerly both the Italian and the French smelteries belonging to this company produced regulus alone from the treatment of its ores, at the present time its Italian works make no metal, but only pigments; and the French smeltery makes both metal and pigments, but is increasing its paint production at the probable expense of the regulus output.

New Uses for Antimony.—Both the Italian and the French governments are encouraging the use of antimony pigments, which have the advantage over white lead of being innocuous, and over zinc paint of being permanent and sun proof; so that, favored with the growing demand in England for these pigments and with the protection of the Continental governments, the Société Anonyme Franco-Italienne is not likely to change the policy it adopted several years ago of replacing its production of metal by that of oxides, sulphides, and other salts. It is probable that the example of the French company has been followed by other works.

If we examine the other attributed reasons for the present condition of the market, we find the soundest is that of the increased demand in the machine and industrial world, where antimony, as above stated, is obtaining a wide application in the manufacture of bearing metal and other alloys and is replacing soft lead for acid-proof linings and for other purposes. Further, the sulphide is being used in greater quantities than ever in the rubber industry for the purposes of vulcanization.

Increased Demand for Hard Lead.—It was, however, just at the time when the French and Italian regulus production was being withdrawn, that the first tightening of the market was felt. This showed itself, first of all, in an increased demand for hard lead with a corresponding rise in its price, and was afterward followed by the rise in the price of regulus. At the beginning of 1905 antimonial lead was selling at a slight advance on the price of soft lead; in May, 1905, dealers offered for 20 per cent. hard lead £1 per ton over the price of soft lead. On applying, however, to consumers it was found that the demand was much greater than the supply, so that in June £2 10s. to £3 over-price was obtained. In October this over-price increased to £4, although antimony was quoted at between £50 and £60, while in December £4 10s. was obtained and prices have increased since then in almost direct ratio to the increase in the price of regulus.

Value of Antimony Ore.—With regard to the reports of the scarcity of ore and particularly of high-grade ore, and the suggested consequent diminution of the supply of antimony, we find that the argument does not bear examination, because, although the ore on the market is of low grade, much more is being sold now than at any other time; so that the actual quantity of antimony going to metallurgical works is increased rather than diminished. As to the question of speculation, buying in and "cornering" of the supply, it is true that dealers and brokers have attempted to coerce the market, and have to a certain extent succeeded. It is reported, for instance, that one French dealer in Lyons bought in large quantities in the early part of the year, and he is supposed to have realized at a considerable advantage. We may, however, assume that any stocks now held by dealers have been bought at such a price as precludes selling at less than £80 or £90 per ton. Accordingly, unless fresh and large sources of supply drown the market with ore, the prices will probably be maintained for some time.

With regard to the price paid per unit of antimony in ore:

In 1904, 4@5s. per unit was expected for 50 per cent. sulphide ore, when regulus was worth £25@£35.

In 1905, 6@8s., when regulus was being sold at £35@£60.

In 1906 prices have ranged from 8 to 15s., and in some cases above 15s., when the price of regulus rose from £60@£120. In fact, in May, dealers were still offering ores to smelters on the old formula: $0.009\ T\ (P - 330) = V$, c.i.f., in which T represents the content of antimony in percentage, P the market price of the regulus, and V the value in francs c.i.f. Havre, Marseilles, Antwerp or Hamburg. Of course the price for low-grade, mixed, and refractory ores was considerably less per unit of antimony.

Classes of Ore.—For the purposes of discussing the prices paid for antimony-carrying materials, and also with a view to describing the various processes used for extracting the metal, I shall classify the different kinds of ore now appearing on the market under the following heads:

(a) Auriferous, with 40 to 70 per cent. of antimony in the form of sulphide, several ounces of gold, together with, in some cases, payable silver. The chief sources of these ores for European markets are Australia, France, Italy and Hungary. The value per unit of antimony is 13@17s.; gold is generally paid for at 75s. per oz.; sometimes, however, no pay is given for the precious metals.

(b) The same class of auriferous ore, but with less than 40 per cent. of antimony; sometimes with payable silver. At the present time this is worth from 8@13s. per unit of antimony. Pay is given for precious metals as under (a).

(c) Antimony ore, both sulphide and oxide, free from precious metals, with from 40 to 70 per cent. antimony. Sources, Austria-Hungary, Australia, Iberian peninsula.

(d) The same class of ore with under 40 per cent. antimony, from the same sources.

(e) Mixed copper-antimony-zinc ores nearly always carrying some lead and silver, in which antimony is paid for at prices varying from the price of lead to 7s. per unit.

(f) Lead concentrate and high-class lead, ore, with 40 to 60 per cent. lead, 3 to 10 per cent. antimony and generally carrying some silver. Antimony is generally sold at lead price by the mines, and also sometimes by dealers on the consideration that they receive the resulting hard lead at an advantage on the market price.

Of these ores the antimony smelters, using what is known as the English process, can treat only (a) and (c); pigment makers and smelters using the French process (a), (b), (c) and (d); only general metallurgical works will touch (e); while (f) is eagerly sought by lead smelters. The ores (b) and (d) are relatively cheaper than any other class.

Methods of Smelting.—The processes now in use for the recovery of the metal may be classified under the names of (1) liquation process; (2) crucible process; (3) open-hearth process; (4) the English process; (5) the French process.

1. *The Liquation Process.*—This is used now only in remote and primitive places. The sulphide is placed in a crucible of flower-pot shape, having a hole at the bottom. Under this is placed another small crucible. The first crucible is heated by a charcoal or coke fire placed around it; the fused antimony sulphide runs through the hole and collects in the lower pot. Most of the gold and silver remains in the sinter, which averages about 8 to 10 per cent. antimony. Only rich sulphide ores are amenable to this process. The crude product is shipped to European smelteries to be refined.

2. *The Crucible Process.*—The ore is generally placed in crucibles of small dimensions, with a suitable reducing agent, such as metallic iron for sulphides, and carbon for oxides of antimony. The crucibles are placed in a reverberatory furnace and exposed to heat until the reduction of the ore to the metallic state has been effected. Metallic antimony forms at the bottom of the crucible; floating upon it is a mass of sulphide of iron in the more general case of treatment of sulphide ores, and slag formed by the fusion of the gangue. The antimony is obtained in a crude state by pouring the slag and iron sulphide off the top and casting the metal into molds.

This process is clumsy and wasteful. Any gold and silver content of the ore is divided between the sulphide of iron and the crude antimony regulus, and in most cases no return is given for it by the buyers of these two products. It is costly, owing to the rapid deterioration and final destruction of the crucibles employed; to the loss of metal as fume in the early stages of the process; to the large relative amount of labor necessarily employed;

the quantity of plant required to produce a ton of the metal; and because of the fact that the final product is very impure and contains so much iron that the process of cleaning and refining is tedious and expensive.

3. *The Open-Hearth Process.*—This is an improvement on the crucible method, and enables larger quantities to be treated by the same reduction process. The ore, mixed with reducing agents, and possibly with a flux for the gangue, is smelted by a reverberatory flame on an open hearth. The slag is drawn off the top of the bath, and the antimony metal tapped from the furnace. The loss of metal owing to volatilization during the heating and working of the charge is so considerable that the process can scarcely be called more successful than the crucible method, while the resulting regulus is but little, if at all, purer.

4. *The English Process.*—This consists in dropping the preheated charge of antimony ore through a hopper into a fused bath of iron sulphide and iron of very high temperature, from such a hight that it splashes through the bath. Thus the particles of ore are distributed and exposed to the fierce attack of the hot bath, whereby rapid decomposition of the ore ensues. Attempts are made to minimize loss by closing the flue dampers when charging; by preheating both reducing agent and ore, so that the reaction may take place in the shortest possible time; by securing a covering of iron sulphide on the ore when decomposition is taking place; and by working the charge rapidly. After the partial decomposition of the ore, the bath consists of slag and sulphide of iron, together with considerable antimony sulphide mixed with it; underneath this is whatever antimony has separated out.

To free the iron sulphide of contained antimony, more iron is now added. By this means the antimony is parted from the sulphur, which combines with the iron, and sinks to the bottom as metal. This crude antimony metal is tapped from the bottom of the furnace, while the slag and such excess of iron sulphide as is not required for the next process is drawn off from the top. The remaining iron sulphide is again heated, and the furnace is ready to receive the next charge. This is the best of the three reverberatory processes mentioned. It works quickly and cheaply, but yields an impure regulus, which necessitates tedious resmelting in crucibles or hearths. The gold and silver content of the regulus is high. Some of the precious metals are lost in the slag and iron sulphide drawn off from the bath; but the greater part of these concentrate in the iron sulphide bath, which is sold eventually to silver-lead smelters, when a return is obtained for the precious metals. If the regulus is refined electrolytically, the antimony is recovered free from precious metals. For information on this electrolytic process I refer readers to THE MINERAL INDUSTRY, 1905, Vol. XIV, pp. 24-26.

Notwithstanding its defects, the companies using the English process

do very profitable work. They estimate their working costs, including loss, at from 200 to 250s. per ton of pure regulus produced.

5. *The French Process.*—This is a volatilization process. In it the heat used to accomplish the volatilization of the antimony, which is deposited in the form of oxide, reduces the oxide to metal in an open-hearth reverberatory furnace with a refractory basic lining, and in some cases also generates the power necessary to drive the plant. The process is simple and complete in two simultaneously working operations; namely, volatilization of the antimony sulphide in a shaft furnace and reduction of the oxide in an open hearth. The gold and silver content of ore is collected in a cinder at the bottom of the shaft furnace. Loss of metal is controlled by sufficient equipment for collecting the fume. The resulting regulus is quite pure and is sold in the market as "French star." The reduction of the oxide, which is mixed with charcoal and soda, is effected by the reducing atmosphere of the furnace. Some slag forms above the bath of regulus. This is either reduced again or is sold to alloy makers. Most kinds of antimony ores are amenable to this process. The working costs amount to about 200 francs per ton of clean regulus produced from ore containing 50 per cent. antimony.

Other Processes.—In addition to these processes, various wet and dry methods of winning antimony in the form of paints or regulus have been proposed. Of these, the method of A. S. Plews, U. S. patent No. 704,367, for making white antimony oxide is instructive and interesting.

With regard to the treatment of ores under (f), considerable progress has been made in the desilverization of hard-lead bullion and in the manufacture of alloys rapidly and cheaply from lead concentrates carrying antimony. Without going into detail it may be stated that the principle made use of is that adopted in refining base lead, namely, the greater affinity of antimony over other metals for oxygen. In treating these and other kinds of mixed ores, every metallurgist works out his own salvation; local conditions determine the modifications necessary to insure progress and success, and must of course be carefully studied if that salvation is to be permanently secured.

ARSENIC.

By Edward K. Judd.

The production of white arsenic in the United States in 1906 amounted to 1,663,000 lb., with a value of $83,150, as compared with 1,545,400 lb. worth $50,225 in the previous year. The consumption of this material in this country is largely supplied by imports, as the statistics in the following table indicate:

ARSENIC STATISTICS OF THE UNITED STATES.

Year.	Production.			Imports.			Consumption.	
	Pounds.	Value.	Per lb.	Pounds.	Value.	Per lb.	Pounds.	Value.
1897	....	....	....	7,242,004	$352,284	$0.05	7,242,004	$352,284
1898	....	....	....	8,686,681	370,347	0.04¼	8,686,681	370,347
1899	....	....	....	9,040,871	386,791	0.04¼	9,040,871	386,791
1900	....	....	....	5,765,559	265,500	0.04¾	5,765,559	265,500
1901	600,000	$18,000	$0.03	6,989,668	316,525	04.0½	7,589,668	334,525
1902	2,706,000	81,180	0.03	6,110,898	280,055	0.04½	8,816,898	361,235
1903	1,222,000	36,691	0.03	7,146,362	256,097	0.03½	8,368,362	292,788
1904	996,456	29,504	0.03	6,391,566	226,481	0.03½	7,388,022	255,985
1905	1,545,400	50,225	0.03¼	6,444,083	219,198	0.03⅓	7,989,483	269,423
1906	1,663,000	83,150	0.05	7,639,507	336,609	0.04½	9,302,507	419,759

The increased demand and higher price for white arsenic have greatly stimulated attention to the recovery of this substance as a by-product from many ores in which heretofore it has been disregarded. The manufacturers of sulphuric acid produce a considerable amount of sulphide of arsenic in purifying their acid, which at present is thrown away. It is possible that some economical process for the utilization of that waste product may be devised if attention be given to it. Indeed a process proposed for this purpose is outlined further on in this article. Another possible source of arsenic is the work-lead of the silver-lead smelters, which sometimes contains as much as 0.5 per cent. arsenic, and frequently contains 0.3 per cent. The introduction of the electrolytic process of lead refining may easily lead to the recovery of a portion of this arsenic either as oxide or as arsenical lead suitable for shot manufacture.

A further potential source of arsenic is the anode slime of the electrolytic copper refiners. In American copper refineries high-grade anodes are universally used, carrying 98 to 99 per cent. copper, up to 300 oz. silver per ton, and up to 40 oz. gold per ton. The arsenic is sometimes as high as 2 per cent., and there are small amounts of antimony, bismuth, iron, nickel, sulphur, selenium, tellurium and silicon. Arsenic is the

most difficult impurity to deal with. The slimes commonly contain 40 per cent. silver, 2 per cent. gold, 25 per cent. copper, 5 per cent. selenium and tellurium, 10 per cent. arsenic and antimony, and 18 per cent. lead, silica, sulphuric acid, etc.

At present nearly the entire output of white arsenic in the United States is derived as a by-product in the smelting of copper and gold ores; the deposits of purely arsenic ore, though numerous, either are too remote from railways or are of no great magnitude. The producers of white arsenic in 1906 were: The Puget Sound Reduction Company of Everett, Wash., a subsidiary of the American Smelters Securities Company which recently purchased it; the Anaconda Copper Mining Company, operating the Washoe smeltery at Anaconda, Mont.; and the Mineral Creek Mining and Smelting Company, which owns a deposit of realgar and a reduction plant at Mineral, Lewis county, Wash.

The Everett plant gets its mispickel ores from mines in the Monte Cristo range in Washington, and its operations during 1906 were on the same scale as during the previous year. The Washoe smelter uses the Butte copper ores, in which enargite (Cu_3AsS_4) is an important constituent, and part of the flue-dust is further treated to recover its arsenic. This plant was worked continuously throughout 1906, and in the course of the year its roasting capacity was doubled, so that it now yields an output of two tons of arsenious oxide per day. The process is described on a following page. The Mineral Creek plant made an experimental run in 1906, yielding a few tons of white arsenic, and is expected to produce regularly in 1907. There was no output from new producers in 1906 in either Montana or Washington.

Among the potential producers of arsenic may be mentioned the United States Arsenic Mines Company, of Pittsburgh, Pa., and the Summit Mining Company, of Wellsville, N. Y. The former has an expensively equipped reduction plant at Rewald, Va., but for more than a year now its activities have been confined to mine development so as to provide a sufficient ore supply. The Summit Mining Company, on the other hand, owns a fairly well developed vein on White Horse mountain, in the Darrington district of Washington, but finds it necessary to build a smelting plant at the mine. The Putnam County Mining Corporation, of New York, took over 1000 tons of arsenical pyrites from its mine in Putnam county, N. Y., of which about 600 tons averaging 28 per cent. was exported. The ore contains: SiO_2, 2.90; S, 22.72; As, 36.00; Cu, 2.17; Fe, 36.11 per cent.

Market and Prices.—In common with many other metallurgical products, white arsenic during 1906 showed a strong advance in value. The price of white arsenic in America is influenced primarily by fluctuations of the English market, while that of the sulphides and the arsenic salts is

governed mainly from Germany. The makers of paris green are the largest consumers of arsenic, and in calico printing white arsenic is indispensable as a mordant. The several arsenic compounds are used in varying amounts also in the manufacture of colors, wood preservative, sheep dip, in the tanning industry and in medicine, and a small proportion of metallic arsenic is added to the lead intended for shot manufacture.

The ruling prices for white arsenic at New York during 1906 ranged as follows in cents per pound: January, $4\frac{1}{2}$ @ 7; February, 7 @ 12; March, $6\frac{1}{2}$ @ 8; April, $5\frac{1}{4}$ @ 8; May, $5\frac{1}{8}$ @ 6; June, 5 @ $5\frac{3}{4}$; July, $4\frac{7}{8}$ @ $5\frac{1}{4}$; August, $4\frac{7}{8}$ @ $5\frac{1}{4}$; September, 5 @ $6\frac{1}{2}$; October, $6\frac{3}{4}$ @ 7; November, $7\frac{1}{8}$ @ 8; December, $6\frac{3}{4}$ @ $7\frac{1}{4}$. Average for year, 6.448. These prices refer to the trade of retail dealers in New York for reasonably large lots, and do not represent the prices received by the manufacturer. The average price obtained by the latter was 5c. per lb.

WORLD'S PRODUCTION OF ARSENIC.
(In metric tons.)

Year.	Canada. (a)	Germany. (b)	Italy.	Japan.	Portugal.	Spain. (a)	United Kingdom. (a)	United States. (a)	France. (d)
1896	*Nil*	2,632	320	6	....	271	3,674	....	....
1897	*Nil*	2,987	200	13	524	244	4,232	....	....
1898	*Nil*	2,677	215	7	751	111	4,241	....	....
1899	52	2,423	304	5	1,083	101	3,890	....	2,600
1900	275	2,414	120	5	1,031	150	4,146	....	4,705
1901	630	2,549	....	10	527	120	3,416	272	7,491
1902	726	2,828	....	12	736	71	2,165	1,226	5,372
1903	233	2,768	....	6	698	1,088	916	554	6,658
1904	66	2,829	....	(c)	1,370	400	992	452	3,117
1905	*Nil.*	2,535	....	(c)	(c)	(c)	1,552	701	3,627
1906	*Nil.*	3,052	....	(c)	(c)	(c)	(c)	754	(c)

(*a*) Arsenious acid. (*b*) Oxide, sulphide, etc. (*c*) Not yet available. (*d*) Ore.

Arsenic Production in Foreign Countries.

Canada.—Arsenical gold ore exists at Deloro, Ont., and this, together with other occurrences, has been worked for the production of white arsenic; but the industry never attained a secure footing, and in recent years the production has been insignificant. Some of the ore of the Cobalt district is highly arsenical and it is probable that in the near future it will be attempted to recover arsenious oxide as a by-product in its treatment for gold and silver extraction. The arsenic content of the ore produced in the Cobalt district in 1906 was 1919 tons. Some of the ore assays as high as 55 per cent. arsenic.

France.—The increasing shortness in the supply of arsenic has led to a considerable investigation of several arsenic deposits, particularly of mispickel, in France. Among others, the following new companies may be noted: Société de la Bellière, whose mine is at St. Pierre-Montlimart,

near Montrevault in Maine-et-Loire. The gold-bearing mispickel here was worked by the Romans. It is intended to erect at the mine a complete plant for recovering both the gold and the arsenic. Le Mispickle d'Auvergne will develop deposits at Chapdes-Beaufort in the canton of Pontgibaud, Puy-de-Dôme. More than 40 distinct veins have been exposed. A Herrenschmidt arsenic furnace will be erected at once and another plant for treating the gold-silver residues. Another company, Carballino Gold and Arsenic Mines, will develop a deposit at Carballino, Orense, the ore of which is said to yield 500 kg. of arsenious oxide and 54 grams of gold to the ton. L'Arsenic will exploit a deposit of orpiment at Luseran.

Progress in Technology.

The Washoe Arsenic Plant.—According to L. S. Austin,[1] out of the total quantity of flue-dust removed from the main flue of the converter plant, 60 to 80 tons daily, about 22,000 lb. of the finest portion, collected in the double flue, is taken to the arsenic plant. The cars in which the dust is received are covered, and hold from 1400 to 1700 lb. of flue-dust, which, as it is received in them, weighs about 55 lb. to the cubic foot. The dust is dropped into the feed hopper of a Brunton revolving-hearth roasting furnace of 14 ft. diameter. The charge, as the hearth revolves, is stirred by sets of blades, 33 in all, which pass through the roof of the furnace. The capacity of the furnace is 22,000 lb. daily, and another one has been added, doubling the output. The roasting is performed at a low heat, being at the finish of a just visible red (480 deg. C.), the heat being kept up by a wood fire in a fire-box adjoining the hearth. The hearth itself revolves 10 times an hour, and the dust delivered at the axis is discharged at the periphery, arsenic free, into hopper cars set below in a tunnel. The arsenic fume escaping from the roaster is conducted through a flue 240 ft. long, 18 ft. wide, and 7 ft. high, to an outlet pipe connected directly to the main flue. The arsenic flue is interrupted every 7 ft. by baffle walls, upon which and upon the floor of the flue the crude arsenic condenses and accumulates in crystalline form, containing from 85 to 92 per cent. of crude arsenious oxide.

When a sufficient quantity of these crystals has accumulated in the flue, the roaster is shut down, and the condensed fume, containing 85 per cent. arsenious oxide and copper 4.3 per cent. with 8.1 oz. silver per ton and 0.025 oz. gold, is removed for further refining. This operation is performed in a reverberatory roasting furnace in six-ton charges, the arsenic being volatilized and again condensed in another flue, of dimensions like the former one. The residues of the operation, which remain on the hearth, are about 2 per cent. of the whole, and con-

[1] *Trans.* A. I. M. E., Vol. XXXVII.

tain 26 per cent. of arsenious oxide. The arsenic recovered in the flue, condenses in large white crystals on the roof and walls, and contains 99.8 per cent. of arsenious oxide.

The crystals are next ground in a Sturtevant buhr mill, 30-in. diameter, and fall into a hopper having a screw feed, which delivers to a second Sturtevant mill for finer grinding. From the hopper of this second mill, having also a screw feed, the powder is transferred and packed in barrels holding 50 lb. each, which are periodically jolted by power so as to settle the contents in compact form.

From the 11 tons of flue-dust put through daily, 11 to 18 tons of white arsenic are recovered monthly, being from 3.3 to 5.5 per cent. of the flue-dust. The product finds a ready market.

The workmen are protected against the poisonous effect of the fumes by the use of cotton in their noses and ears, by aspirators worn over the mouth and nose, by not working hard enough to perspire, by washing carefully after working, and by anointing the exposed portions of the face with a paint of freshly precipitated ferric hydroxide rubbed on with the fingers.

Utilization of Crude Arsenic Sulphide.—F. Howles, in *Revue Chimique*, describes a process for the lixiviation of arsenic sulphide by a solution of soda, or other alkali, whereby sulpho-arsenates of the alkaline metal will be formed. The solution containing these can than be decomposed, preferably while warm, by iron or manganese hydroxide; the sulphur of the sulpho-arsenates is thus replaced by oxygen, forming commercial arsenite in solution and a precipitate of iron or manganese sulphide, which can easily be filtered out. The arsenites can then be recovered by evaporation of the filtrate. In the practical application of this process, use can be made of steam coils for heating the reagents, and filter-presses for preparing the required hydroxides as well as for making the final separation. It is claimed that it will be possible by this method to utilize large amounts of arsenic sulphide heretofore wasted, as, for example, in the arsenic precipitate from sulphuric acid refining.

Effect of Arsenic on Brass.—E. S. Sperry has examined[1] the effect of arsenic up to 0.5 per cent. on the properties of Muntz metal (copper 60, zinc 40 per cent.). Sharper and cleaner castings were obtained from the alloys containing arsenic than from those free from that substance. The ductility of the alloys was improved by the addition of arsenic up to 0.02 per cent., probably owing to the reducing action of the latter on cuprous oxide. With higher percentages of arsenic the alloys became brittle, and cracked in the rolls.

[1] *Mech. Eng.*, 1906, XVII, 763–764.

ASBESTOS.

The asbestos mining industry of the United States continues to be insignificant. There are many occurrences of the mineral, but in general they lack the strong, long-fiber character that determines the commercial value of this mineral. The statistics of production and imports are given in the following table:

ASBESTOS STATISTICS OF THE UNITED STATES.

Year.	Production.			Imports.		
	Short Tons.	Value.	Value per Ton.	Manufactured.	Unmanufactured.	Total.
1897	840	$ 12,950	$15.42	$10,570	$264,220	$274,290
1898	885	13,425	15.17	12,899	287,636	300,535
1899	912	13,860	15.20	8,946	303,119	312,068
1900	1,100	16,500	15.00	24,155	331,796	355,951
1901	747	13,498	18.08	24,741	667,087	691,828
1902	1,010	12,400	12.27	33,313	729,421	762,734
1903 (a)	887	16,760	18.90	32,058	657,269	689,327
1904 (a)	1,480	25,740	17.40	51,290	700,572	751,862
1905	3,100	126,300	40.74	70,117	776,362	846,479
1906 (a)	1,695	28,565	16.85	96,162	1,010,453	1.106,615

(a) Statistics of the United States Geological Survey.

Two distinct minerals supply the asbestos of commerce. The true tremolitic asbestos goes under the name of amphibole asbestos. It is by far the more abundant in its occurrence. On account of its lack of tensile strength, this variety of asbestos is not adapted to spinning or weaving or to any use where strength is necessary. Therefore its chief application is in boards, lagging and various molded forms, used for insulating purposes. Chrysotile asbestos is a variety of serpentine. It is fibrous and tenacious and well adapted to spinning into rope and weaving into cloth, and is by far the more valuable of the two kinds of asbestos.

Occurrence in the United States.

Asbestos of the amphibole variety occurs in Georgia, Virginia, Pennsylvania, Texas and Massachusetts; in other States, small deposits of uncertain size are known. The Georgia deposits are rather small and of medium quality. The fibers are short and crumble away at the first attempt to twist them. The chief use of the Georgia asbestos is for various forms of insulation, for which purpose the mineral is ground, mixed with cement, and pressed or molded into suitable form. The asbestos from Floyd country, Va., is of the same general nature. The reddish-yellow,

short fibers crumble and powder very readily, having no strength. This is also true of deposits in eastern Pennsylvania and Texas, which have thus far been exploited. In Texas, in the Llano mountains, there are deposits of amphibole asbestos of some magnitude which as yet are undeveloped.

Arizona.—In Arizona, in the Grand Canyon, there occurs a deposit of chrysotile asbestos of excellent quality. This is being worked by the Hance Asbestos Company, which first entered the ranks of asbestos producers in 1905. The deposits are at the bottom of the Grand Canyon, on the north side of the Colorado River and about 70 miles north of Flagstaff, Cocinino county. The company has many mechanical difficulites with which to contend in getting its product to the market, having to transport it across the river, elevate it to the rim of the canyon, and haul it 16 miles by wagon to the nearest railroad. A projected aerial tramway across the river will reduce the labor of ferrying and hoisting the mineral and will greatly facilitate shipping.

California.—In this State asbestos claims were reported to have been located in June, 1906, on the north side of Mill Mountain near Lebec, Kern county, by R. C. Cuddy and J. A. Hazlett. It is stated that J. C. Dillon and F. P. Smith, of Nevada City, have located a deposit of asbestos on the north side of the South Yuba River, near Washington, Nevada county.

Virginia.—The American Asbestos Company produces a small amount of amphibole asbestos from its mines at Chestnut Fork Post Office, in Bedford county. The other deposits of asbestos which are known to occur within the crystalline area of the State are practically undeveloped.

Wyoming.—Deposits of chrysotile asbestos have been recently reported again in the Casper mountains. This region has long been known to contain asbestos, but so far remoteness from transportation facilities has prevented any exploitation of the deposits.

Asbestos in Foreign Countries.

Canada.—By far the largest proportion of the world's production of chrysotile asbestos comes from the Canadian deposits, from which the mineral is exported chiefly to the United States and to Europe. The principal Canadian deposits are located at Black Lake and Thetford, in Sherbrooke county, and also in and around Quebec at Templeton, Danville, and East Broughton. The only amphibole asbestos produced in Canada is mined at Elvezir, Hastings county. A new discovery of asbestos has been made on the northerly limits of Coleman township in the Cobalt district, Ont., and bordering on the township line of Bruce. The vein here is 10 in. wide.

STATISTICS OF ASBESTOS IN CANADA. (a)
(In tons of 2000 pounds.)

Year (b)	Production. (b)				Exports (c)		Imports (d)
	Asbestos.		Asbestic.				
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Value.
1897	13,202	$ 399,528	17,240	$45,840	10,969	$ 510,916	$ 19,032
1898	16,124	475,131	7,661	16,066	18,424	510,368	26,389
1899	17,790	468,635	7,746	17,214	14,520	453,176	32,607
1900	21,621	729,886	7,520	18,545	18,164	490,900	43,455
1901	32,892	1,248,645	7,325	11,114	26,715	864,573	50,829
1902	30,219	1,126,688	10,197	21,631	33,072	1,131,202	52,464
1903	31,129	915,888	10,548	13,869	30,661	955,405	75,465
1904	35,635	1,167,238	13,011	13,006	34,636	984,836	83,827
1905	50,670	1,486,359	17,594	16,900	41,127	1,311,524	116,836
1906	59,283	1,970,878	20,127	17,230	57,075	1,578,137	138,000

(a) From *Annual Reports* of the Geological Survey of Canada, and the *Statistical Year Book* of Canada. (b) Production is given for calendar year; exports and imports are for fiscal years ending June 30. (c) Mainly crude asbestos. (d) Manufactured articles entirely.

Italy.—In Italy the exploited deposits are situated in the mountains, 8000 to 10,000 ft. above Susa valley. These deposits are worked under so many difficulties that production is small. Recently a discovery of asbestos was reported at Mt. San Victore, near Lanzo, Torina. The quality is said to be nearly equal to the Canadian variety, and to be capable of easy working.

Russia.—The Ural deposits contain an immense amount of white, long-fiber, asbestos which is readily accessible and easily mined. Its quality, however, is not very high and the mineral is used almost entirely in Europe to mix with Canadian fibrous asbestos. For some purposes where an inferior grade can be utilized, the Russian brand is used without any mixture.

South Africa.—In this country, deposits of asbestos which have been exploited for some time, occur in considerable quantities in West Griqualand, Cape Colony, on the borders of the Orange River. The mineral is very fibrous and of exceptional strength. However, it contains relatively so little lime and magnesia and so much iron (40 per cent.), that it crumbles and weakens when exposed to heat, and therefore is not particularly valuable.

During 1906, new asbestos deposits of very different quality from the above variety were opened and developed in the Carolina district. The asbestos here is of exceptionally high grade, and sample shipments to Germany brought offers of £40 per ton. The deposit is pockety, but the area through which the pockets are distributed is considerable. Three companies are operating in this field. These are the Transvaal Asbestos Syndicate, now taken over by the Consolidated Gold Fields Company, the South African Mineral Option Syndicate, and the Anglo-Swiss Asbestos Company. Naturally, the importance of these deposits as competitors

with Canadian asbestos depends upon how abundant is mineral of the same quality as that already shipped. So far reports from this field indicate that the available quantity of high-grade asbestos is considerable.

Market and Prices.

During 1906 there was a considerable increase in the price of asbestos which was due in great measure to the stopping of Russian production on account of the internal troubles of that country. Part of the European demand which Russia had supplied was thrown upon Canada and an inevitable increase in price followed.

In June, 1905, the price of crude No. 1 asbestos, was $175 per ton, but rose rapidly until it was $275 per ton in June, 1906. Crude No. 2 was quoted at $125 per ton in June, 1905, and rose rapidly and steadily to $160 per ton in April, 1906, since when it has been fairly regular. The price of "paper stock" has had an irregular upward course during the last three or four years, and in July, 1906, it was $25 per ton.

It is only because of these high prices that the Arizona deposits can be exploited, as the delivery of asbestos from the Grand Canyon at a railroad in readiness for a long haul to Chicago, is much more costly than the cheap transportation charges of the Canadian asbestos down the lakes to the same city.

Use of Asbestos.

Asbestos is used largely as an insulator of heat and of electricity, and on account of its acid-resisting powers finds more or less application in chemistry. The number of increased uses to which asbestos is being applied are legion. The more recent uses are patent building lumber, known as "Transite" and various forms of molded asbestos. Transite is much used in roofing and in place of wood on warships for finishing. For decorative purposes asbestos is mixed with Portland cement and other substances and molded under pressure into a variety of forms. Asbestos rope and cloth are used in valve packings, and asbestos lagging in one form or another is applied to boilers and steam pipes to prevent wasteful radiation. It is making very marked advance in use as an electric insulator as it is more flexible than porcelain or glass, and more durable than rubber.

Literature.

The most important recent publication on asbestos is a monograph by Fritz Cirkel, entitled "Asbestos, its Occurrence, Exploitation and Uses," published by the Mines Branch, Department of the Interior, Ottawa, Canada, 1905. This gives minute particulars as to the commercial conditions of the industry, and the methods of mining and preparing the mineral for market. Some of the more important points which have been discussed

in previous volumes of THE MINERAL INDUSTRY are summarized below.

In Vol. VII there is a short summary of tests made by Paul Kersting to determine the relative merits of the Canadian and the South African varieties of fibrous asbestos. The tests showed that the Canadian asbestos was much superior in the matter of resisting heat, but the South African variety, as regards fineness, softness of fibre and suitability for spinning, was considered superior to all others, and especially suitable for acid filters.

In Vol. XII there is a brief description of some of the principal Canadian mines and the equipment by which the asbestos is cleaned and prepared for market after being mined.

In Vol. XIII an article by J. Obalski outlines very briefly the developments of the Canadian asbestos industry, describes the occurrence of the mineral in the serpentine rock, and also gives an account of separating the fiber from the rock during the process of milling.

In Vol. XIV there is a summary of tests made by Prof. Seaver of Columbia University, New York, upon the relative merits of asbestos and of magnesia boards in various forms. The test showed that asbestos board possesses better mechanical properties than the magnesia, but that the magnesia is a better electric insulator than asbestos. The asbestos, however, is a better heat-resisting medium than is the magnesia, indicating that asbestos is more advantageous than magnesia for fire protection.

The following two articles appeared in the technical press during 1906:

"Carolina (South Africa) Asbestos. The Promise of a New Industry in the Transvaal." (*So. Afr. Mines*, June 11, 1906) Names of operators and review of the work done in this district of South Africa.

"Utilidad del amianto." (*Madrid Cientifico*, July 30, 1906). An outline of the various uses to which asbestos can be put, with a note of its progress in the arts.

ASPHALTUM.

By Edward K. Judd.

The varieties of asphaltic materials regularly produced in the United States may be classified by the following scheme:

I. Natural asphalts. (*a*) Asphaltic coals. (1) Grahamite (albertite). (2) Uintaite (gilsonite). (*b*) Mineral waxes. (1) Ozocerite (elaterite). (2) Wurtzilite. (*c*) Maltha. II. Prepared asphalts. (*a*) Oil residues. (*b*) Coal tar. (*c*) Mastic. III. Bituminous Rock. (*a*) Sandstone. (*b*) Limestone. (*c*) Shale.

The asphaltic coals are brittle, black varieties of asphalt commonly occurring in veins, and are not found in any abundance in this country; the producing regions are in the Arbuckle and Ouachita mountains of Indian Territory and in the southeastern part of the Uinta Indian reservation, in Utah.

The mineral waxes are of variable composition. In general they are more elastic and less soluble than the asphaltic coals. They resemble beeswax in everything but adhesiveness, which they lack. Ozocerite is found in veins in the vicinity of Soldier Summit, Utah , on the Denver & Rio Grande Railway, and wurtzilite is mined on a considerable scale in Indian cañon, about 40 miles east of that point. Most of this product goes under the name of elaterite.

Maltha, or "brea," is a fluid, natural tar, although the name is sometimes applied to artificial oil residues. The type locality for maltha is near Carpenteria on the shore of southern California. Here a bed of Quaternary beach sand, 20 to 30 ft. thick, is impregnated with liquid bitumen, which forms about 20 per cent. of the whole. A layer of richly bituminous shale underlies the sand and is the source of the tar. Attempts to extract a pure asphalt from this sand have not been successful, but the stuff, as mined, is used extensively for road-building material.

Oil residue is now produced in the United States in greater quantity than any other kind of asphalt. Almost any petroleum, after the volatile oils and waxes have been expelled, will yield a tarry residue, but the petroleums of Texas and California are exceptionally rich in tarry matter, and are said to have an "asphalt base" in distinction to the "paraffine base" oils of the central States. In fact many of the California petroleums are so deficient in illuminating hydrocarbons that their chief usefulness consists in being converted, by a special method of heat treatment, into asphalt. Oil residue is used for thinning the more brittle natural asphalts and also by itself in the preparation of paving material.

The list of California asphalt producers is too long for insertion here.

The majority of the output is marketed through the California Asphalt Sales Agency, with offices in San Francisco, New York, and other cities. The two largest Texas producers are the Sun Company, of Philadelphia, and the Gulf Refining Company, of Pittsburg.

Coal-tar is recovered in important quantities at coal-gas works and in by-product coke ovens. At present about the only use to which coal-tar is put in this country is in the manufacture of creosote and other preservatives and of roofing and paving materials. The preparation of coal-tar dyes and drugs, which is carried on in Germany on a great scale, is not yet a well-established industry in the United States; the growing employment of the by-product coke oven will undoubtedly stimulate the manufacture of "coal-tar products," which are now imported at the rate of over $10,000,000 per year.

PRODUCTION OF ASPHALTUM AND BITUMINOUS ROCK IN THE UNITED STATES.
(Tons of 2,000 lb.)

States.	1904 (a)			1905			1906		
	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.
Bituminous Sandstone									
California	6,814	$19,264	$2.83	22,500	$63,000	$2.80	....	....	.....
Kentucky	5,670	31,185	5.50	7,530	34,885	4.63	(a)4,172	31,488	7.55
Indian Territory	5,457	12,516	2.30	(e)5,000	11,500	2.30	....	....	....
Arkansas	1,700	8,500	5.00	(e)1,500	7,500	5.00	(a) 900	5,400	6.00
Total	19,641	$71,465	$3.64	36,530	$116,885	$3.20	....	....	....
Asphaltic Limestone	1,798	$4,495	2.50	2,000	5,000	2.50	....	....	....
Asphaltum (b)									
California	53,465	626,165	11.71	57,687	545,503	9.46	....	....	....
Indian Territory (c)	*Nil.*	....	....	*Nil.*	....	....	....	....	....
Missouri	937	16,676	17.80	(e)1,000	17,500	17.50	....	....	....
Texas	25,000	250,000	10.00	113,500	851,250	7.50	....	....	....
Total Asphaltum	79,402	$892,841	$11.24	172,187	$1,414,253	$8.20	....	....	....
Gilsonite (d)									
Utah	3,528	155,080	43.93	10,516	110,144	10.47	....	....	....
Indian Territory (c)	1,000	25,000	25.00	(e)1,000	25,000	25.00	(a)2,690	18,562	6.90
Mastic:									
California	*Nil.*	....	....	*Nil.*	....	....	....	....	....
Kentucky	*Nil.*	....	....	*Nil.*	....	....	....	....	....
Pennsylvania	1,200	10,800	9.00	2,000	18,000	9.00	....	....	....

(a) From the Mineral Resources of the United States. (b) Includes hard and refined, or gum, liquid or maltha, and oil residues. (c) Includes production of Oklahoma Territory. (d) Includes gilsonite, elaterite and grahamite. (e) Estimated.

Mastic is an artificial mixture of sand or crushed rock and bitumen. A naturally asphaltic rock is generally taken as the starting material. This is crushed and is then mixed with as much other crushed rock as the tarry matter in the first rock will serve to bind. If the asphaltic rock is sandstone, limestone is generally added to it, and vice versa. Mastic is used for paving. A number of firms in Laurel county, Kentucky, used to engage in the manufacture of mastic, using a bituminous sandstone mined in the neighborhood, but the Wadsworth Stone and Paving Company, of Pittsburg, Pa., is now the only active operator.

Bituminous sandstone is quarried in California, Kentucky, Indian Territory and Arkansas. The California rock is of similar origin to the maltha-bearing sand, just described, but has been subjected to additional evaporation of its volatiles, yielding a harder rock. In the other States, the sandstone is quarried for road building, and some of the Kentucky output is made into mastic, as described. Kentucky is the only State now producing asphaltic limestone, which also goes into mastic.

Asphalt Mining in the United States.

California.—Under the stimulus of advanced prices, the production of asphalt in California during 1906 surpassed that of all previous years. Three reasons combined to increase the consumption of asphalt: the restoration of San Francisco required large amounts for construction purposes; many towns on the Pacific coast began campaigns of street improvement, calling for asphalt pavement; and in the paving industry generally throughout the United States the last vestiges of the old hostility to oil asphalts began to disappear. Every asphalt refinery in the State was driven to its utmost capacity during 1906, and many were unable to supply all their orders. The California crude oils range in their asphaltic contents from 10 to 50 per cent.; many refiners make no attempt to save the volatile components of the oil, but treat it by a process of slow evaporation so as to recover the maximum possible amount of asphalt. On the other hand, the largest refinery in the State, that of the Standard Oil Company, on San Francisco bay, after extracting the illuminating oils and lighter products, does not save the asphalt, but converts it into coke, and sells it for fuel. The California asphalts have long been used to temper the harder and more brittle products of Trinidad and Venezuela, to form a more durable material for street paving. There has been considerable prejudice against the use of oil residues alone for paving, but the California producers maintain stoutly, with a good deal of evidence to support them, that as durable a pavement can be made from their asphalt, used alone, as from any other single or compounded asphalt. It is certain that no natural asphalt contains such a high proportion of bitumen.

Indian Territory.—A new concern, the Southern Asphalt Company, of Memphis, Tenn., was recently incorporated. Its holdings consist of a 99-year lease on 350 acres of ground at Woodford, I. T., about 16 miles from Ardmore, containing extensive deposits of asphalt. The company has a refining plant on the property with a capacity of 10 tons of refined asphalt per day, but will erect additional refineries. Its product is of a soft nature, separating readily, and runs about 20 per cent. bitumen and petrolene, about equally divided.

Utah.—This State is now the leading producer of the mineral waxes in the United States and its output is competing seriously in the varnish and

insulating trades with the best product of the West Indies and Germany. The mining centers practically all lie within the boundaries of the Uinta Indian reservation, in the northeast corner of the State, but the productive strata extend over into Colorado. The General Asphalt Company of Philadelphia operates a gilsonite mine in the eastern end of the Reservation; the Raven Mining Company, of Chicago, works a deposit of elaterite in Indian cañon, a southern tributary of the Duchesne river; and ozocerite is produced by a number of operators at several points in the upper Price River valley at, and around, Soldier Summit, on the Denver & Rio Grande Railway.

The occurrence of ozocerite, and its mining development in this latter district have recently been described by J. A. Taff and C. D. Smith.[1] The chief localities are Colton, Soldier Summit and Midway. The productive rocks are shales, shaly sandstones and limestones of lower Tertiary age; in these, fissures and brecciated zones, usually vertical, have been filled with thin seams and films of ozocerite, rarely more than a few inches thick. The shattered zones occasionally reach a thickness of 4 or 5 ft., but are subject to irregular variation as to width, depth, length, and percentage of ozocerite. In general, 10 per cent. of salable product must be obtained from the mined rock, in order to be profitable. In some places, the mineral has been crushed by fault movements, and has been made brittle. Ordinarily it is only slightly friable, and cannot be molded in the hand. It cuts easily, but is not adhesive. It is generally black, but occasionally of a yellowish color. Its melting point ranges from 54 to 70 deg. C., and it is nearly or perfectly soluble in boiling ether. On heating, it behaves like paraffine.

Five mines in the district have been equipped more or less elaborately, and development has been carried as deep as 225 ft., with shafts, drifts and tunnels. Three of the mines are equipepd with plants for separating the wax from its adhering rock. Such a plant comprises boiler and engine, a crusher, steam-heated vats, and melting pans. The rock is crushed and placed in long vats with narrow bottoms, containing water maintained at boiling temperature. The melted ozocerite flows off into cooling vats, while the rock is ejected by screw conveyors from the bottoms. When cool, the ozocerite is melted again in dry pans to eliminate moisture.

Asphalt Mining in Foreign Countries.

Barbados.—A superior kind of manjak, or glance-pitch, has been mined for several years on the island of Barbados, in the West Indies, but production has lately been greatly restricted through competition of the Utah products in the American market. This manjak has a lower melting point than elaterite, and is particularly valuable on account of its jet-black

[1] U. S. Geological Survey, Bull. No. 285, pp. 369-372.

color, whereby the necessity for mixing any black pigment with it is avoided, when it is used as a basis for black varnish or japan. It used to sell at $6 to $10 above the price of domestic gilsonite, or around $35 per ton. The principal concessions are owned by R. H. Emtage of Bridgetown.

Germany.—The asphalt mines at Labsann, Alsace, have been in operation for the last 75 years, but until 11 years ago were operated in a primitive manner, with but little machinery. Ten years ago a German-English company bought the mines and leased extensive tracts of land in the Vogesen mountains, near the town of Sulz u. Wald. This new company has installed modern machinery and an electric plant. The asphalt is found in great quantities under the mountains, the veins varying from 3 to 30 ft. thick, frequently dipping at an angle of 20 to 30 deg. The asphalt is rich in quality and dark brown in color, but when exposed to the air becomes light gray. About 100 men are employed.

EXPORTS OF ASPHALT FROM TRINIDAD.
(In tons of 2240 lb.)

Year.	Pitch Lake Asphalt.		Land Asphalt.		Total.	
	To United States.	Elsewhere.	To United States.	Elsewhere.	To United States.	Elsewhere.
1900 (*a*)	70,938	51,805	34,796	448	105,734	52,253
1901	80,449	55,605	31,767	3,150	112,216	58,755
1902	104,956	34,220	25,153	290	130,109	34,510
1903 (*b*)	123,582	41,950	18,478	4,886	142,060	46,836
1904 (*c*)	63,033	48,655	22,582	660	85,615	49,315
1905 (*c*)	47,947	54,054	12,126	770	60,073	54,824
1906 (*c*)	64,198	60,968	4,850	280	69,048	61,248

Note.—A small proportion of the output undergoes a slight refining process before shipment. The above figures give the total exports, after recalculating, the "épuré and dried" to their equivalent original amounts of crude material. Shipments elsewhere than to the United States are mainly to Europe. (*a*) Epuré and dried are not reduced to original crude in 1900. (*b*) For thirteen months ending January 31, 1904. (*c*) Years ending January 31, 1905, 1906 and 1907.

Trinidad.—According to the first report of the Inspector of Mines of Trinidad, which covers the two years from Sept. 1, 1904, to Sept. 1, 1906, the American company holding the concession produced 87 per cent. of the yield, which declined from 193,215 tons in 1903 to 145,169 tons in 1904 and 117,808 tons in 1905. Measurements have shown that in 13 years the level of the lake has fallen 7.1 ft by the extraction of 1,500,000 tons of pitch. The manjak business also shows decline, the output from the two mines working in 1905 being 1077 tons, against 3023 tons in the previous year. It is anticipated that many more veins remain to be discovered, and that a substantial industry may be developed.

According to the report of the General Asphalt Company, to whose courtesy the statistics in the following table are due, the total exports of lake asphalt in the 12 months ending Jan. 31, 1907, included 93,140 long tons of crude, and 21,681 tons of *épuré* and dried (equivalent to 32,026

tons crude); and of land asphalt 5130 tons, all crude. As compared with the output of the previous year, the principal change is noted in the falling off in the production of land asphalt. There was no asphalt of either kind exported in 1906 elsewhere than to the United States and Europe.

Venezuela.—Although the ownership of Bermudez lake has been decided in the Venezuelan courts adversely to the New York & Bermudez Company (subsidiary of the General Asphalt Company), an appeal in this case, and in the one arising out of the alleged participation of the company in the Matos revolution, has been taken to the United States Department of State. In the meantime the deposit is being worked for the benefit of other American companies, the A. L. Barber Paving Company, Warner-Quinlan, and others, but they have modified somewhat their earlier policy of cutting prices to the low level made possible by their inexpensive acquisition of property and plant in Venezuela. The output of asphalt from that country in 1906 is estimated, on the best authority, at 25,325 long tons.

WORLD'S PRODUCTION OF ASPHALT AND BITUMINOUS ROCK.
(In metric tons.)

Asphalt.

Year.	Cuba.	Germany.	Hungary	Italy.(c)	Russia.	Spain.	Trinidad. (b)	United States.	Venezuela.	Total.
1900......	....	89,685	2,900	33,127	25,090	2,331	161,299	8,326	17,981	340,739
1901......	4,554	90,193	2,878	31,814	26,622	4,182	173,707	19,882	22,115	375,947
1902......	4,966	88,374	2,773	33,684	12,360	6,064	167,253	36,923	10,770	363,167
1903......	7,368	87,454	2,422	32,000	(d)	4,372	196,883	54,521	14,567	399,587
1904......	8,926	91,736	2,221	31,327	(d)	3,761	137,089	77,250	23,535	375,845
1905......	....	103,006	173	27,256	(d)	5,752	116,735	156,193	(e)25,000	(e)434,115

Bituminous Rock.

Year.	Austria.	France. (a)	Italy.	Spain.	United States.	Total.
1900.....	887	34,093	101,738	4,193	41,029	181,940
1901.....	541	29,815	104,111	3,956.	37,393	175,816
1902.....	901	34,000	64,245	6,301	35,072	140,519
1903.....	1,273	....	89,690	6,277	37,334	(f)134,574
1904.....	1,435	....	111,390	100	19,454	(f)132,379
1905.....	4,363	191,509	106,586	750	32,337	335,545

(a) France produces a large amount of bituminous shales, used for distilling oil, which is not included in these statistics. (b) Exports (crude equivalent) reported by the New Trinidad Lake Asphalt Co. (c) Including mastic and bitumen (d) Not yet reported. (e) Does not include the production of Cuba. (f) Does not include the production of France.

Note.—There is a considerable production of asphalt stone in Switzerland of which no account is taken in the above table, the Swiss Government not publishing any mineral statistics. The production of manjak in Barbados is not included in the statistics given

BARYTES.

BY EDWARD K. JUDD.

The output of crude barytes in the United States in 1906 amounted to 63,486 short tons with an estimated spot value of $252,719 as compared with 53,252 short tons, worth $196,041, produced in 1905. Missouri was the largest single contributor, its output being about equal to that of the other producing States combined, which were, in order of importance, Tennessee, Virginia, and North Carolina.

STATISTICS OF BARYTES IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.			Imports.				Consumption.	
	Short Tons	Value.		Crude.		Manufactured.			
		Per Ton	Total	Sh. Tons	Value.	Sh. Tons.	Value.	Sh. Tons.	Value.
1897........	26,430	$4.00	$105,720	502	$ 579	1,300	$13,822	28,232	$120,121
1898........	28,247	4.00	112,988	1,022	2,678	687	8,678	29,956	124,344
1899........	32,636	4.20	137,071	1,739	5,488	2,111	22,919	36,486	165,478
1900........	41,466	3.90	161,717	2,568	8,301	2,454	24,160	46,488	194,178
1901........	49,070	3.22	157,844	3,150	12,380	2,454	27,062	54,674	197,286
1902........	58,149	3.21	186,713	3,929	14,322	3,908	37,389	65,986	238,424
1903........	a50,397	3.02	152,150	6,344	22,777	5,716	48,726	62,457	223,653
1904........	a65,727	2.66	174,958	6,689	27,463	5,920	48,658	78,336	251,070
1905........	53,252	3.68	196,041	7,879	36,796	4,827	39,803	65,418	272,649
1906........	63,486	3.98	252,719	9,189	27,584	4,808	37,296	77,483	317,599

(a) Statistics of the U. S. Geological Survey.

BARYTES MINING IN THE UNITED STATES.

Barytes is found in the United States in a variety of associations. In Missouri and Tennessee it occurs in lumps scattered through beds of clay and is mined by the simplest surface workings. In certain parts of Missouri, and in Illinois and Kentucky it is found as forming, often with fluorspar, the gangue mineral in lead- and zinc-sulphide veins. In Virginia and North Carolina, barytes is found, by itself, in veins and pockets in unaltered rock. Limestone appears to be the universal home of barytes; in Kentucky it is the Chazy, in Missouri, the Potosi, and in the Appalachian district, the Knox dolomite to which the productive deposits of barytes are confined.

Missouri.—Washington county yields about three-quarters of the output of this State, and the neighboring counties, to the southwest of St. Louis, supply the remainder. The barytes, or "tiff," is collected from the

surface, or in shallow workings, by the farming population during off times, carried to the nearest railroad point and shipped to St. Louis. The two consumers in that city are the Nulsen, Klein & Krausse Manufacturing Company, and the Fink Mining and Milling Company. A comparatively new plant at Mineral Point, Washington county, owned by the Point Mining and Milling Company, now affords a more convenient market for part of the output. Nulsen, Klein & Krausse's plant at Quincy, Ill., which used to bleach and grind barytes from the Missouri district, has been dismantled on account of the additional freight charge to that point. The mining of barytes in Washington county now surpasses in importance the mining of lead ore, which industry attained its earliest importance in this vicinity. A considerable amount of the crude barytes treated at St. Louis now is brought from Tennessee.

Tennessee.—The Tennessee barytes is found irregularly scattered through the residual clay left by decomposition of the Knox dolomite, which, in its upper portion, is known to carry this mineral throughout much of its extent. The productive areas, as now developed, center around Sweetwater, on the Knoxville division of the Southern Railway, in southeastern Tennessee, and around Bristol, in the northeastern corner. The richest deposits in the vicinity of Sweetwater are usually found on the summits of knobs and ridges, and consist of nothing but thick beds of stiff, red clay, carrying lumps and stringers of barytes, which follow no apparent rule as to position. The lumps vary from small nuggets to pieces of nearly one ton in weight, and extend from the surface down to the underlying dolomite. Bed rock has thus far been reached at only one point, where it was struck at a depth of 40 ft. below the summit of a gentle hill. The mineral in some lumps has a translucent, coarsely crystalline structure and in others a chalky appearance. In addition to a closely adhering coating of limonite on the outside, each lump is seamed with minute fissures, into which iron stains have penetrated. This objectionable feature of the mineral is more pronounced in some deposits than in others, and its presence or absence is the predominating factor in determining the value. The factor next in importance is softness, and in this the Tennessee mineral is about on equality with the notably soft Missouri mineral. In some deposits the barytes is thickly incrusted with fluorspar, which is not a very troublesome element, and with brown hematite, which does give difficulty in subsequent treatment. Fragments of chert are a constant ingredient of the clay beds.

It has been suggested that, if the clay covering were removed and attention directed to the dolomite beneath, workable veins of pure white barytes, not contaminated by brown iron stains, would be found. While this view has never been disproved, it appears to be not well founded. As the clay has certainly been derived from the dolomite—traces of the

original stratification are frequently visible in it—a certain amount of concentration of the resistant barytes has occurred, so that the clay probably carries a greater proportion of barytes than the original rock. Then the lateral distribution of the barytes lumps would argue in favor of a similar distribution in the rock, since this region is far removed from that affected by the glacial period. Finally, mining in clay is so much cheaper than work on the same scale in rock, that no attention need be wasted on this question as long as the clay deposits remain, and, in spite of a lot of indiscriminate digging, these have by no means yet been exhausted.

The working methods employed at the deposits are simple in the extreme. A cut, wide enough for a dump-cart, is driven into the slope of the hill until a satisfactorily rich spot is found. The clay banks are then dug down with pick and shovel, the lumps of barytes picked out and the refuse carted away and dumped. The lumps are spread on the ground or on a board platform, and allowed to remain until sun and rain have loosened the adhering clay, when they are scratched over to remove most of the dirt, forked into wagons, and hauled to the railroad, or cleaning or grinding mill. In this way, about half the output is lost, for the soft barytes easily breaks away into fine pieces which go through the fork and stay on the dump. A recent run of dump material through a five-compartment jig revealed about 30 per cent. of barytes recoverable with this simple treatment. Water is usually scarce in the immediate neighborhood of the deposits; otherwise log-washers and simple jigs would probably be generally employed at the mines.

The consumers of crude barytes in Tennessee are: William D. Gilman Company, Sweetwater; Commercial Mining and Milling Company, Knoxville; John T. Williams & Son, Bristol. Part of the last concern's supply of crude comes from Virginia.

Virginia—The valley of the Clinch river, flowing through Russell and Tazewell counties in the extreme southwest of the State, is the most productive barytes area of Virginia. The leaders of the industry in this section are the Clinch Valley Barytes Company, at Honaker, and the Pittsburg Baryta and Milling Corporation, at Richlands. In addition to these producers and the Bennett mine near Toshes, in Pittsylvania county, small shipments were made in 1906 from mines reopened in Smyth, Bedford and Nelson counties. Barytes mining on a small scale was recently resumed to the west of Marion in Smyth county. Work in connection with the opening of a barytes property in Bedford county, about three miles west of Thaxton station, has been in progress for several months past. In Louisa county, barytes mining was again begun about three miles east of Lindsay, a station on the Chesapeake & Ohio Railway. At Lynchburg, the Nulsen, Klein & Krausse Manufacturing Company, of St. Louis, maintains a well equipped plant for bleaching and grinding,

which is supplied by railroad shipments from a few of its own mines in Virginia, and from other producers in adjoining States. There are no productive mines in the vicinity of Lynchburg.

North Carolina.—The principal producer, and, at present, the only consumer of barytes in North Carolina, is the Carolina Barytes Company, at Stackhouse. The Hot Springs Manufacturing Company has erected a grinding plant at Hot Springs, but it had not gone into service at the end of the year. Part of the crude barytes treated at Lynchburg, Va., is mined at Hot Springs, while some more comes from Sandy. North Carolina possesses numerous deposits of barytes which will become available whenever additional railroads shall have been built nearer to them.

Other States.—The discovery of barytes 4½ miles west of Platteville, Wis., is reported. Although the mineral is of good quality, the vein has a maximum width of only 18 in. The Monitor Mineral Company, operating fluorspar mines in Mercer county, Kentucky, has erected a mill for the separation of barytes from its mixed ores, together with a plant for grinding the separated product and for making barium salts.

Commercial Conditions.

Imports.—A considerable part of the consumption of barytes and its products in this country is supplied by importation, mainly from Germany; some crude barytes is sent by the Ainslie Mining and Railway Company, whose mine is on Cape Breton Island, to its plant in Brooklyn, N. Y. The natural sulphate is imported both in its crude condition, and as a bleached and pulverized material. Witherite, the natural carbonate, is also imported. Besides these natural products, the chief artificial compounds of barium brought into this country are the binoxide, which is used in the preparation of hydrogen peroxide and in beet-sugar refining, the artificial sulphate or "blanc fixe," and the chloride. Certain indeterminate amounts of precipitated carbonate, nitrate and hydroxide also are brought in under the general name of chemical salts.

Market and Prices.—The barytes industry during 1906 developed a marked shortage in the supply of crude material, particularly of that suitable for the manufacture of the highest grade of finished product. The exceptionally heavy demand for ground barytes put a severe strain upon the grinding plants, but these, in turn, were limited in activity by the insufficient deliveries of crude. There was probably only one mill in the country that ran at its full capacity throughout the year. The irregularity in the output of crude barytes is mainly due to the dispersion of the mining interests. Almost without exception, barytes mining operations are conducted on a small scale by individuals in widely scattered localities; this involves long wagon hauls, subject to perennial obstacles, with usually the added trouble and expense of a railroad shipment to the

desired grinding plant. The low price of the crude material, as well as the uncertain limits of any given deposit, do not warrant an expensive mining outfit or a costly transport system, so that the whole mining side of the industry is characterized by a certain unavoidable fortuitousness. In but few cases does the operator of a grinding plant depend wholly, or even extensively, upon the product of his own mines. A prime cause of the shortage toward the end of the year 1906 was the rainy weather that prevailed in all the producing districts. The autumn is usually the most active season of the year among barytes miners, when supplies for the winter's milling are accumulated.

On account of the diverse haulage conditions, just described, it is difficult to fix a value for the crude product at the mine; the following quotations must, therefore, be understood to represent the average prices paid per short ton upon delivery at the grinding mills, which include thereby a varying charge for wagon or railroad haul. In the Missouri field the price ranged \$4 @ \$4.10 during 1906, but contracts for 1907 delivery were made at \$5 per ton. In Eastern Tennessee, \$3.50 @ \$4 was the customary price during 1906; an advance of about \$1 per ton was expected on contracts covering 1907, owing to the prevailing shortness. Some of the Virginia grinders state that they paid as much as \$5 per ton for their 1906 receipts. The cost of the North Carolina output, delivered at mill, ranged around \$3 per ton during the year. A general advance in the price of crude throughout the country is warranted by the advancing prices of the finished materials, as well as by the higher rate of mine wages now prevalent.

The market for finished barytes was exceedingly active throughout 1906, owing to the growing demand of the paint trade and of the rubber manufacturers, culminating, toward the end of the year, in a decided rise of prices. The following quotations represent fairly the average of the year's prices in the New York market: Water-floated, \$18 @ \$19 per short ton; No. 1 white, \$17.50 @ \$18.50; off-color goods, \$15 @ \$17 per short ton. Imported, German, finished products were sold at about the same prices. Whatever superiority is shown by the German material arises solely from the superior quality of the crude barytes from which it is prepared. The German ores are noted for softness and whiteness, so that but little attention is paid to the process of water-floating them. German No. 1, dry-ground, product compares well therefore with domestic water-floated.

Technology of Barytes.

Treatment in the United States.—The largest consumers of crude barytes in the United States are the grinding mills, wherein the crude mineral is pulverized to extreme fineness to suit the requirements of the paint trade. This is universally accomplished in buhr-mills. Nearly all of the output of crude requires a bleaching process before the desired degree of whiteness

can be secured in the finished product; the bleaching practice varies in the different regions, according to the amount and nature of the impurities in the mineral. Water-floating, whereby a particularly finely pulverized product is secured, is practiced only in Missouri. One plant in the United States is equipped for cleaning barytes mechanically, i. e., by jigging; and at least three plants are equipped with roasters and other apparatus for preparing chemical salts of barium. A number of concerns consume crude barytes in the manufacture of lithophone. In the following table are given the names and addresses of all the consumers of crude barytes in the United States known to us, together with a note of the operation conducted by each:

Name.	Address.	Operation.
Nulsen, Klein & Krausse Mfg. Co.	St. Louis, Mo.	Bleach, grind, float.
Finck Mining and Milling Co.	St. Louis, Mo.	Bleach, grind.
Point Mining and Milling Co.	Mineral Point, Mo.	Bleach, grind, float.
Nulsen, Klein & Krausse Mfg. Co.	Lynchburg, Va.	Bleach, grind.
Pittsburg Baryta and Milling Corp.	Richlands, Va.	Bleach, grind.
Clinch Valley Baryta Co.	Honaker, Va.	Beach, grind.
Commercial Mining and Milling Co.	Knoxville, Tenn.	Bleach, grind.
William D. Gilman Co.	Sweetwater, Tenn.	Crush, jig, roast, salts.
John T. Williams & Son.	Bristol, Tenn.	Bleach, grind, roast, salts.
Carolina Barytes Co.	Stackhouse, N. C.	Bleach, grind.
Hot Springs Mfg. Co.	Hot Springs, N. C.	Bleach, grind.
Delaware Barytes and Chemical Co.	Dover, Del.	Bleach, grind, salts.
Ainslie Mining and Railway Co.	Brooklyn, N. Y.	Bleach, grind.
Hammill & Gillespie	Stamford, Conn.	Bleach, grind.
Cawley, Clark & Co.	Newark, N. J.	Lithophone.
N. Z. Graves & Co.	Philadelphia.	Lithophone.
Harrison Bros. & Co.	Philadelphia.	Lithophone.
Excelsior Mfg. Co.	Newport, Del.	Lithophone.
Grasselli Chemical Co.	Grasselli, N. J.	Lithophone..
New Jersey Zinc Co.	Palmerton, Pa.	Lithophone.
Cheeseman Chemical Co.	Scranton, Pa.	Lithophone.
Becton Chemical Co.	Becton, N. J.	Lithophone.

The Ainslie company imports its crude mineral from its own mines on Cape Breton Island, but its supply from this source is not equal to the capacity of the mill. Hammill & Gillespie treat only imported crude, but barytes is merely one of their products. Of the lithophone manufacturers it may be said that all but two are now using imported mineral only, although in the past nearly all have used a certain amount of domestic crude.

Barytes Cleaning.—Mechanical processes for the elimination of the usual impurities in crude barytes have undoubtedly received a considerable amount of investigation, and the almost unanimous disfavor into which they have fallen has probably resulted after due consideration. At present only one plant in the country is using a mechanical process, and this operation appears to be attended with considerable success. A mechanical process, as contrasted with the prevalent chemical method of cleaning barytes, appears to have certain advantages in cheapness and convenience of operation which strongly suggest the advisability of further

investigation along this line. The two most objectionable impurities in a ground barytes intended for the paint trade are lime and iron; the former on account of its disintegrating reaction in a paint whose other ingredients may have some acidity, and on account of its light weight, and the latter because of its influence on the whiteness of the pigment. Calcite and gypsum are almost invariable associates of barytes in nature, and both can be completely and readily separated, by a jigging process, from barytes by virtue of the wide disparity of their specific gravities. On the other hand, gypsum is not affected at all by the acid bath, which will be described, and calcite is dissolved only at the expense of much acid. Iron occurs with barytes, in varying amounts according to locality, generally in the form of rinds and scales of brown hematite and as minute flakes and filaments of limonite. The latter forms of iron are removed, without much difficulty, by bleaching with sulphuric acid, provided that the crude is sufficiently well granulated to give the acid access to the iron-bearing crevices. In the case of a soft ore, however, this elimination seems to be equally possible by means of a jigging process; the ore can be crushed sufficiently fine to expose most of the iron-coated surfaces, when the motion removes the film and discharges it. There is no great loss of barytes in this method, for its weight, even in a highly comminuted condition, carries it into the hutch, whence it can be recovered. The more massive forms of hematitic iron give great difficulty by either method. Crude barytes carrying this impurity is practically debarred from the acid treatment on account of the resistance of the iron to the dilute acid. The jig is not much more successful, for the pieces of hematite, even when thoroughly detached from the crushed barytes, do not pass off from the jig, owing to their weight. Here seems to be an opening for the magnetic separator; this machine has, in fact, received some casual experimentation, with no result thus far.

At the plant of the William D. Gilman Company, Sweetwater, Tenn., a jigging process is conducted as follows: The impurities in the ore coming to this plant are clay, fluorspar, chert, brown hematite and limonite. The ore is unloaded from wagons to a platform and then passes through a Carterville jaw-crusher, where water is added to insure smooth flowing of the still sticky material. From the crusher, the stream falls through a pair of geared rolls, 15-in. face, and is then elevated to the head end of a jig. The jig used here possesses no peculiar features. Its five cells are each $2\frac{1}{2}$ ft. wide and 3 ft. long; the screens are of 6-mesh, iron-wire cloth, and are allowed to bed themselves. The coarse concentrates are drawn off through side discharges; one-half or more of the clean barytes goes into the hutch, owing to its softness, and is there recovered. In the tailings, fluorspar and chert pass off without the least difficulty, but the hard rinds of brown hematite, having a high density, remain obstinately in the concentrate.

Magnetic separation would appear to be the method of removing the hematite, since it readily breaks loose from the barytes, and a moderate heating with a suitable reducing agent would both decrepitate the barytes and render the iron oxide magnetic. The brown stains of limonite are nearly all removed by the jigging, which rubs them off from their holding places; the completeness of their removal can be determined by the fineness of the crushing. The concentrates from this jig are shipped to other manufacturers for further treatment.

Barytes Bleaching.—This process consists, briefly, in treating the crude ore, crushed to a size dependent upon the intimacy of the iron impurity, in a hot bath of dilute sulphuric acid. A lead-lined, wooden vat, round or rectangular, is employed. The only noteworthy variations in the practice are seen in the size to which the ore is reduced before bleaching, and in the method of agitating the charge. In the South, the crude is crushed to about 0.5 in. size before going into the vats; in Missouri the general practice is to reduce the ore nearly or quite to its ultimate fineness before bleaching. The latter method insures a more nearly perfect solution of the iron, but involves additional mechanical difficulties in the handling of finely ground material. For agitating the charged vats, steam is the common agent, led in through a punctured, lead pipe coiled on the bottom. It has been found advantageous in some plants to employ compressed air to replace part of the steam, thereby avoiding too great dilution of the bath, but in not so large part as to reduce the temperature.

At the Lynchburg, Va., works, belonging to Nulsen, Klein & Krausse, the ore is crushed to less than 1 in. and is treated with sulphuric acid and steam for 12 hours. The solution of iron sulphate is run to waste, while the cleaned ore is washed with water and then dried in a mechanical oven. With some of the more impure ore received at this plant, the sulphate solution, containing still some available acid, is used as a preliminary bath for the next succeeding vat-full of ore, and additional fresh acid is put on, after the first bath has been drawn off. With the particular ore now most in evidence at this plant, which is a hard, grayish, finely crystallized barytes, no advantage is gained by treating the charge for more than 12 hours. An ordinary charge weighs about 10 tons, and 15 to 20 gal. of commercial sulphuric acid are required, diluted with sufficient water to cover the ore. At the Ainslie company's Brooklyn plant, a log washer is used to remove the acid from the cleaned ore; ordinarily the charge is washed in the vat.

Barytes Grinding.—Whether required for the paint or the rubber trade, barytes must be finely ground, and for this purpose no machine has yet displaced the ancient buhr-stone. Disintegrators have been tried, but their best product, after repeated passes, falls far short of the required fineness. At the Lynchburg plant, the ore, after drying, is passed through

as many as seven buhr mills, one after the other. After the third mill, only an expert can distinguish the successive degrees of fineness. After the last mill is passed, the powder is bolted through cloth and shipped in bags and barrels. A sugar barrel holds 700 to 800 ℔., while as much as 1100 ℔. has been put into one sack.

The ores of the various districts vary greatly in softness and thus offer greater or less difficulty in grinding. The Missouri ores are probably the softest in this country, and are not much inferior to those of Germany. The Tennessee ores are not much harder, but most of the Virginia product is difficult to grind.

Barytes Roasting.—As barium carbonate, witherite, is almost an unknown mineral in this country, the manufacturer of barium salts is deprived of this easily soluble starting material, except by importing it, and has to start with the sulphate. The first step in the getting of barytes into solution consists in roasting it with carbon, which reduces it to sulphide; this is then easily dissolved by water, and can be converted into any desired salt or precipitate. In ore intended for this process, a higher proportion of impurity can be tolerated, since the barium reduction product alone is soluble in water. The roasting process at the William D. Gilman Company's plant is conducted in the following manner: The crude ore is mixed, on the receiving platform, with coal, in the proportion of four parts (by weight) of barytes to one of coal. The coal for this purpose should be low in ash, and not too volatile; the former qualification avoids the subsequent treatment of a mass of inert, though harmless, material, and the latter insures a more gradual and thorough reduction. The coal used here comes from La Follette, in the Jellico district of Tennessee, is guaranteed to have not over 3 per cent. ash, and costs $1.75 per ton delivered at Sweetwater. It is, if anything, rather too volatile, but this fault can be overcome by mixing it with hard coal, or coke.

The mixture is then put through a jaw-crusher, followed by a Sturtevant emery mill, and then elevated to a hopper over the center of a cylindrical, horizontal, revolving furnace, driven, through cog-wheel and pinion, by a small independent engine. The furnace has an outside diameter of 12 ft., and a length of 18 ft., is supported on two tires resting on rollers, and is lined with two courses of fire-brick. A fire-box is at one end, and at the other the heat passes under boilers which supply power for the plant. This heat is amply sufficient to provide 125 h.p. The charging door is on the side of the cylinder, at its middle. A space is left underneath to receive the discharge.

The roaster is charged with 3.3 tons of mixture, every three hours, or eight times a day. In addition to the coal in the mixture, only 1.3 tons of coal are required per 24 hours, in the exterior fire-box. Carbonate begins to form as soon as the first sulphide appears, so that roasting must not

be unduly protracted. The clinker from this furnace contains 65 to 68 per cent. of sulphide, soluble in water, and is sold on guarantee of over 60 per cent. solubility. Additional equipment is now being installed to leach this clinker, all of which has heretofore been shipped as made, and to manufacture barium salts. The carbonate in the furnace clinker, though insoluble in water, is easily dissolved in nitric or hydrochloric acid, and the corresponding salt recovered.

Bibliography.

The usual statistics of barytes may be found in every volume of The Mineral Industry, together with an account of the distribution of the industry in various States. In addition to these features various special articles will be found as follows: In Vol. II, brief accounts of the history of the barytes mining industry, the process of preparing barytes for use as a pigment, and a summary of prices of barytes. In Vol. VIII, an account of the process of making caustic baryta, also a procedure for the determination of barium in barytes. In Vol. X, an account of the technology of barytes, including a description of the process of manufacturing barium oxide, and a review of analytical chemistry applying to the determination of barium. In Vol. XIII, a discussion of the process of bleaching barytes and a history of the barytes industry by William D. Gilman, from 1887 to 1894.

BAUXITE.

By Edward K. Judd.

The output of bauxite in the United States in 1906 is estimated at 78,331 long tons, with a value of $352,490, as compared with 47,991 long tons, worth $203,960, in 1905. The productive areas of the United States are confined to Arkansas, Georgia and Alabama. The Aluminum Company of America is now the only producer in Arkansas, having bought the American Bauxite Company's interests within that State in 1906; it has properties also in the Georgia-Alabama district which it acquired two years ago from the General Bauxite Company. These southern deposits are now being drawn upon to afford additional raw material for aluminum reduction, this industry having, until lately, been sufficiently supplied by the output of the Arkansas field alone. In the Georgia-Alabama bauxite district the active producers in 1906 were: The Aluminum Company of America, Pittsburg, Pa.; Republic Mining and Milling Company, Philadelphia, Pa.; John H. Hawkins, Rome, Ga.; National Bauxite Company,

STATISTICS OF BAUXITE IN THE UNITED STATES.

Year.	Production.			Imports.		Exports.		Consumption.	
	Long Tons.	Value.	Per Ton.	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1896	17,096	$42,740	$2.50	2,119	$10,477	....	....	19,215	$53,217
1897	20,590	51,475	2.50	2,645	10,515	2,537	$5,074	20,708	56,916
1898	26,791	66,978	2.50	1,201	4,238	1,000	2,000	26,992	69,216
1899	36,813	101,235	2.75	6,666	23,768	2,030	4,567	41,449	12,436
1900	23,445	85,922	3.66	8,656	32,968	1,000	3,000	31,101	115,889
1901	(a)18,905	97,914	4.23	18,313	66,107	1,000	3,000	36,218	144,021
1902	29,222	128,206	4.39	15,790	54,410	*Nil.*	....	43,112	175,875
1903	(a)48,087	171,306	3.56	14,889	49,684	*Nil.*	....	62,976	220,990
1904	48,012	166,121	3.46	15,475	49,577	*Nil.*	....	63,487	215,698
1905	47,991	203,960	4.25	11,726	46,517	*Nil.*	....	59,717	250,477
1906	78,331	352,490	(e)4.50	17,809	63,221	*Nil.*	....	96,140	415,711

(a) Statistics of the United States Geological Survey. (e) Estimated.

PRODUCTION OF BAUXITE IN THE UNITED STATES.
(In tons of 2240 lb.)

State	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905	1906
Alabama	9,796	13,083	13,848	14,144	650	(a) 18,038	5,577	(a) 22,374	7,087	17,094	27,131
Georgia	7,300	7,507	12,943	19,619	20,715		19,000		16,909		
Arkansas	....	....	....	3,050	2,080	867	4,645	25,713	24,016	30,897	51,200
Total	17,096	20,590	26,791	36,813	23,445	18,905	29,222	48,087	48,012	47,991	78,331

(a) Statistics of the United States Geological Survey.

Rome, Ga. The last mentioned company was organized in 1906 by some of the former members of the General Bauxite Company, the concern whose property had previously been sold to the Aluminum Company of America. The location of these several companies' operations will be noted in a subsequent paragraph.

Imports.—About one-quarter of the consumption of bauxite in the United States is supplied by imports, all of which now come from France. The French mineral ranges higher in iron than that mined in this country, but it is so cheaply mined and put on board ship at French ports, and is carried (as ballast) at such favorable rates, that it is profitable to import it, particularly at New England and Eastern ports, to which the railroad haul from the South and West is long and expensive. The duty is $1 per long ton. The principal use to which imported bauxite is put is in the preparation of refined alumina for subsequent use in aluminum reduction, but about one-quarter of it is used for making aluminum salts.

The Bauxite Industry of the Southern States.

Occurrence.—Unlike the Arkansas bauxites, which were plainly derived by the decomposition, in place, of highly aluminous igneous rocks, the Southern bauxites offer no unmistakable clue as to their origin. The ore-bodies are found in the shape of irregular funnels, tapering from a diameter of 200 to 500 ft. at surface to less than 100 ft. at a depth of, say, 75 ft. which is as far as any mine has yet been operated. The inclosing walls of these bodies consist entirely of boulders and gravel, while an exceedingly fine and unctuous clay, beautifully colored in red and blue, accompanies the bauxite, forming streaks throughout its mass, and often extending into the surrounding gravel. The mineral itself is markedly pisolitic, the spherules varying from a minute size to that of marbles. Occasionally the globular concretions constitute almost the entire substance of the ore, but more often they are imbedded in a much larger proportion of fine and granular matter.

Distribution.—At the present moment active bauxite mining in the Southern district, which extends along the Coosa valley from Adairsville Ga., to Jacksonville, Ala., is confined to three centers, viz., in the vicinity of Hermitage, Ga., at the northeast extremity, and of Cave Spring, Ga., and Rock Run, Ala., toward the southwest.

Around Hermitage only four mines are now active. The Julia, five miles east, and the Ward mine, two miles south of Hermitage, are worked by the Republic Mining and Milling Company, of Philadelphia. The Aluminum Company of America is taking mineral from the South Watters, one of a group of several mines three miles south of Hermitage, which it recently acquired from the General Bauxite Company; and J. H. Hawkins, of Rome, Ga., maintains a steady production from his mine, two miles south of the Julia. The Republic company operates a drier at the Julia

mine and another at Hermitage, hauling crude mineral to the latter from the Ward and one or two other mines whose stocks of mined mineral are not yet depleted. Mr. Hawkins also has a drier at his mine.

At Cave Spring, Ga., the National Bauxite Company, which has lately been organized by the leading interests in the former General Bauxite Company, is operating the Hampton mine and ships its mineral for drying to its excellent plant at Rome. This same company is also developing mines at East Lake and Sherman Heights, near Chattanooga. Both the Aluminum and the Republic companies own other bauxite-bearing land at Cave Spring, but neither is mining extensively. The Aluminum company has a drier at this place.

At Rock Run, Ala., the Republic Mining and Milling Company owns a group of four mines, of which only one, the Dykes, is now productive; the others, however, are not all exhausted. The same concern has a drier in the vicinity. Another small mine now operating is the Bigelow, 12 miles south of Rome, formerly owned by Harrison Bros. & Co., of Philadelphia, but now by the National company.

This list accounts for all driers and all the now active mines in the South, but the catalog of the district's bauxite resources does not end here. Exploration for new deposits is constantly in progress and as the deposits are easily found, quickly developed and often speedily exhausted, the number and location of producing mines is variable and is influenced mainly by the market demand for bauxite. It may safely be stated that in no case has one of the deposits been completely exhausted; for, owing to the system of mining, hereafter described, the mines reach their economical limit long before all mineral, even that of shipping grade, is removed. Then there are numerous instances of mines with abundant reserves abandoned for reasons not apparent to an outside observer.

Mining.—The customary method of winning bauxite, as exemplified in the South Watters and the Julia mines, may be briefly described. The South Watters is singularly fortunate in being opened on a hill top and is, in fact, the only important mine in the whole district so situated. A narrow cut was dug into the hillside, 30 ft. below the top of the deposit, a trestle extended outwardly from it to facilitate the handling of both waste and bauxite, and the breaking down of the bauxite began. Pick and shovel were the only tools employed; blasting, although it would have greatly assisted in loosening the ground, was out of the question on account of the intimate association of bauxite, clay, gravel and soil.

The pit now presents a series of benches, the bottom being a level floor over which movable tracks radiate to the various points on the wall at which bauxite predominates over clay. Everything taken down—bauxite, clay, gravel, and even overburden—is loaded into the same wooden cars and pushed out over the trestle. The bauxite is deposited at a point con-

venient for wagons, but the rest is conveyed as far as practicable. The chief difficulty in bauxite mining is to keep the bauxite uncontaminated with worthless material. It is usually accomplished by removing waste alone until a good face of bauxite is uncovered, which is then attacked. This process occasionally renders it permissible in a richer mine to employ a blast in the bauxite, thus reducing the expenditure of pick labor. In this mine it is found necessary to move about six tons of material to recover one ton of bauxite. This bauxite is usually sold on guarantee of 52 per cent. alumina, and not more than 2 per cent. ferric oxide, nor over 5 per cent. silica.

The adjoining North Watters mine, now nearly exhausted, was not amenable to this same treatment, since it was some distance from a convenient hillside. A tunnel was therefore driven to tap the deposit at a depth of 65 ft., and through this opening, besides securing drainage, all the ore was extracted by the glory-hole method. This body presented an unusually clean mass of mineral, so that this process, whose principal drawback is the impossibility of keeping things separate, could be applied. In this mine, six tons of material moved were found to yield about five tons of bauxite.

The Julia mine more truly displays the usual adverse conditions of bauxite mining in the district. The ground is low and level so that not only are natural drainage and level car tracks not obtainable, but the cavity made by taking out mineral acts as a collector for all the surface water within reach. At the Julia an inclined track extends with a 30-deg. slope to the level bottom of the pit, and there branches. A steam hoist at the top pulls the cars from the bottom and they then proceed by trestle to the drier. A small plunger steam pump at the bottom is required to run nearly constantly to keep pace with the water. In the hope of restricting the inflow of water, a trench was once dug around the entire circumference of the pit, draining at the lowest point, but the results were not the ones anticipated, for some of the water, taking an underground course to the bottom of the excavation, made a slippery plane down which most of the adjacent earth slid into the pit. The most successful method for retaining the loose banks is to leave a skin of bauxite, wherever this can be done, but even then so large a part of the circumference is devoid of mineral, and composed entirely of clay and gravel, that a heavy rain will wash in enough mud to retard mining for several days.

Drying.—As the mine output has, in nearly all cases, to be carted several miles to railroad, and as a long train haul is then required to get the bauxite to market, it becomes a matter of economy to remove as much water as possible before shipping the bauxite. The Julia drier may be described as typical of the plants commonly employed. Some of the Julia bauxite

is obtained in hard lumps; these are shipped, with no preliminary treatment, to Niagara Falls to be used in the manufacture of alundum.

As the crude bauxite arrives at the drying plant, some of that which is most contaminated by clay is first put through an inclined double-log washer. The mineral is fed into the low end of the trough, and a copious stream of water falls into the high end. The logs work the bauxite up the incline, and discharge it, cleaned, at the upper end, while the clay goes off at the bottom, through a launder. A number of shallow settling boxes are provided at points along this launder, and a considerable saving of fine bauxite is effected in them.

This washed bauxite, together with the higher-grade crude, is then put through the drier. This apparatus is a simple sheet-iron cylinder, of 3-ft. diameter and 26 ft. length, inclined at about 10 deg. from horizontal, and rotated slowly by a sprocket chain. At the low end are a firebox, and a bucket elevator for raising the dried bauxite to the shipping bin; the damp bauxite is shoveled into the upper end, whence the smoke and steam pass up a chimney. The interior of the cylinder is not lined, but is provided with three rows of buckets precisely like those on a bucket elevator. The rows are set staggering, so that a forward and downward motion is given to the mineral as the buckets raise it and then let it fall through the current of hot air.

The drier has a capacity of 25 to 30 tons in 12 hours. Wood is the most satisfactory fuel for bauxite drying, not because it is cheap, but because the smoke from soft coal blackens the mineral. With the present inadequate supply of labor for obtaining firewood, this plant has actually had to haul in coke to burn in its drier. The fireman maintains a thick bed of burning coke and occasionally throws in a heavy chunk of wood to provide a flame.

This mine is now shipping one carload of bauxite per day, mainly to chemical manufacturers, with some to basic open-hearth steel plants, and to Niagara Falls, as noted. The highest-grade, or so-called "Alum" bauxite ranges between 56 and 58 per cent. Al_2O_3 and around 8 per cent. SiO_2. The No. 2 bauxite averages 50 per cent. Al_2O_3 and 12 per cent. SiO_2. All the bauxite from this mine carries less than 1 per cent. iron, and some of the hard lump is said not to exceed 2 per cent. of silica.

The mine is five miles from the nearest railroad point, Halls, on the Western & Atlantic Railroad, and the bauxite is hauled out in wagons, eight of them making two trips each, per day, over exceedingly difficult roads. The necessity for this kind of transport seems inherent in, and must be a heavy burden on the bauxite industry. The difficulty has been solved at Hermitage, in particularly favorable circumstances, by building a narrow-gage track, down which a couple of loaded cars run by gravity 1½ miles to Shannon, on the Southern Railway, whence the empties are hauled back by mule. As a rule, however, the mines are too widely scat-

tered, too remote from the railroads, and of too doubtful endurance to warrant any elaborate arrangement for transporting the bauxite.

Bibliography.

The customary statistics of production and notes on occurrences of bauxite are to be found in every volume of The Mineral Industry and these are supplemented by the following special articles: In Vol. II, a 12-page article by Henry McCalley, describing the characteristics of bauxite and its physical occurrences in Europe and the United States; many analyses of bauxite from various regions are given. There is also a short discussion of methods of mining as carried out in America, with some remarks on the cost of production and uses of bauxite. In Vol. V, J. Sutherland describes the manufacture of alumina from bauxite as carried out at Larne, near Belfast, Ireland. In Vol. VII there are some notes on the occurrence of bauxite in foreign countries and the technology of bauxite, together with procedures for its analysis, are described. In Vol. X there is a brief description of the limits, general geology and modes of occurrence of the bauxite deposits of the southern Appalachian region. In Vol. XIV there is an article by A. J. Aubrey, dealing with the refractory uses of bauxite.

BISMUTH.

By W. R. Ingalls.

The United States consumes considerable bismuth, but produces only a little, and so far none in metallic form. Such ore as is marketed is exported to Europe for smelting, and then comes back to this country in the form of bar bismuth. There was no marketed production of bismuth ore in the United States in 1902 or 1903. In 1904 sales amounted to 5184 ℔., valued at $314; in 1905 they amounted to 20,504 ℔., valued at $4187. This ore was obtained chiefly from the Ballard mine, at Leadville, Col. Small shipments of ore also were made from a newly discovered deposit, situated 55 miles east of Banning, California. In 1906 there were no shipments from Leadville, and we have not yet had reports of any shipments from other places in the United States, except an experimental lot from a mine near Keystone, South Dakota.

An important discovery of ore containing bismuth was reported in the Washington basin, Custer county, Idaho, during 1906. At this place, the old Empire group of mines was taken over by the Idaho-Montgomery Mining Company, Ltd., the main office of which is at Hailey, Idaho. The property is traversed by five large veins, which dip to the southeast about 40 deg. Bismuth, both in the form of maldonite and tetradymite, is said to occur in large quantities throughout all the veins, in connection with galena of high grade in silver and gold. The bismuth ore itself is said to be high in silver and gold and to assay 55 to 58 per cent. bismuth.

IMPORTS OF BISMUTH INTO THE UNITED STATES.

Year.	Pounds.	Value.	Av. per lb.
1896	124,263	$ 90,950	$0.73
1897	151,374	172,236	1.14
1898	137,205	162,846	1.19
1899	176,668	208,197	1.18
1900	180,433	246 597	1.37
1901	165,182	239,061	1.45
1902	190,837	213,704	1.12
1903	147,295	235,199	1.60
1904	185,905	339,058	1.82
1905	148,589	318,007	2.14
1906	254,733	318,452	1.25

It is probable that the introduction of lead refining by the Betts electrolytic process at the works of the United States Metals Refining Company, at Grasselli, Ind., near Chicago, will lead to a production of bismuth in marketable form in this country, inasmuch as the bismuth content of

work-lead, which in the Parkes process of refining goes into the merchant lead, in refining by the Betts process is concentrated in the anode slime, from which it may be recovered. So far, a suitable process for its extraction from the anode slime has not been devised, but this appears to be nothing more than an ordinary metallurgical problem, and there is little doubt that its solution will soon be found and bismuth will be produced from this material. The results of such a new competition in the bismuth market will be awaited with great interest. It is well known that bismuth is one of the most objectionable impurities in corroding lead, for which purpose the content must not exceed 0.025 per cent., while many varieties of work-lead contain considerably more than that.

The chief bismuth smelters and refiners of the world are Johnson, Matthey & Co., of London, the royal smelteries of Freiberg and Oberschlema, Saxony, and the Deutsche Gold- und Silberscheideanstalt, of Frankfurt am Main. These three concerns form the convention, or trust, which until recently held the monopoly in treating bismuth ores, by-products and crude metal, and which still determines the market price of the metal.

The sources of supply of ores have been Queensland, Bolivia, Mexico, the United States, the Erzgebirge (Saxony) and Spain. The bismuth ore obtained from the United States has come from Leadville, Colo., and has been of comparatively small amount. Formerly only high-grade and pure ores were bought, and even for such only low prices were given. With increasing competition, however, ores of a mixed character began to fetch fair prices, and for the purer kinds reasonable terms can now be expected. Frequent parcels of mixed bismuth and copper ores, generally in the form of oxides, with about 10 per cent. Bi and 10 to 20 per cent. Cu, have appeared on the market, and in these the bismuth has been paid for at the rate of about 10c. per ℔.; when the metal was worth $2.50 per ℔., the bismuth in these ores realized 25c. per ℔. The higher grades of ore, containing 30 per cent. Bi and upward, were sold at $1 per ℔. of bismuth when the metal was at $2.25 to $2.50 per ℔., but now that the metal is worth only about half that amount, prices for ore are proportionately lower.

It is possible that if the market were not artificially contracted, and the metal were mined and smelted without restriction, new uses would be found for it, and manufacturing on a large scale might become a commercial success. Most of the metal now produced is made into pharmaceutical preparations, like the subnitrate, tannate, iodide and bromide. Some is used in the alloy business for making fusible metal.

A considerable part of the bismuth supply of the world is now obtained from Bolivia, whence it is exported both in the form of ore and in the form of crude bar bismuth. The production of bismuth in Bolivia in 1903 was 288,613 kg.; in 1904, it was 405,936 kg., and in 1905 it was 592,412 kg. The Chorolque mines, from which this metal is principally obtained, are

under the control of the European combination. In 1906 the Bolivian government raised the export duty on bar bismuth from $5 (8s. 4d.) to $10 (16s. 8d.) per 100 ℔.

The present price of bismuth, in a wholesale way, at London, is 5s. per pound. The recent fluctuations have been as follows: From Dec. 1, 1904, to March 6, 1905, the price was 10s. per pound. March 6, 1905, it was reduced to 9s. per pound, and Dec. 1, 1905, it was further reduced to 5s. per pound. The reductions in price were made by the combination for the purpose of putting out of business a new competitor in Germany. This was accomplished, but since then the combination evidently has not ventured to elevate the price again.

Bibliography.—Articles on bismuth appear in THE MINERAL INDUSTRY, Vol. II and Vols. V, to XIV inclusive. Aside from the general mention of new discoveries in the United States and in foreign countries, and the usual review of market conditions which are in every volume, various special contributions have appeared as follows: In Vol. II, a complete account of occurrences of bismuth ore, method of reduction and the uses of the metal; in Vol. V, the metallurgical methods employed in Saxony for the production of bismuth; in Vol. VIII, a very thorough set of notes on the metallurgy of bismuth, by W. Borchers, including the extraction of bismuth in crucible and reverberatory furnaces, by the precipitation and the wet methods; also an account of the refining of crude bismuth, with cuts of the principal types of furnaces used in this art; in Vol. X, a review of several methods of determining bismuth analytically; in Vol. XIII, a procedure for the determination of bismuth by gravimetric means and a method for the electrolytic separation and determination of bismuth in the presence of copper and lead.

BORAX.

Borax is obtained mostly from the mineral colemanite and to some extent from ulexite. Crude borax (tinkal) is also mined, but at the present time is of less importance as a source of borax than colemanite. Practically all of the borax produced in the United States is obtained from San Bernardino, Ventura and Inyo counties of California. A small production is made during the summer months from marshes in Nevada, where the mud contains limited amounts of borax minerals, sufficient to make exploitation profitable when the deposits are worked under specially advantageous conditions. Oregon has deposits of sodium borate in the marshes of Harney county, but little, if any, production is made from that region as present. The subjoined table shows the production of borax in the United States for a series of years.

PRODUCTION OF BORAX IN CALIFORNIA. (a)
(In tons of 2000 lb.)

Year.	Tons.	Value.	Year.	Tons.	Value.	Year.	Tons.	Value.
1895........	5,959	$595,900	1899......	20,357	$1,139,882	1903......	34,430	(c) 661,400
1896........	6,754	675,400	1900......	25,837	1,013,251	1904......	45,647	(c) 698,810
1897........	8,000	1,080,000	1901......	7,221	982,380	1905......	46,334	1,019,158
1898........	8,300	1,153,000	1902......	(b)17,202	2,234,994	1906......	....	....

(a) Reported by the California State Mining Bureau. (b) Mostly refined borax, whence the apparent discrepancy in value. Output of the other years is given as crude material. (c) Spot value.

Market and Prices.—The market for borax is under the control of a few eastern concerns, chiefly the Pacific Coast Borax Company, which have refining works in Bayonne and in Jersey City, N. J. A large proportion of the crude material is shipped from California across the country, refined and distributed to consumers from eastern points. A small part only of the total production is refined at the point where it is mined.

During 1906 borax was quoted in New York at 7¼@7½c. per lb. in large lots, which is about ½c. per lb. higher than in 1905, and is a return to the level reached in 1904. Chicago prices are somewhat lower than those of New York, although the material is shipped from California to New Jersey, refined, and shipped back again to Chicago. The borax of commerce, as quoted above, contains about 36.6 per cent. anhydrous boric acid, the remainder being soda and water of crystalization. The crude material as it comes from the ground contains from 5 to 35 per cent. boric acid, according to the nature of the deposit. The figures given for 1905 in the table represent the total tonnage of crude material, regard-

less of its contents of boric acid, but the value of the material is based upon its boric acid content.

Refining.—The principal refiners of borax in the United States are: Pacific Coast Borax Company (Borax Consolidated, Ltd.), of Bayonne, N. J.; Pfizer & Company, Brooklyn, N. Y.; Brighton Chemical Company, New Brighton, Pa.; Thomas Thorkildsen & Co., Chicago, Ill.; and the Stauffer Chemical Company, San Francisco, Cal. During 1906 comparatively little was published as to the technology of borax refining. Henry Blumenburg, Jr., of Daggett, Cal., patented (U. S. pat. 809,550, Jan. 9, 1906) a process of making boric acid, the same being assigned to the American Borax Company, of Pittsburg. It is adapted to the reduction of borosilicates of lime, borate of lime or combinations having alkali-earth metals as a base. The apparatus consists of a closed vessel or generator in which sulphur is burned, air being introduced under pressure to produce sulphur dioxide. A portion of the sulphur dioxide is diverted to the outer atmosphere, and led into the presence of finely divided wet mineral, which is decomposed, releasing the boric acid, which is taken up by the water in solution, and may be separated by evaporation.

SOME OF THE PRINCIPAL SUPPLIES OF BORAX PRODUCTS.
(In metric tons.)

Year.	Chile. (a)	Germany (b)	Italy.			United States. (d)	Total.
			Borax Refined.	Boric Acid.			
				Crude.	Refined.		
1897	3,154	198	990	2,704	260	7,257	14,563
1898	7,028	230	702	2,650	166	7,529	18,305
1899	14,951	183	709	2,674	129	18,466	37,112
1900	13,177	232	858	2,491	283	23,437	40,478
1901	11,457	184	544	2,558	347	6,550	21,640
1902	14,327	196	....	2,763	....	15,512	32,798
1903	16,879	159	....	2,583	....	31,232	50,853
1904	(c)	135	569	2 624	314	41,407	45,049
1905	(c)	183	(f)1,007	2,700	(f)749	42,030	42,213

(a) Borate of lime exports except for 1903, which year gives production figures. (b) Boracite. (c) Statistics not yet available. (d) Crude borax. (f) Obtained by treating a part of the crude boric acid reported for 1905.

Borax in Foriegn Countries.—In Chile the Borax Consolidated, Ltd., is in control of the deposits, having a monopoly from the Chilean Government. The deposits of the company are situated at Chilcaya and Ascota. The control of the deposits at Chilcaya is now the subject of litigation, the right of the Borax Consolidated to operate them having been questioned by the Sindicato del Norte. Considerable borax mineral (pandermite) comes from Turkey, but no recent statistics of the quantity are available. In India, the borax industry, as reflected by exports, has been decreasing slightly. In 1905 there were exported 4198 cwt., against 4246 cwt. in 1904. Recently a discovery of borax and soda was reported in the

lake bottoms of the district around Ashcroft, B. C. On some of the smaller lakes of the district the crust of borax along the shores are said to be thick enough to be cut and handled like ice. Further details of the discovery are lacking.

Bibliography.—The usual statistics of borax and enumeration of producers and notes on new discoveries are to be found in every volume of THE MINERAL INDUSTRY. A few special features of various articles are summarized herewith. In Vol. I, a review of the principal deposits of borax throughout the world together with brief notes on the methods of treating native borax and ulexite. In Vol. V mining conditions and borax deposits of California are described and the progress of the borax industry and the conditions which have affected market prices are treated. In Vol. VII the Chilean deposits of borax are described, and the progress in the technology of borax is treated, and there are also a few notes on methods of estimating boric acid analytically. In Vol. VIII the process of refining borax as carried out at Bayonne, N. J., is described, and there is a procedure given for the determination of boric acid by an iodimetric method. In Vol. X a brief description of the deposits of borax in Oregon and a résumé of analytical chemistry in so far as it deals with the determination of borax are given. In Vol. XIV there is brief description of the methods of refining borax as carried out by the American Borax Company at Daggett, California, also a new method of determining boric acid is outlined.

BROMINE.

Of late years the production of bromine and of bromide salts has been increasing at a remarkable rate in the United States, and 1906 was no exception, as will be seen by reference to the accompanying table. The increase in production in 1906 was the greatest yet recorded; the remarkable increase in the production of Michigan was especially noteworthy. In 1905 Michigan's product showed a decrease as compared with 1904, but all the ground which was lost has been more than regained. In all districts, except a part of the Michigan area, bromine is produced as such, but of Michigan's total product approximately four-fifths is in the form of bromide salts. It is always customary to report the production of Pennsylvania in connection with the State of Ohio, in order to conceal identity in those States, there being but one producer of bromine in Pennsylvania.

PRODUCTION OF BROMINE IN THE UNITED STATES.
(In pounds.)

Year.	Michigan. (a)	Ohio and Penna.	West Virginia.	Total.	Metric Tons.	Value.	
						Total.	Per lb.
1897	147,256	241,939	97,954	487,149	221	$136,402	28c.
1898	141,232	226,858	118,888	486,978	220	136,354	28
1899	138,272	193,518	101,213	433,003	196	125,571	29
1900	210,400	196,774	114,270	521,444	237	140,790	27
1901	217,995	227,062	106,986	552,043	250	154,572	28
1902	226,452	194,086	93,375	513,913	233	128,742	25
1903	320,000	180,000	97,000	597,000	271	170,145	28½
1904	646,249	147,807	85,256	879,312	399	215,431	24½
1905	579,434	223,000	97,000	899,434	408	139,492	15½
1906	955,000	203,000	71,000	1,229,000	553	184,350	15

(a) Includes the bromine equivalent of the bromides recovered.

The production and output of bromine in the United States is in the hands of a single combination, as is also the case of the German production, Germany being the only other considerable producer of bromine outside of the United States. The history of the bromine industry is a series of agreements between the United States and the German syndicate to control prices, the agreements leading to high quotations and the breaking of them being signalized by a drop in American prices due to aggressive German competition.

During 1906 the German bromine syndicate sold 1,995,380 lb. of bromide salts as against 1,578,350 lb. in 1905, an increase of about 27 per cent. Exports to the United States were 269,376 lb. showing a decrease as compared with 1905. It is reported that a meeting has recently taken place

between representatives of the German syndicate and of American producers with a view to checking unlimited competition with the latter. If the agreement can be carried through, it is likely that bromine prices in the United States will in the future be somewhat higher.

During 1906 the average price of bromine at New York in large lots to jobbers was 15c. per lb. which was about the same as during 1905. Bromide salts for export were quoted at 17 @ 18c. per lb.

Bibliography.

The sources of bromine, the producers and the general conditions of the bromine industry are discussed in every volume of The Mineral Industry but several noteworthy features in addition to these customary notes are as follows: In Vol. II there is a description of the method of recovering bromine from the mother liquors of Stassfurt and a discussion of current market conditions. In Vol. VIII, D. C. Hinman gives further notes on the production of bromine as carried out at Stassfurt, and an electrolytic method for the recovery of bromine from waste liquors is described under the caption "Progress in Electro-Chemistry during 1899" in the same volume. In Vol. IX, A. C. Lane describes the production of bromine as it is carried on in Michigan.

CALCIUM CARBIDE.

The production of calcium carbide in the United States is controlled by the Union Carbide Company, operating at Niagara Falls and at Sault St. Marie. The company acts as a distributor of carbide to various houses throughout the country which serve as local depots and centers of supply. The policy of the Union Carbide Company is against giving out information, and so no statistics of production of calcium carbide can be given. The only other producer in the United States is a small concern which makes an insignificant production, and at present the users of its product are being sued by the Union Carbide Company which claims to control fundamental patents broadly covering crystalline calcium carbide as a product; also the processes, apparatus and devices applicable to its manufacture and use.

Of late much attention has been paid to spreading knowledge concerning calcium carbide and acetylene to all possible prospective users, and this policy has resulted in a considerable increase in the importance of the industry. Especially has attention been turned toward inducing towns and small cities to establish municipal acetylene lighting plants, and to this end, carbide is now prepared and shipped in special packages to towns and cities at a considerable reduction in price, in some instances as much as $16 per ton, the average reduction being $10 per ton. A uniform price of $70 per ton for calcium carbide in ton lots was quoted during 1906 throughout a large zone extending from the Atlantic Ocean on the east to Omaha and Kansas City on the west, and as far south as Charleston and Memphis. Calcium carbide is carried at nearly every important shipping center within that zone, and sold at the price named, f.o.b. those distributing points. On the Pacific Coast and at Gulf points higher charges are made. This price is stated to be as low a figure as it is sold for in Europe in spite of the fact that the labor-cost in making calcium carbide is much lower there than here.

Uses.—In addition to the town lighting plants above mentioned acetylene is being used to an increasing extent in railroad and car lighting. Nearly a hundred different railroads are using it for locomotive headlights, station and car illumination. Some of the large systems which have adopted and are using this form of lighting are the Great Northern, the Delaware, Lackawanna & Western, the Canadian Pacific, and the New York, New Haven & Hartford. The question of the use of acetylene for power

purposes and welding has been brought up several times, and these uses appear not only practicable, but economical. Considerable attention is being given to the use of acetylene lamps in mining, wherein they have a promising field.

In Europe, large producers of calcium carbide are reported to be undertaking experiments with a view of producing cyanamide, a new fertilizer, which it is hoped will largely supplant Chilean nitrate. If such happens, a large increase in European carbide production will take place at once. At present the chief European countries which manufacture calcium carbide are France, Germany, Norway and Italy.

Bibliography.

The subject of calcium carbide has been treated at length in articles which have appeared in The Mineral Industry, Vols. VI-XI inclusive, as follows: Vol. VI contains an article of 24 pp., by Alfred H. Cowles, which gives a complete history of calcium carbide as a source of acetylene for illuminating purposes, and a sketch of the development of the carbide industry. The various means of producing acetylene are discussed, and the properties of that gas are considered. Descriptions of various means of storing and transporting acetylene and calcium carbide are likewise described. In Vol. VII an eight-page article describes the manufacture of calcium carbide in France, Germany and Switzerland. The regulations of the New York Fire Department for the storage and transportation of calcium carbide and acetylene are summarized. The operation of various standard types of furnaces for manufacturing calcium carbide is described. In Vol. VIII, John B. C. Kershaw gives an account of the industrial conditions of the various calcium carbide companies. The progress of the art of illumination by means of acetylene is summarized and some of the principal public installations of this system are described. In Vol. IX, L. K. Böhm contributes a 12-page article, which gives the progress of the calcium carbide industry during 1900 and describes the leading manufacturing plants in Austria-Hungary, France and Germany. In Vol. X there is a discussion of the most recent type of acetylene generators, and the application of acetylene to car illumination. Central plants for the production of acetylene for illumination are discussed.

The following articles and patents on calcium carbide were published in the technical press and patent-office gazettes during 1906:

"Die Abschiedung des Kohlenstoffes aus Carbiden." C. Hahn and A. Stutz. (*Metallurgie*, Nov. 8, 1906; 5 pp.). A discussion of the thermal and chemical reactions that occur when carbides are treated with water.

"Acetylengruben Lampen für Schlagwetter freie und Schlagwettergruben." (*Oest. Zeit.f.B.u.H.* June 2, 1906). Illustrated description of

acetylene lamps, both safety lamps and open lamps, employed in Europe, with data of cost, etc.

"Acetylene Lamps for Miners." F. W. Parsons, (*Eng. and Min. Journ.*, July 21, 1906). Illustrated description of the Baldwin acetylene mining lamp, which is a new, cheap and convenient form of this type of lamp.

"Calcium Carbide, Acetylene." (*Insurance Engineering*, Oct., 1906). Gives a historical review of the advance in the acetylene industry, and technically treats acetylene illumination and the apparatus for that purpose.

Among the new patents of 1906 were the following:

"Manufacture of Carbide." Herman L. Hartenstein, Chicago, Ill., assignor to Electro-Chemical and Development Company, Pierre, S. D. U. S. pats. 819,219; 819,220; 819,221; 819,222.

"Process of Reducing Calcium Oxide." Thomas L. Wilson, New York, N. Y., assignor to Union Carbide Company. U. S. pat. 820,031.

"Process of Making Hydrochloric Acid and Calcium Carbide." William H. Seamon, El Paso, Tex. U. S. pat. 826,614.

"Process of Reducing Metallic Compounds and Producing Carbides." Edgar F. Price, Niagara Falls, N. Y., assignor to Union Carbide Company. U. S. pats. 826,742; 826,743 and 826,744.

"Apparatus for Reducing Compounds and Producing Carbides." Edgar F. Price, Niagara Falls, N. Y., assignor to Union Carbide Company. U. S. pat. 826,745.

"Apparatus for the Production of Calcium Carbide." Edgar F. Price, George E. Cox, and James G. Marshall, Niagara Falls, N. Y., assignors to Union Carbide Company. U. S. pat. 809,842.

CARBORUNDUM.

The Carborundum Company of Niagara Falls, N. Y., is the only producer of carborundum in America, and during 1906 it produced 6,225,280 ℔. an increase of 629,000 ℔. over the production for 1905, which did not, however, equal the high mark reached in 1904. The increasing demand for carborundum may be seen from the following table:

PRODUCTION OF CARBORUNDUM IN THE UNITED STATES.

Year.	Pounds.	Metric Tons.	Value.	Year.	Pounds.	Metric Tons.	Value.
1897...........	1,242,929	564	$153,812	1902..........	3,741,500	1,697	$374,150
1898...........	1,594,152	723	151,444	1903..........	4,760,000	2,159	476,000
1899...........	1,741,245	790	156,712	1904..........	7,060,380	3,203	706,038
1900...........	2,401,000	1,089	216,090	1905..........	5,596,280	2,539	559,628
1901...........	3,838,175	1,741	345,435	1906..........	6,225,280	2,779	(*e*)622,528

(*e*) Estimated.

Abroad, the Carborundum Company has been engaged in the construction of a plant in Germany, the demands of its European trade having increased so rapidly that the establishment of a branch works on the Continent became necessary. A German company has been formed under the title "Deutsche Carborundum Werke, G. m. b. H." The new works are located at Reisholz, near Düsseldorf. Five kilns for the manufacture of vitrified wheels are provided for, two of these to be built at once.

Value.—Carborundum is now displacing corundum and emery as an abrasive, for which purpose it is far superior to the natural product. It is more expensive than these substances, costing 10 @ 17 c. per ℔. f.o.b., Niagara Falls, in large quantities against 3.5 @ 4.5 c. per ℔. for emery grains. However, this difference in cost is more than offset by the longer time that carborundum will wear as against emery used for the same purpose.

Technology and Uses.

Manufacture.—Carborundum is made by fusing in the intense heat of the electric arc, a mixture of granulated coke, glass sand, salt and sawdust. The essential reaction takes place between the silica of the sand and the carbon of the coke, the exact transformation being the reduction of the silicon dioxide by the coke when a sufficiently high temperature is reached, and the combination of the excess of carbon, with the reduced silicon to form a carbide of silicon, having the composition SiC, which is carborundum. The salt and sawdust are added merely to make the mixture coarse and to allow the free exit of the gases which are evolved when fusion is going on.

Another product of the electric furnace when it is used in making carborundum is carborundum fire-sand, which is amorphous carborundum and is found in the cooler parts of the furnace after it has been broken down at the end of a run. This fire-sand is used as a refractory material for the lining of furnaces, especially those using oil fuel. Siloxicon is a second product and is obtained when insufficient coke is present in the original charge, so that complete reduction of silica cannot take place. The composition of this product is somewhat variable but a typical formula is Si_2C_2O. This substance is somewhat cheaper than carborundum, but it does not last as long in use. Details of furnace construction and operation, and a review of progress in the process of manufacturing carborundum, may be found in Vol. XIV of The Mineral Industry.

Behavior at High Temperature.—When carborundum was first discovered, it was appreciated for its remarkable hardness which made it a valuable abrasive. But in the patent covering its manufacture, its refractory character was mentioned as a valuable property, and recently its application as a resistant to high temperature has been greatly extended. Carborundum apparently has no fusing point, as it decomposes into graphite and silicon without giving any evidence of having fused at its decomposing temperature. Tucker and Lampen[1] made some experiments to determine the formation temperature of carborundum, and also the temperature at which it is decomposed into graphite and silicon. As fairly sharp lines of demarkation are observed between the layers of graphite, carborundum and siloxicon after a run of a carborundum furnace, the authors reasoned that the temperature might be obtained by measurements at different points of the cross-section, after the furnace is brought to a stationary condition. A horizontal graphite tube was, therefore, placed through the center of the furnace, which tube contained a plug which could be pushed to any desired section, and its temperature measured by an optical pyrometer. The laboratory furnace, in which the experiments were conducted, was built on the general plan of a furnace used in the manufacture of carborundum, and filled with about 8 kg. of a charge consisting of coke, 34.2 parts by weight; sand, 54.2 parts by weight and sawdust, 9.9 parts by weight. A tube, bored out of Acheson graphite, 330 mm. long, 25 mm. outside and 18 mm. inside diameter, was put transversely through the center of the furnace. A graphite plug, 17.5 mm. in diameter and 9 mm. long, was put into the tube. The instrument used was a Wanner optical pyrometer. The authors found the decomposition point of carborundum into graphite and silicon to be at 2218 deg. C., and its formation temperature, that is, the point where the amorphous turns into crystalline, at about 1920 deg. C. From another experiment the decomposition point of carborundum was found to be 2223 deg. C.,

[1]*Journ. Am Chem. Soc.* July, 1906.

and the formation point 1980 deg. C. The line between siloxicon and carborundum is always less defined than that between carborundum and graphite. The average of the formation point is given as 1950 deg. C., and of the decomposition point 2220 deg. C.

Uses.—The hardness of carborundum lies somewhere between nine and 10, the diamond being 10. Carborundum is insoluble in water and acid and has a less specific gravity than emery which is advantageous inasmuch as a carborundum wheel is lighter than an emery wheel of the same dimensions. In making carborundum wheels and blocks for grinding purposes, a hydraulic press is used to mold a mixture of carborundum grains and vitrified bond. The molded forms are placed on clay supports and vitrification is carried on in kilns similar in construction to those used in glazing porcelain. The bond is made of a mixture of six parts of carborundum powder to one part of fire-clay. Water-glass can be used as a bond, but the solution should be dilute, as too much of the silicate of soda renders the carborundum less refractory. Tar, glue and shellac are sometimes used as bonding material.

The use of amorphous carborundum as a refractory is increasing, the principal fields of its application being in the lining of oil-burning furnaces and especially in furnaces for melting brass and copper. In the form of brick it has given very satisfactory use in the arches of reverberatory furnaces. The use of both the crystalline and amorphous carborundum for the manufacture of zinc retorts is increasing, especially among foreign smelters. The best results are secured by making the inner lining of the retort of carborundum and the exterior of fire clay.

Recent Patents.—The following recent patents relate to the production of carborundum or similar substances:

Siloxicon process. B. Seaboldt, Salt Lake City. A method of making siloxicon into specified shapes, by heating with clay and carbonaceous matter to a temperature at which the clay melts, but below the melting point of siloxicon. Brit. pat. 17,890 of 1905.

Manufacture of silicides and silicon alloys. Frank J. Tone, Niagara Falls, N. Y. U. S. pat. 833,427.

Refractory brick. E. R. Stowell, Corydon, Indiana, A refractory brick consisting of amorphous carborundum mixed with 10 or 15 per cent. of portland cement. Brit. pat. 9906 of 1906.

Process of making refractories. Siemens & Co., Charlottenburg, Germany. Forming bodies of definite shape out of carborundum, by mixing the carborundum with uncombined silicon and submitting to the action of an electric furnace, the bodies so formed being exceedingly refractory, dense and unaffected by acids, etc. Brit. pat. 21,347 of 1905.

Process for the manufacture of artificial emery. Adrian Gacon, Montval, near Marly-le-Roy, France. U. S. pat. 820,859.

The Carborundum Industry in 1906.

By F. J. Tone.

During 1906 no changes of importance were made in the methods of manufacturing carborundum. The plant of the Carborundum Company at Niagara Falls has now been enlarged to utilize an electric furnace capacity of 7000 h.p., and a branch plant for the manufacture of carborundum wheels and abrasive articles has been established at Düsseldorf, Germany. This plant was put in operation in February, 1907. It receives carborundum in graded form from the American works and manufactures a full line of carborundum products for the Continental trade. Many new applications of carborundum have recently been made in the arts but notwithstanding its varied uses in the mechanical, chemical and metallurgical fields its application as an abrasive is still of chief importance and consumes the major part of the production.

Abrasive Uses.—In this field a development of great interest is the application of carborundum to the marble industry. The methods of cutting, dressing and polishing marble are now in a process of rapid and complete revolution owing to the use of carborundum wheels. A complete line of machinery has been developed for the various operations of coping, countersinking, molding, rubbing and polishing which largely dispenses with the old style machine tools and also with skilled labor. The molding machine equipped with carborundum wheels is capable of removing stock at the rate of 60 cu. in. per minute. The carborundum drum rubber displaces five of the old style rubbing beds.

The methods of bevelling plate glass have undergone radical changes and one operator is now able to bevel 6500 ft. per day using in this time 10 lb. of carborundum grains. The use of carborundum has long been general throughout the granite industry and its introduction in the marble and glass industries naturally follows and promises equally important economies.

Carborundum paper is now being introduced in the woodworking trades where it displaces garnet paper, and into the hat trade in competition with fine flint paper for pouncing and finishing hats. The paper industry now utilizes carborundum in the form of blocks for the construction of bed plates in the beaters and for the lining of Jordan engines.[1] It assists in refining the pulp.

Carborundum applied to non-slipping stair treads, carriage treads and to non-slipping horse shoes has been made the subject of several recent patents.[2] This branch of the trade consumes an important amount of product. Carborundum is being introduced for the same purpose in the construction of cement pavements and sidewalks.

[1]U. S. pat., 1904, S. R. Wagg.
[2]U. S. pats., 771, 772, Etheridge; 783, 288, Lamson.

Refractory Uses.—Amorphous carborundum, or as it is commercially called, carborundum fire-sand, is now widely used as a refractory material and the methods of using it in the form of bricks and various furnace linings have been the subject of a large number of patents[1] both in this country and abroad. This product occurs in the carborundum furnace immediately outside the crystalline zone and contains carbon, silicon and oxygen in the form of various compounds representing the partial reduction of silica by carbon. It is used in lining crucible furnaces for melting brass and also in the later designs of tilting brass furnaces, especially in those burning crude oil fuel. It resists severe flame action as do few refractory materials available to the furnaceman. The material is ground to the fineness of about No. 20 mesh and is mixed according to the following formula: carborundum fire-sand 70 parts, fire clay 15 parts, silicate of soda 52 deg. B. eight parts, water seven parts. This mixture is tamped in place and slowly dried. When subjected to furnace temperatures it burns into a strong refractory body. Amorphous carborundum is one of the few refractories which can withstand the heat of the powdered coal flame. Fire bricks made from carborundum are now on the market and have given favorable results in the arches of copper reverberatory furnaces and also in boiler furnaces where special smoke consuming devices are used. The use of both crystalline and amorphous carborundum for the manufacture of zinc retorts[2] is increasing, especially among foreign smelters and many hundreds of tons were exported for this purpose during 1906. The best results are obtained by making the inner lining of the retort of carborundum and the exterior of fire clay. W. A. McAdam (Brit. pat., No. 16,168, July 17, 1906) uses powdered carborundum in molds for the casting of aluminum and obtains a rapid chilling of the metal which increases its tensile strength.

Electrical Uses.—Carborundum has found important use as a resistance material especially in the manufacture of resistance rods for lightning arresters. In one method of manufacture[3] the rods are made up from a mixture of plastic clay, powdered carborundum, graphite, etc., are and fired in a potters' kiln. The rods are then glazed throughout their length to prevent the absorption of moisture and the ends are electroplated or otherwise treated to provide good electrical contact to the terminals. These rods are generally made in small sizes, from 6 to 10 in. long, and are used to protect circuits of comparatively low voltage. For high potential lines rods made of No. 40 to 60 carborundum grains with a vitrified porcelain binder are giving satisfactory service for potentials of 60,000 volts. These rods are 6 ft. in length and 3 in. in diameter. When measur-

[1] U. S. pats., 770, 083, May; 770, 730, Muller; 772, 262, Tone; 796,459, Seaboldt; 774,003, Stowell; 798,788, Hayt; 819,467, Stowell.

[2] Brit. pats., 14,567 of 1901, Engels; U. S. pat., 755,222, Engels.

[3] U. S. pat., 792,638, Whitney.

ing the electrical resistance of the rods no readings can be obtained with the ordinary Wheatstone bridge. When, however, they are subjected to a potential of 10,000 volts current readings are obtained which indicate a resistance of from 120,000 to 150,000 ohms. Siemens Bros. of Charlottenburg in a recent patent[1] propose to make electric resistance rods and anodes for electrolytic baths from a mixture of silicon carbide and silicon. Another inventor makes resistance rods for heaters, rheostats and the like, using carborundum with a vulcanized rubber bond.[2]

A carborundum wireless detector[3] was developed in 1906 by General H. H. C. Dunwoody of the American DeForest Wireless Telegraph Company and is being used in a large number of the company's installations. The device consists of a minute fragment of carborundum held in place between two metallic terminals or conductor plugs of copper or brass. This detector has recently been made the subject of exhaustive tests by G. W. Pickard who has found that it is somewhat less sensitive than the magnetic detector of Marconi, which in turn follows the electrolytic detector of Fessenden. Notwithstanding this difference in sensitiveness, in the actual practice of wireless telegraphy the difference in receptiveness is barely preceptible over similar distances. The reason for this lies in the human ear which does not estimate differences in sound intensity directly, but rather in the ratio of the square roots of the intensities.

The crystals of carborundum employed in the Dunwoody detector are microscopically selected and those having the sharpest edges have been found to give the best result. The flat side of carborundum is a very poor conductor and in order to obtain good electrical contact, the sharp edges of the carborundum fragment must be clamped between the exposed surfaces of the plug ends of the detector. Actual contact is limited to an exceedingly small area, probably not more than one-millionth of an inch. In the first experiments with carborundum as an electric wave detector it was found that its sensibility to the electrical oscillations set up in the circuit of which it was a part, was a maximum when a certain critical potential prevailed in a local circuit of which it also formed a part. The fragment of carborundum is placed between a spring and an adjustable screw plug and by varying the pressure of the spring by means of a screw the point of maximum sensitiveness can easily be obtained.

As a result of several years research work, Prof. H. C. Parker and W. G. Clark brought out in 1906 the Helion lamp.[4] The filament of this lamp is formed by subjecting a heated carbon filament to vapors of silicon whereby there is produced a conductive compound of silicon and carbon which doubtless coincides chemically with the silicon carbide. The Helion

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[1]Brit pat., 21,347, of 1905.
[2]U. S. pat., 882,232, Gardner.
[3]*Scientific American*, March 16, 1907; *Elec. World*, Nov. 24, 1906. U. S. pat. 837,616.
[4]U. S. pat., 821,017; *Elec. World*, January 5, 1907.

lamp is claimed to have remarkable life and efficiency, surpassing in this respect the new tantalum and tungsten lamps.

Temperature of Formation.—S. A. Tucker and A. Lampen have measured the temperatures of formation and decomposition of carborundum.[1] The method consisted in placing through the center of the furnace a horizontal graphite tube, containing a plug of graphite which could be pushed to any desired section of the furnace. Readings on this plug were then taken with a Wanner optical pyrometer. The results gave the temperature of formation of carborundum as 1950 deg. C and the temperature of decomposition into graphite and silicon as 2220 deg. C. These results would seem to be too low to accord with previous estimates of these temperatures or with facts upon which such estimates have been based.

Physical Properties.—At the request of The Carborundum Company a report has recently been made by Dr. Joseph Hyde Pratt on hardness determinations of carborundum and various samples of corundum which were obtained by the Carborundum Company directly from the producers of the respective abrasives. The following is abstracted from this report:

Six samples were received from the Carborundum Company of Niagara Falls as follows: (1) Crystallized carborundum; (2) very pure piece of corundum from Laurel Creek mine, Rabun county, Georgia; (3) massive alundum; (4) Chester emery showing but very little free corundum; (5) specimen of crystallized corundum from Canada exhibiting superficial alteration on the crystal faces; (6) Corundum in chlorite associated with the emery at Chester, Mass.

In order to determine the comparative hardness of these six samples, they were tested as to their relative hardness with each other and also as to whether or not they would scratch blue sapphire, crystallized transparent corundum and ruby. As is well known, corundum itself varies somewhat in hardness according to the character of the mineral. Thus, to the blue transparent sapphire variety of corundum a hardness of 9 is given while the transparent red corundum (gem ruby) is slightly less in hardness, estimated at 8.8 to 8.9. The samples of gem corundum that were used in testing the six samples were the blue Montana sapphire; polished pieces of North Carolina ruby; and transparent greenish and yellowish gem corundum from Rock Creek, Montana.

With Cowee ruby: Sample 1 scratched the ruby very readily with slight pressure and gave a clean, deep cut. Samples 2 and 3 scratched the ruby but only very slightly and required considerably more pressure than sample 1. The scratch was not as clear and sharp cut as by sample 1. Samples 4, 5, and 6 did not scratch the ruby.

With Montana sapphire: Sample 1 scratched a clean surface of the

[1] *Journ. Amer. Chem. Soc.*, 1906, XXVIII, 853-858.

Montana sapphire easily, making a clean cut. Samples 2, 3, 4, 5, and 6 did not scratch the sapphire.

With green and transparent Montana sapphires: Sample 1 scratched the various specimens of these gem corundums very readily, giving clean, sharp cuts. None of the other samples would scratch any of these. Samples 2 and 3 were not scratched by these corundum gems. Sample 5 seemed to be readily cut by these gems while sample 6 was not scratched by them. Sample 4, which is the Chester emery, would not make any impression on any of the corundums.

With each other: Sample 1 scratched all the other samples readily and where clean, smooth surfaces could be obtained, showed deep, sharp scratches. Sample 2 will not scratch 1 or 3 but will scratch readily 4 and 5. Sample 3 will not scratch 1 or 2 but will readily scratch 4 and 5. Sample 4 will not scratch 1, 2, 3, or 6, but certain pieces of it would readily scratch sample 5. Sample 5 will not scratch 1, 2, 3, 4 or 6. Sample 6 will not scratch 1, 2, 3 or 4, but will scratch 5 slightly.

In making the above tests good, clean surfaces were broken so as to give a sharp cutting edge each time a piece was tested. The scratches were examined under the microscope. It must be taken into account that sample 5 shows superficial alteration and that sample 4 is a Chester emery, a mixture of magnetite and corundum in finely divided particles and that it is the corundum grains that give a hardness to the emery, although the grains of corundum themselves cannot be distinguished.

I should arrange then the samples sent according to hardness as follows: 1. Carborundum (sample 1); 2. Georgia corundum and alundum, (samples 2 and 3); 3. Chester blue corundum, (sample 6); 4. Chester emery, (sample 4); Canadian corundum, (sample 5).

CEMENT.

By Robert W. Lesley.

To the onlooker, the development of the cement industry in the United States appears to be one of the marvels of the world and would certainly seem to justify indulgence in the alleged American propensity to boast. This especially applies to the manufacturer of portland cement, whose product during 1906 put the United States ahead of all countries manufacturing this important building material. When one considers that in 1880 only 82,000 bbl. of portland cement were manufactured and that in 1876, at the time of the Centennial, but three factories were exhibiting samples of what might well be called "experimental cements," it is startling to realize that the production of portland cement alone in 1906 reached the enormous total of 46,610,822 bbl. and that with the construction under way in the United States, it is figured by those in the trade that 1907 will show a production of close on to 50,000,000 bbl. These figures are certainly convincing that the development of this industry in this country has been truly marvelous.

PRODUCTION OF CEMENT IN THE UNITED STATES. (*a*)
(In barrels.)

Year	Portland.			Natural Hydraulic.			Slag Cement.			Total.	
	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.
1897	2,430,903	$3,724,905	$1.54	7,890,573	$3,976,050	$0.50	40,000	$60,000	$1.50	10,361,476	$7,760,955
1898	3,584,586	6,168,106	1.72	8,168,106	3,819,995	0.47	157,662	235,721	1.50	11,903,326	10,223,822
1899	5,805,620	10,441,431	1.80	9,686,447	5,058,500	0.52	244,757	360,800	1.47	15,736,824	15,786,789
1900	8,482,020	9,280,525	1.09	8,383,519	3,728,848	0.45	446,609	567,193	1.27	17,312,148	13,576,566
1901	12,711,225	12,532,360	0.98	7,084,823	3,056,278	0.43	272,689	198,151	0.73	20,068,737	15,860,731
1902	17,230,644	20,864,078	1.21	8,044,305	4,076,630	0.50	478,555	425,672	0.81	25,753,504	25,366,380
1903	22,342,973	27,713,319	1.19	7,030,271	3,675,520	0.50	525,896	542,502	1.03	29,899,140	31,931,341
1904	26,505,881	23,355,119	0.90	4,866,331	2,450,150	0.50	303,045	226,651	0.75	31,675,257	26,031,920
1905	35,246,812	33,245,867	0.94	4,473,049	2,413,052	0.54	382,447	272,614	0.71	40,102,308	35,931,533
1906	46,610,822	51,240,652	1.10	3,935,151	12,362,140	0.60	481,224	412,921	0.86	51,027,321	54,015,713

(*a*) Statistics of production for 1900 and subsequent years are from the "Mineral Resources of the United States." The barrel of portland cement contains 380 lb. of the material; of natural cement, 265 lb.; of slag cement, 330 lb.

To present the matter in another form, the output for 1906 would mean a per capita production of a half barrel of cement for every man, woman and child in the United States, whereas the output in 1880, distributed in like proportion, would not have supplied the residents of a single city of the size of Washington. It is by looking at the situation in this way that the true perspective is obtained.

The great growth of the cement industry from the manufacturer's standpoint was the natural sequence in the remarkable development in the uses of cement. The United States stands preëminent in this respect if measured by the quantity used. Her engineers and builders were not content to be entirely on the side of theory and experiment, but possessed the courage of their convictions and thus the work of the builder has kept pace with the research of the laboratory.

In considering this remarkable development, American engineering and American exactitude in standard requirements are not only to be taken account of, but there is beyond it all a philosophical reason for the development of the cement industry in this country. In the early days, development in paving, sewerage, water supplies and house and factory construction went along the lines of least resistance, and the material selected for almost all these forms of construction was wood, which was the material

STATISTICS OF CEMENT IN THE UNITED STATES.
(In barrels of 380 lb.)

Year.	Production.		Imports.		Exports. (*b*)		Consumption.	
	Barrels.	Value.	Barrels.	Value.	Barrels.	Value.	Barrels.	Value.
1897..	10,361,476	$7,760,955	2,200,871	$2,688,122	62,761	$103,389	12,499,586	$10,345,688
1898..	11,903,326	10,223,822	2,119,880	2,624,228	55,969	98,121	13,967,237	12,749,929
1899..	15,736,824	15,860,731	2,219,246	2,858,286	116,079	213,457	17,839,991	18,505,560
1900..	17.312,148	13,576,566	2,512,300	3,330,445	147,305	289,186	19,677,143	16,617,825
1901..	20,068,737	15,786,789	994,624	1,305,692	303,380	752,057	20,759,981	16,340,424
1902..	25,735,504	25,366,380	2,100,513	2,582,281	367,521	575,268	27,486,496	27,373,393
1903..	29,899,140	31,931,341	2,439,948	3,027,111	312,163	466,140	32,026,925	34,492,312
1904..	31,675,257	26,031,020	1,101,361	1,383,044	816,640	1,158,572	31,959,978	26,256,392
1905..	40,102,308	35,931,533	891,134	1,102,041	1,060,054	1,428,489	39,933,308	35,605,085
1906..	51,027,321	54 015,713	2,321,803	2,950,268	600,386	964,373	52,748,738	56,001,608

nearest at hand, the most adaptable and easiest obtained. This material, subject to destruction by fire, rotting and weather, was used in the early railway bridges and in early municipal improvements, as well as house building in the United States. With the development of wealth came the demand for more permanent structures—buildings of a more fireproof character—and it so happened that co-incidently with this condition, the denudation of the forests began to cause a scarcity in timber and a consequent advance in the price of lumber. American constructive and engineering talent was driven to the consideration of a building material which should be substituted for wood, and which should have the elements of greatest economy, greatest permanence and greatest facility in its erection. Cement, and especially portland cement—the manufacture of which was beginning to be developed in this country—appeared to be the ideal material, being economical, permanent, fireproof, and when used in the form of concrete, capable of construction by the cheaper classes of uneducated labor. It is this combination of circumstances which has led

to the enormous development of the portland cement industry in the United States, while in other countries where permanent construction is centuries old and where iron, stone and brick have been used for many generations, the market for cement has not proportionately increased.

The statistics of production of cement in the United States since 1896 are given in the accompanying tables.

The estimate of 36,038,812 bbl. of portland cement for the year 1905, made by the Geological Survey of the United States, was subsequently corrected to 35,246,812 bbl., and taking the latter figure as the basis for 1905 and comparing it with the figures for 1906, it will be seen that the amount of cement of all kinds made in 1906 exceeded that of 1904 by over 60 per cent. as against an increase of 26 per cent. recorded by 1905 over 1904, and as against an increase in 1906 over 1905 of 21 per cent. The year 1906 was one of large construction, especially by the railroads, and also in some of the larger cities where great municipal works were undertaken.

The improvement in prices was noticeable and continued throughout 1906. In 1905 the production was 40,102,308 bbl. having a value of $35,931,533. Comparison of totals shows an increase in 1906 of 10,925,013 bbl. in production and $18,084,190 in value. Of the above total amount of cement manufactured in the United States in 1906, 46,610,822 bbl. were portland cement, with a value of $51,240,652; 3,935,275 bbl. were natural rock cement, valued at $2,362,140, and 481,224 bbl. were slag (or puzzolan) cement valued at $412,921. From these figures it will be noted that even though the prices of portland cement in 1906 were firmly maintained and were in excess of those of 1905, yet the natural cement industry showed a slight decrease from 1905.

The result of the great demand for cement in this country had a material effect upon the foreign business, illustrating again the tendency of American manufacturers in prosperous times to curtail their foreign markets. While in 1905 the cement business for the first time in the history of the United States showed an excess of exports over imports, this condition was reversed in 1906 when the exports were 600,386 bbl. and the imports 2,321,803 bbl. It is greatly to be regretted that owing to the fair prices prevailing during 1906 and the enormous demand for cement in the United States, many of the foreign markets where American manufacturers had firmly established their brands were neglected and lost. The permanence of the export business, which in time will become a very important factor in the country's cement trade, can be maintained only by regular and constant supply of well known brands to the foreign markets, without regard to more favorable conditions prevalent in this country in good years.

The Market Conditions.

While 1906 produced better results to the cement manufacturers than the bad years of 1904 and 1905, nevertheless many hold-over contracts from these years of low prices tended to keep down the average figures for portland cement. In some cases these contracts covered only the early months of the year, while in others they ran during the whole year. It may be stated, however, in a general way, that the open winter of 1905-1906 had much to do with the distribution of the stocks of cement which usually accumulate in cement mills during the winter months, and that for this reason there was no attempt on the part of the smaller manufacturers to break the market by throwing upon it quantities of cement at sacrifice prices at a time when there was no demand. The open winter and the continuance of building operations practically through all the winter months enabled these manufacturers to find a market for their surplus, and money for their pay rolls, and the end of the winter of 1905-1906 saw a condition of affairs that promised good results to the cement industry.

With the ascertainment of the 1906 crop production of the United States and the permanence of the general prosperity of the country, many of the large contemplated improvements became facts and this gave rise to an unexampled demand for portland cement for all forms of construction. Midsummer found all the mills sold up on contracts and many short on actual demands. The San Francisco disaster accentuated this condition in the far western cement territory, as well as among the mills on the Pacific coast. This section of the country was unusually prosperous so far as the cement industry was concerned, the demand from the new mining districts, from the Government and from San Francisco creating such a condition of affairs as to exhaust the supply of the adjacent territory and to draw to the Pacific coast cement from the mills as far east as the Mississippi River.

The demand throughout the United States continued during the fall months of 1906 and this demand was only disturbed in the cement industry —as in the iron, steel and other great industries of the country—by the unfortunate shortage in the car supply, which tended to reduce production for lack of the necessary fuel and to reduce output for lack of the cars required for shipment. The end of the year saw a continuing demand and a good outlook for 1907.

Distribution of the Industry.

The figures for 1905 show for the first time a transference of the center of the industry. Up to that year the Lehigh district, comprising the counties of Lehigh and Northampton, Pennsylvania, and Warren county, New Jersey, had for many years manufactured more than 50 per cent. of all

the portland cement produced. In 1905 this district manufactured a total of 49.3 per cent. of the total output, or 17,368,687 bbl.

During 1906, however, there has been a large additional capacity in the Lehigh field, many of the older mills having increased their works, and several new plants being now under construction. For many years it was believed that this section would be to the cement industry what Pittsburgh is to the iron trade—that it would in output always dominate. The figures for 1906 seem to indicate that this condition will no longer prevail, though there is no question that the wonderful adaptability of the argillaceous limestones and cement rocks of the Lehigh district for cement making will, for many years, make that district a great factor in the trade. The softness of these rocks; their practical uniform composition, and their juxtaposition to limestones of similar texture have had much to do with the firm hold the Lehigh field has had upon the industry.

Further than this, the geographical relations to large markets and the coal fields have also been causes contributing to the above results; but more than this, the labor market in the Lehigh region has always been a stable and satisfactory one and under fair and reasonable conditions, made so in many cases by the fact that in this district there are many silk, thread and lace mills and other like industries, where the women of the workingmen's families find steady employment all the year round.

Industrial Conditions.—In the report of the Geological Survey for 1905, and in many interesting articles published on this subject, Edwin C. Eckel has presented most enlightening figures and statistics to show that the portland cement industry is a national industry and incapable of being monopolized by any trust. He says upon this subject:

"At intervals an excited and not particularly well-informed newspaper press inveighs against the enormous profits realized by a purely imaginary cement trust. To those acquainted with the highly competitive nature of the cement industry, in its present stage, such denunciations may seem too absurd for denial, but there is reason to believe that they are accepted as true by the general public. It may, therefore, be of advantage to state that there never has been a cement trust in this country and that at present there seems to be little opportunity for arranging any satisfactory combination of the cement producers. Several years ago a few fairly large companies took up this matter, but as three of the largest plants could not be induced to consider it, the question never reached an advanced stage.

"The nature of the industry renders it improbable that any combination or non-competitive arrangement can be carried through to such a point as to result in a monopoly of the industry and permanently high prices. Good raw materials are so widely distributed throughout the United States that there is hardly a county which could not produce portland cement if prices were forced high enough. The only limitation now on the erection

of cement plants is the fact that they cost too much for an individual or a small firm to enter the business. A plant producing 2000 bbl. per day will require an investment, for land, plant, and working capital, of between $1,000,000 and $1,500,000.

"Setting aside as impracticable the question of trusts and combinations based on monopoly of raw materials, it can be said that there is noticeable a certain concentration of interests in the cement industry, and that this will probably become more marked year by year. The 88 plants in existence in 1905 are owned by 78 companies, and several of these nominally independent companies are closely connected in ownership.

"Of these 78 companies, 15 produced over two-thirds of the entire American portland cement output. Seven of these showed an annual output of over one million barrels each, and these seven together produced somewhat over half of the entire output of the country. To place the matter in a comparative form, the five largest companies together produced about the same percentage of the American cement output that the United States Steel Corporation does of the American output of pig iron. The cement industry is at present, therefore, in a more concentrated condition than was the iron and steel industry at the date of formation of the Steel Corporation, but further concentration, to such a point as to bring about unfairly high prices, does not seem practicable.

"Figures obtained unofficially from various sources would indicate that the total authorized capitalization of all the American portland cement companies now in operation will fall between the limits of $110,000,000 and $125,000,000. This estimate takes no account of outstanding bond issues, but on the other hand gives no credit for stock authorized but not issued, so that on the whole it can be accepted as representing very fairly the total capitalization of the American portland cement industry. This capitalization cannot be considered excessive, in view of the fact that it would cost probably $75,000,000 to $85,000,000 to replace the plants and properties now in existence."

Certainly these figures, coupled with those gathered by the Geological Survey for 1906, and also by THE MINERAL INDUSTRY, emphasize the fact that the production of portland cement is an industry capable of development in almost any of the States of the Union, and that the danger of a cement trust is one of the myths that ought to be relegated to the back ages.

New Plants.—In connection with the development of the industry in 1906, so far as geographical distribution is concerned, the first and most noticeable feature is the increased production in western states where, the supply of lumber being curtailed and the prices having advanced, an enormous demand for cement has sprung up. In Indiana there was a marked increase in the vicinity of Mitchell; new works were also brought into

being in Kentucky, near Louisville, and at Cosmos, on the Ohio river, while further west additional construction in the Chicago district by the Illinois Steel Company, at Gary, and by several of the works at Lasalle, added materially to the capacity. In Michigan many of the old plants have been rebuilt and several new plants using rock materials have been completed. The Labor Bureau of that State reports 17 plants, with an investment of $8,300,000 and an output of 4,032,000 bbl., in operation. New works at Dixon, Ill., increases at Yankton, South Dakota, and large projected developments in the vicinity of Mason City, Iowa, form stepping stones to the enormous increase in the Iola, Kansas, field, which by figures recently compiled show a capacity estimated at nearly 6,000,000 bbl. per annum. Further west, Colorado increased her portland cement capacity, as did Utah, and projected works near Butte, Mont., and Seattle, Wash., blazed the way to the California coast. The San Francisco disaster found the cement industry on the Pacific coast at almost the hight of its prosperity, the great demand from the Government for cement for its irrigation projects having more than exhausted the supply of the Pacific field. This led to increases at the two existing mills on the coast—the Standard and the Pacific—and to the development of several new enterprises, all of which are under way, at Santa Cruz, San Juan, Sky-Blue and Colton, with the probable result that California will, within a few years, become an exporter of portland cement. In the State of New York two of the plants on the Hudson are increasing, another is substituting rotary kilns for the old style dome kilns, and still another is rebuilding at its marl and clay plant. In the southern States several new plants have been constructed to meet the increasing demands of that section of the country.

Materials.—Among the noticeable features during 1906 was the substitution of dry materials for wet by several of the mills in the Michigan region, limestones and shale being substituted for the marl and clay used in the wet process. Considerable attention has been drawn to the deposits of limestone under the Tombigbee river at Demopolis, Ala., where English owners have had a mill for several years. These soft limestones are found in many parts of the United States and seem to extend in a broad belt—almost northwest—for many hundreds of miles through many of the western States.

Machinery.—In reference to machinery, it may be stated that the use of the Fuller mill for coal grinding has become general in the East, excellent results being claimed. The introduction of this mill, as a substitute for the Griffin and ball and tube mills for the grinding of raw materials, has also made some progress, while in an experimental way it has been used on clinker grinding. New mill construction has shown very few changes from the previous year, except in the line above described. The Griffin

mill and the ball and tube mill installations have gone into general use in a large proportion of the new plants. During the early part of 1906 a number of large kilns were installed, some running as high as 140 ft. in length, and figures for the first time were obtained on this form of construction. While satisfactory results were obtained, in many cases it was found that the long kilns had one disadvantage, namely, that when one of them was out of use by reason of repairs or otherwise, it lessened production to a much greater extent than where kilns of shorter lengths were used. Experience with the long kilns, however, especially in some of the larger plants, has shown very excellent results.

The Engineering Investigations.

The year 1906 witnessed the general adoption, practically, by prominent engineers all over the country of the standard form of specification adopted by the American Society for Testing Materials under the rules of the American Society of Civil Engineers. Many thousands of these specifications were sent out by the American Society for Testing Materials to all parts of the world, and practically almost every important piece of work, under the supervision of leading engineers, was under specifications falling within the lines of the standards adopted.

The appointment by President Roosevelt of a National Advisory Board on Fuels and Structural Materials had an important bearing on the standardization of methods for concrete and reinforced concrete. During the session of Congress in 1906 there was appropriated to this Board the sum of $100,000 for experiments to be made by it at the Government laboratory at St. Louis. This enabled the Government expert in charge, Richard L. Humphrey, to employ not less than one hundred men continuously in the making of various tests of cements, sands, aggregates and of beams of different forms. The work of the laboratory, which it is stated is going on constantly, has so far produced more beams than have ever, up to this date, been made in all the experimental work done in the United States. This work is done at the Government's expense, under the general supervision of the National Advisory Board on Fuels and Structural Materials, above mentioned, and upon lines suggested by the Committee on Tests of the Joint Committee on Concrete and Reinforced Concrete. This important engineering body, which it is expected in the course of a year or more will be able to furnish to the world a specification (based upon the largest amount of experimenting hitherto done) for structures of all kinds in concrete and reinforced concrete, held many meetings during 1906, and will, later on, have many bulletins to issue upon experiments so far conducted. It has also collated from many of the college laboratories the work done in connection with the programme adopted. It is to this coöperation of the American Society for Testing Materials

in the making of specifications, the American Society of Civil Engineers in the formulation of methods of manipulation and of the Committee on Concrete and Reinforced Concrete, which last body is composed of members representing the American Society for Testing Materials, the American Society of Civil Engineers, the American Railway Engineering and Maintenance of Way Association, and the Association of American Portland Cement Manufacturers, all working in unison with the Government through its Advisory Board on Fuels and Structural Materials, that cement engineering and cement construction in the United States owes its great development, and with the final adoption of standard specifications for concrete and reinforced concrete by the bodies having to do with this subject, it is expected that (with that large spirit of exactitude and progress which animates American municipalities) all the large cities, through their building departments, will adopt these specifications and the construction of work in concrete will become as standardized, as uniform and as regular as work hitherto done in iron, steel, stone or brick.

Another interesting subject in this connection was the work of the National Association of Cement Users, whose convention in January in Chicago was a remarkable success in the large number of exhibits of cement block machinery, concrete mixing machinery and other kindred classes of machinery, and whose papers and deliberations were signalized by the greatest intelligence and learning.

State associations on similar lines have been formed in Iowa, Nebraska, Minnesota and many of the other western States, and are doing much good work in the dissemination of knowledge as to the uses of cement and proper methods of construction.

"Universal" Portland Cement.

The Universal Portland Cement Company, a new subsidiary company of the United States Steel Corporation, took over the plants and business of the cement department of the Illinois Steel Company, Oct. 1, 1906, and will continue the manufacture of "Universal" portland cement. The present output of 6500 bbl. a day at South Chicago, Ill., and Buffington, Ind., is being increased by a new plant at Buffington, with a capacity of 6000 bbl. a day and a plant at Universal, Penn., near Pittsburgh, to have a capacity of 4500 bbl. a day. When the new plants are in operation in the summer of 1907 the total output of the company will be 17,000 bbl. a day. The cement manufactured in the new mills as well as the old will be made from slag and limestone by the same special process and under the same supervision which has been employed since 1900. Both the new plants will be driven entirely by electric power from generators connected to gas engines using blast-furnace gases. The total production of slag cement by the Universal Portland Cement Company in 1906 was 2,076,000 bbl.

The Cement Industry in Foreign Countries.

By John T. Glidden.

Belgium.—There are about 35 cement works in Belgium producing an annual output of about 1,000,000 tons. This output is about equally divided between home consumption and export trade. The present selling price of cement is from 84c., to $1.04 per bbl. of 396 lb., gross weight, according to quality, cost of barrel included, f.o.b. Antwerp. The price of portland cement for home consumption is from $3.66 to $3.86 at works, per ton of 2204.6 lb. Barrels and canvas bags only are employed for shipping the cement. The works will repurchase the sacks but not the barrels. It is reported that an agreement was entered into between a German syndicate and 12 Belgian cement works to the following effect: The German syndicate agreed not to sell its cement in Belgium, and the 12 Belgian cement works pledged themselves not to sell their product in Germany; and to prevent competition the German syndicate is to confine its sales in Holland to towns lying near the German frontier and reached at a less cost than could be reached by the Belgian concerns, and vice versa as concerned towns in Holland nearer to the Belgian frontier and more advantageously accessible to the Belgians.

Canada.—The total quantity of portland cement made in Canada in 1906 was 2,152,562 bbl., as compared with 1,541,568 bbl. in 1905. The total sales were 2,119,764 bbl., as compared with 1,346,548 bbl. in 1905. Fifteen companies were operating plants during 1906, with a total daily capacity of about 10,500 bbl., viz.: one in Nova Scotia, two in Quebec, eleven in Ontario, and one in British Columbia. At least four plants were

STATISTICS OF CEMENT IN CANADA.

Year.	Production (*a*).				Imports (*b*).			
	Natural Rock.		Portland.		Natural Rock.		Portland.	
	Bbl.	$	Bbl.	$	Bbl.(*c*)	$	Bbl. (*c*)	$
1900	125,428	$99,994	292,124	$ 562,916	3,742	$ 4,711	342,463	$ 498,607
1901	133,328	94,415	317,066	565,615	4,680	6,865	424,324	654,595
1902	127,931	98,932	594,594	1,028,618	7,786	17,755	518,846	833,675
1903	92,252	74,655	627,741	1,150,592	3,603	6,333	609,700	868,131
1904	56,814	50,247	900,358	1,272,992	3,181	5,391	651,681	995,017
1905	14,184	10,274	948,274	1,346,548	4,463	10,690	849,577	1,234,649
1906	8,610	6,052	2,139,164	3,164,807	2,840	4,034	753,004	998,988

(*a*) Sales. (*b*) Fiscal year ending June 30. (*c*) Barrels of 380 lb. Exports of Canadian manufacture were valued at : $3,296 in 1900; $1,514 in 1901; $2,267 in 1902; $2,851 in 1903; $3,205 in 1904; $5,430 in 1905; $8,913 in 1906.

under construction during the year, of which the total initial daily capacity will be about 4700 bbl. Among the new plants now under construction are the works of the Fordwick company, to be located on the outskirts of Montreal, and to have a capacity of 800,000 bbl. per year. This plant will be entirely electric driven. The property of the company is

about two miles in length by 600 ft. in width and borders along the St. Lawrence river. The Commercial Cement Company is making active preparations to begin the manufacture of cement at Carman, about 70 miles southwest of Winnipeg. The rock which supplies the raw material is procured at a considerable hight above the mill and is transported through the various processes of manufacture almost entirely by gravity.

Detailed statistics of production in 1905 and 1906 are as follows:

	1905.	1906.
Portland cement sold	1,346,548 bbl.	2,119,764 bbl.
Portland cement manufac'd	1,541,568 bbl.	2,152,562 bbl.
Stock on hand Jan. 1	111,446 bbl.	269,558 bbl.
Stock on hand Dec. 31	306,466 bbl.	302,356 bbl.
Value of cement sold	$1,913 740	$3,164,807

The average price per bbl. at the works in 1906 was $1.49, as compared with $1.42 in 1905.

The imports of portland cement into Canada in 1906 were 2,430,760 cwt., valued at $778,706. This is equivalent to 694,503 bbl. of 350 lb. each, at an average price per barrel of $1.12. The duty is 12½c. per 100 lb. The imports in 1905 were equivalent to 917,558 bbl., valued at $1,138,548, or an average price per barrel of $1.24. There is very little cement exported from Canada. The consumption is therefore practically represented by the Canadian sales, together with the imports.

France.—According to Consul General Skinner of Marseilles any contemplated effort to obtain a foothold for American cement in France might better be devoted at the present time to an effort to sell cement in those remote markets to which large quantities of French cement are now being exported. There are two principal producing centers in France, one being in the northern part of the country, and the other lying in the district around and about Marseilles. All shipments of domestic cement are in jute bags which cost about 19c. each and are returned to the manufacturers. The cost of the bags is not included in the prices for the cement. The best French cement sells for approximately $6.95 per metric ton at the factory. The largest cement manufacturer is the concern known as the Société J. & A. Parvin de Lafarge, which exports practically all of the cement now going from Marseilles to the United States.

Germany.—Germany has in all 320 cement mills, in 117 of which portland cement is manufactured. The output of portland cement from factories in the syndicate in 1904 was 27,950,000 bbl., which does not represent the entire German output. German cement ranks very high in respect to quality, but severe competition and the pressure of outside

factories are so great as to keep the price down to a very close margin of profit and to render the position of the large manufacturers one of great difficulty. It is reported that cement is sometimes sold in Berlin at a profit of only 11.9c. (50 pfennigs) per barrel to the manufacturer. During 1905 prices in Berlin averaged from $1.14 to $1.19 per bbl., at the building, including the barrel. In the price of cement, bags are usually valued by the manufacturers at 4.76c. each, and barrels at 5.95c., and are taken back at the same price, free of charge, if they are returned in good condition within six weeks after being used. Toward the end of November, 1905, the different German cement associations made an agreement whereby middlemen were eliminated from trade. Much complaint arose, and the union of German dealers in building materials, which includes 700 firms, with an annual consumption of 1,300,000 tons of cement, planned the organization of a buying syndicate, and demanded from the cement manufacturers an extra rebate of from 6 to 10 per cent. for members of the union. The syndicated factories refused to comply with this demand and union dealers are now discussing the advisability of erecting their own cement factories but up to the present no actual step has been taken.

Great Britain.—The great cement producing centers, especially those in England and in Germany, which were developed rapidly on the strength not merely of their command of their home market, but also on their reliance on a great export trade, found themselves gradually opposed in those countries to which they were accustomed to export cement, by a growing local manufacture, and in consequence had to be content with a non-progressive or diminishing export, or were obliged to lower their prices in foreign markets to a point that was barely remunerative. This natural shifting of the center of gravity of the cement industry, if it had taken place gradually, would have inflicted little injury to those manufactories whose position was economically sound; but it happened that the largest single customer for exported cement—the United States—took up and developed its home manufacture on so huge a scale and with such rapidity that the European works found themselves deprived of an outlet for a large part of their product with inconvenient celerity, losing a valuable consumer before they had time to find or create other directions in which their cement might be profitably disposed of.

EXPORTS OF BRITISH CEMENTS, 1904 AND 1905.

South Africa, tons	83,629	85,019
India	87,685	97,864
New Zealand	24,805	28,351
Australia	16,292	15,432
Canada	15,561	28,303
Argentina	15,014	17,688
Brazil	6,445	13,919
United States	5,073	11,590
Netherlands	1,925	5,319

Consul W. C. Hamm, at Hull, reported recently: About 4000 tons of limestone are brought to Hull each week, and the production of cement is about 5000 tons per week. The supply could be increased under pressure. The cost of the barrels is larger here than in America and adds quite materially to the cost of the cement. Cement in bags costs 25c. extra per bag, but on the return of the empty bags most of this charge is refunded. Since the San Francisco earthquake the demand for cement has been exceptionally large, and it is only with difficulty that orders can now be executed, as supplies are running short.

Russia.—Statistics and facts concerning the Russian cement industry are difficult to obtain on account of unsettled conditions in that country and the most recent statistics give information only up to 1903. At the beginning of the present century there were in operation 35 different cement works in Russia, pretty evenly distributed over the entire country. Russia exports considerable cement, and the imports are falling off, which indicates a relatively small consumption in the Empire.

RUSSIA.—EXPORTS AND IMPORTS OF CEMENT. (*a*)
(In pounds.)

Year.	Exports.	Imports		
		Portland	Other Brand.	Total.
1898......	5,097,360	50,845,696	39,795,424	90,641,120
1899......	7,222,400	63,809,904	30,947,984	94,757,888
1900......	8,305,760	27,553,456	33,403,600	70,957,056
1901......	15,672,508	11,736,400	25,025,616	36,762,016
1902......	4,297,328	14,914,256	23,292,240	38,206,496
1903......	24,339,488	7,439,072	23,075,568	30,514,640

(*a*) From *Wjestu. Finanzow*," through "*Chem. Zeit.*" Jan. 26, 1907.

Fuel Efficiency of Cement Kilns.[1]

The first rotary kilns used in America for cement burning were 30 ft. long. This length was increased to 40 ft., and finally to 60 ft. for kilns burning dry materials and to 90 ft. for kilns burning wet materials, and at these lengths the advance stopped for a number of years. The great majority of American cement plants are today equipped with 60-ft. kilns 6 and 7 ft. in diameter. In 1904 Thomas A. Edison installed and put in operation at the Edison cement works two kilns 150 ft. long and 9 ft. in diameter. The result of the experiment was successful and cement works engineers, following Edison's lead, began to increase the size of their kilns. None of them felt quite inclined to make the jump from 60 ft. to 150 ft., that he had made, but kilns 100 ft. long, 125 ft. long, and, now 136 ft. long, began to be designed and installed with increasing frequency. It is natural to inquire why this change of practice should come about.

[1]Abstract of paper in *Eng. News.*

The rotary cement kiln is not an economical thermal device. It is very wasteful of fuel; its economy and efficiency as a means for calcining cement mixtures to clinker comes from the economy of handling material and not from high efficiency; it requires about 40 ℔. of coal to burn one barrel of cement with a vertical kiln, while it requires 100 to 200 ℔. of coal to burn one barrel of cement in a rotary kiln. In the case of the vertical kiln the raw material, after being reduced to powder, has to be molded into blocks, which are charged by hand into the kiln, and finally, when burning is completed, the blocks have to be sorted by hand to remove those imperfectly burned, and then crushed before the clinker can be admitted to the grinding machines. The rotary kiln takes the powdered raw material, dry or wet, by mechanical feed and burns it to clinkers varying from the size of a pea to the size of a walnut; it can then be fed directly to the grinding machines; there is no molding of blocks involving preliminary drying with wet mixtures and preliminary wetting with dry mixtures, no sorting out and discarding of underburned material, and no preliminary crushing of the clinker. It is in the saving effected in handling the material and not in its efficiency as a furnace, that the economy of the rotary cement kiln exists.

Several devices have been suggested for improving the thermal efficiency of the rotary cement kiln. It has been proposed to use two jets of coal dust; to water-jacket the hottest zone as is done in blast furnaces; to jacket the kiln to reduce loss of heat by radiation; to introduce interior ribs to retard the movement of the material, and to do various other things. None of these remedies has passed beyond an individual installation and many of them exist only on paper. In fact, the only method suggested for improving kiln efficiency that has been taken up in actual practice is that of lengthening the kiln cylinder. This method is gaining continually increasing favor.

In a general way it is evident that by making the kiln longer the gases of combustion are compelled to leave the kiln at a lower temperature, and hence, a part of the heat now wasted in the shorter kilns is rendered useful, but it is a question as to how much the gain thus made amounts.

Taking the few and incomplete data at hand, we find a notable economy in fuel resulting from the use of long kilns. When Edison's kilns 150 ft. long and 9 ft. in diameter were first installed the managers of the works stated that the monthly average of each kiln was 30 bbl. of cement per hour with a coal consumption of 80 ℔. per barrel. The output of 60-ft. kilns working on practically the same material is from 7 to 10 bbl. per hour with a coal consumption of 80 to 140 ℔. per bbl. Referring to the Edison kiln, A. V. Bleininger states that its fuel consumption probably approaches quite closely to 50 ℔. per barrel. In a recent discussion of rotary kiln efficiency the German authority, W. Michaelis, Jr., states that rotary

kilns 90-ft. long are in use in Germany which (burning dry materials) turn out 300 bbl. per day with a fuel consumption of from 60 to 75 lb. of coal per bbl. This economy is not due alone to the increased kiln length, however, but is brought about in part by supplementary devices for regulating the admission of raw materials, fuel and air to the kiln.

Figures of actual cement output and actual fuel consumption collected by E. C. Eckel and published in his "Cements, Limes and Plasters," show that 60-ft. kilns working on cement rock and limestone may be expected to produce from 180 to 250 bbl. of cement per kiln with a fuel consumption of from 85 to 160 lb. of powdered coal per bbl., while 80-ft. kilns working on the same material may be expected to produce 225 to 300 bbl. per day per kiln, with a fuel consumption of from 85 to 120 lb. of coal per bbl. The same authority states that the 150-ft. kilns of the Edison works turn out 375 bbl. of cement per kiln per day with the fuel consumption of 65 lb. of coal per bbl., and this with practically the same material as gives the figures quoted previously for 60- and 80-ft. kilns. This material—the cement rock and limestone mixture of the Lehigh cement district—is noted, however, for its remarkably easy clinkering. A 60-ft. kiln working on limestone and clay, will on the average, record neither so great an output nor so small a fuel consumption.

One plant tested uses a wet mixture containing 52 per cent. water and burns it with a local coal having a heat value of 10,094 B.t.u. With its 100-ft. kiln it gives an output of 166 bbl. of cement per kiln per day with a fuel consumption of 180 lb. of coal per bbl. These are very satisfactory results, as a brief examination will demonstrate.

E. C. Soper, who reports this test, figures from his records that there is a saving of 48 lb. of coal per bbl. of cement as compared with a 60-ft. kiln working under the same conditions. E. C. Eckel (in the treatise quoted) gives a record of a 100-ft. kiln burning a wet mixture that produces 135 bbl. of cement per day with a fuel consumption of 150 lb. of coal per bbl. Using the same material he gives the following figures as the results that may be expected from the kilns of the lengths more commonly used: For 60-ft. kilns an output of 60 to 140 bbl., with a fuel consumption of 150 to 250 lb. per bbl.; for 80 to 90-ft. kilns an output of 80 to 150 bbl., with a fuel consumption of 140 to 220 lb. per barrel.

Though these records of long kiln efficiency are of rather haphazard character, it will be observed that they all agree in showing a decided gain both in output and in fuel economy resulting from the increase in length. This gain is obtained whether the kiln is working on a dry mixture or a slurry, though naturally it may be expected to be smaller with the wet mixtures. In view of these facts it is apparent why cement mill engineers should, after having apparently settled down to the use of 60-ft. kilns, suddenly begin to build kilns twice that length; it is quite certain that in

the future few new cement plants will be fitted with kilns less than 100 ft. long.

It may not be supposed, however, that the solution of the problem of rotary kiln fuel economy afforded by lengthening the cylinder lacks objections. The cost of the kiln plant is in the first place greatly increased; this is true even in new plants where everything is planned on the basis of long kilns. In many existing plants lengthening the kiln necessitates the reconstruction of a large part of the remaining plant and buildings to give room for the larger kiln units. It may readily occur, therefore, that some method of utilizing or regenerating the heat wasted from the kiln will be a far more economical means of reducing the losses now suffered than will the introduction of long kilns. Practically speaking, only two methods of utilizing waste kiln heat in cement mills have been seriously taken up in America. One method is to trap the hot gases coming from the kiln and pass them under boilers for generating steam; the other method is to utilize the heat from the cooling clinkers to preheat the air used in combustion in the kiln. Papers by A. B. Helbig in *Eng. News* (Feb.16,1905), and by R. C. Carpenter in the Sibley *Journal of Engineering* (March, 1904), give further details regarding each of these methods.

Bibliography of Literature—New Patents.

Cement is exceptionally well treated in every volume of The Mineral Industry, each article giving a review of the general condition of the industry in the United States, together with statistics of production, imports, exports and consumption. Special features are as follows: In Vol. V, a 25-page article upon the manufacture of cement in Great Britain. This article gives the classification of cement and the raw materials used in its manufacture, discusses principles of manufacture, the kinds of material used, the quality of cement obtained, the methods of testing, analyzing and of adulterating cement. Vol. VI contains a 28-page article by F. H. Lewis, dealing with the manufacture of hydraulic cement in the United States. This article gives a brief account of the historical and statistical aspects of cement, geological and chemical features, and discusses in great detail the processes of manufacture and the quality of the product. Vol. X contains an 11-page article by Edwin C. Eckel upon the slag cement industry in the United States in 1901. This article discusses the necessary physical condition of the slag, the proportions of lime and slag to be used, the composition of the lime, and the chemical composition of slag cements in general. A complete description of various manufacturing methods is given, and the article is illustrated by cuts of furnaces, grinders and dryers used. In Vol. XI there is a 32-page article by F. H. Lewis discussing the mechanical equipment of a modern portland cement plant. This article is very completely illustrated by cuts and drawings

of the various appliances used and has several sections of crushing plants and kilns, etc. Vol. XIII discusses the cement industry in foreign countries, reviews the technical progress in cement manufacture during 1904, gives a summary of standard specifications for cement, outlines some procedures for analytical methods as applied to cement, and discusses the preparation and employment of portland cement concrete.

Following are some recent articles, relating to the production and manufacture of cement, which have appeared in the technical and scientific press. The list is admittedly incomplete; a thorough summary of the literature of this important industry would require a volume by itself; however, it is conceived that these scattering references will be useful.

"Wie wird bei Anlage eines Kalksandsteinwerkes die Herstellung guter Kalksandsteine gewährleistet?" W. Drakebush. (*Tonind Zeit.* Jan. 10, 1907; 1 p.) A discussion of the factors which must be investigated and established in order to insure the profitable exploiting of a lime and marl deposit.

"Cement Resources of the Cumberland Gap District, Tenn.-Va." E. C. Eckel. (*U. S. Geol. Surv., Bull. No.* 285, 1906; 2½ pp.) Describes the rocks suitable for the manufacture of cement in eastern Tennessee.

"Cement Possibilities of Alabama." R. C. Lane. (*Mfrs.' Rec.*, Nov. 8, 1906). Draws comparisons between the cement and the iron industries in the United States, and calls attention to the great possibilities of the South as a manufacturing center for cement.

"Cement Industry of Ontario." (*Report of Ont. Bureau of Mines*, Vol. XIV., Part 1; 66 pp.) Describes briefly the marl deposits of Ontario, and discusses in considerable detail the apparatus used in making cement. Also enumerates the principal cement producers of the province.

"Recent Progress in the Cement Industry." Bertram Blount. (*Journ. Soc. Chem. Ind.* Nov. 15, 1906; 12 pp.) Outlines the old and the new methods of making cement, with a discussion of the chemistry involved, and statistics on the production of portland cement.

"The Industrial Relations of the American Cement Industry." E. C. Eckel. (*Eng. Mag.*, Jan., 1907; 8 pp.) Discusses the expansion of the cement industry, its concentration, and the influences of the Steel Corporation upon the cement industry as a whole.

"The Manufacture of Portland Cement." (*Engineering*, July 20, 1906; 4½ pp.) A description, well illustrated, of the Northfleet Works of the Associated Portland Cement Manufacturers of England, which was visited by the members of the Amer. Inst. of Min. Engrs.

"Electrical Equipment of the Bath Portland Cement Co., Bath, Pa." (*Elec. Rev.*, Nov. 17, 1906; 4 pp.) Describes the various engines, generators, boilers, etc., that are in use at this plant. The system of lighting and of crushing and conveying fuel to the boilers is also described.

"Beiträge zur Kenntnis des Sinterungsverlaufes der Portlandzementrohmasse." L. Jesser. (*Tonind. Zeit.*, Nov. 17, 1906.) A study on the burning of portland cement, with reference to heat reactions.

"Fabbricazione moderna del cemento Portland. L. Fabre." (*Rassegna Mineraria*, Nov. 24 and Dec. 1, 1906; 5 pp.) Illustrated description of various plants for making cement.

"Cement for Sea-Water Construction." S. B. Newberry. (*Cement Age*, Jan., 1907; 4 pp.) Brief description of the chemical action of sea-water upon cement, and a discussion of the production of a new cement which shall be unaffected by salt water. Gives analyses of and methods of working the new "ore cement" designed to be used under these new conditions.

"Darf man Kalkmergel mit Schwefelsäure Titrieren?" Dr Schulze. (*Chem.-Zeit.*, Sept. 26, 1906.) Comparison of different methods of analyzing calcareous marl, with reference to its suitability for making portland cement.

"The Chemistry of Hydraulic Cements." J. Waddell. (*Can. Engr.*, Dec., 1906.) Discusses the chemistry involved in the production and the use of cement.

The following patents relating to the manufacture of cement were issued in 1906:

Hydraulic cement and process of making the same. Carle D. Clark, Clifton, Ariz. U. S. pat. 811,902.

Oscillating cement-kiln. Byron E. Eldred, Bronxville, N. Y. U. S. pat. 812,834.

Cement kiln. Thomas A. Edison, Llewellyn Park, N. J. U. S. pat. 813,490.

Cement-burning process and apparatus therefor. Byron E. Eldred, Brookline, Mass. U. S. pat. 813,627.

Heat-regenerator for cement-kilns. Carleton Ellis, New York, N. Y. U. S. pat. 813,630.

Manufacture of hydraulic cement and apparatus therefor. Bernard Enright, Fordwick, Va. U. S. pat. 815,080.

Apparatus for making cement. Rolla C. Carpenter, Ithaca, N. Y. U. S. pat. 815,680.

Rotary cement-kiln. Harry Hitzel, Alpha, N. J. U. S. pat. 815,705.

Slag cement and method of making the same. Joseph A. Shinn, Pittsburgh, Pa. U. S. pat. 816,389.

Apparatus for burning lime. Carleton Ellis, White Plains, N. Y. U. S. pat. 821,996.

Apparatus for manufacturing cement.—Carleton Ellis, New York, N. Y., U. S. pat. 825,305.

Process of burning cement clinker by flame impingement and apparatus therefor. Carleton Ellis, White Plains, N. Y. U. S. pat. 827,517.

Process of burning cement. Byron E. Eldred, New York, N. Y. U. S. pat. 829,956.

Apparatus for making cement clinker. Byron E. Eldred, Bronxville, N. Y. U. S. pat. 829,957.

Apparatus for producing cement. Rolla C. Carpenter, Ithaca, N. Y. U. S. pat. 833,918.

Production of cement. Bernhard Grau, Kratzwieck, near Stettin, Germany. U. S. pat. 813,965.

Lime kiln. The Chalk Power Gas Syndicate, London. A lime kiln so arranged that carbon monoxide shall be given off instead of carbonic acid, and the monoxide subsequently used as a fuel. Brit. pat. 2684 of 1905.

Slag cement. H. Collosens, Berlin. Adding nitrates or sulphates of lime, alumina and magnesia to molten blast-furnace slag, so as to produce a cement that sets more rapidly than the usual slag cement. Brit. pat. 3619 of 1905.

Slag cement. B. Bruhn, Hamburg. In order to increase the hydraulicity of cements made from blast furnace slag, adding small quantities of alkaline carbonates to the slag while still molten. Brit. pat. 8285 of 1905.

CHROMIUM AND CHROME ORE.

Although there is a considerable demand for chrome ore in the United States, the domestic production is inconsiderable and comes solely from California where it is used almost entirely as a refractory lining for furnaces, or is made into chrome brick for the same purpose. The large consumption of ore in the East for manufacture of chromium salts and ferro-chrome alloys is supplied by imports, duty free, from New Caledonia, Greece, Canada and Turkey. The following table shows the relative importance of the domestic ore production in comparison with the consumption:

STATISTICS OF CHROME ORE IN THE UNITED STATES.
(In tons of 2240 lb.)

Year.	Production. (*a*)			Imports.			Consumption.	
	Long Tons.	Value.	Value per Ton.	Long Tons.	Value.	Value per Ton.	Long Tons.	Value.
1897	*Nil.*	......	......	11,566	$186,313	$16.11	11,566	$186,313
1898	*Nil.*	......	......	16,304	272,234	16.70	16,304	272,234
1899	*Nil.*	......	......	15,793	284,825	18.03	15,793	284,825
1900	140	$1,400	$10.00	17,542	305,001	17.39	17,682	306,401
1901	130	1,950	15.00	20,112	363,108	18.05	20,242	365,058
1902	315	4,725	15.00	39,570	582,597	14.73	39,885	587,322
1903	150	2,250	15.00	22,931	302,025	13.13	23,081	304,275
1904	123	1,845	15.00	24,227	348,527	14.38	24,350	350.372
1905	40	600	15.00	54,434	725,301	13.32	54,874	725,901
1906	180	1,800	10.00	43,441	557,594	12.84	43,621	559 394

(*a*) Reported by the California State Mining Bureau except for 1906. (*b*) Estimated.

The expected development of a chrome ore industry in the serpentine area of the Southern States, especially in North Carolina, did not take place in 1906, the reason being that the extent of the deposits is not as great as was hoped for, and also because they are too remote from railroad transportation. In California the chief production was from Camp Seco in Calaveras county, and from the chrome mines near Castle Crags, Shasta county, owned by L. H. Brown and Chas. Dougherty, of Dunsmuir. Other mines which were producers in the past are not idle because of exhaustion, but are only awaiting a more active demand for their product.

Market and Prices.—Quotations on chrome ore during 1906 were very regular, being $17.25 @ $19.75 per long ton for 50 per cent. ore at New York, which is a small advance over the prices in 1905. Nothing new is to be noted in market conditions. The chief consumers of chrome ore for manufacturing purposes are the Kalion Chemical Company of Philadelphia, and the Baltimore Chrome Works of Baltimore; the Harbison-Walker Refractories Company of Pittsburg controls the chrome-brick industry.

Chrome Ore in Foreign Countries.

Canada.[1]—Chrome ore is found in large quantities in the serpentine belt in the eastern townships of Quebec, between Disraeli and Broughton, in the hills to the southeast of the Quebec Central Railway, but it is worked commercially only in the Black Lake district. Until recent years no serious operations had been undertaken for its production.

There are several mines through the Black Lake district that are being successfully developed, and one, that of the Black Lake Chrome and Asbestos Company, has gone beyond the experimental stage and has become an important factor in the chrome trade of this continent. The Black Lake No. 1 mine has a lense of ore, which was 4 ft. wide and 3 ft. thick at the surface, and at the depth of 350 ft. is 175 ft. wide and 30 ft. thick. The company has a mill of 30 stamps for the concentration of the waste from its pits, and is producing a high-class chrome concentrate running 50 to 54 per cent. Cr_2O_3, which is coming into direct competition with the high-grade Caledonia ores. Indeed all the chrome treated in this way through the district is of high quality.

The method of mining and treating the ore is the same throughout the district and is very simple. After it comes from the pit it is sorted and all that will run over 40 per cent. Cr_2O_3 is put aside as crude. This product is divided into two grades, all below 47 per cent. being No. 2 and all above it No. 1. The remainder, or all ore testing below 40 per cent. and over 10 per cent., goes to the mill. Here it goes through a jaw crusher and under the stamps. After it will pass through a 22-mesh screen it goes to Wilfley tables; the best of the tailings from the tables are run to another table and the resulting concentrate is called No. 2, having 40 to 43 per cent. Cr_2O_3.

New Caledonia.—The chrome deposits of New Caledonia are of two kinds[2]: Some are in vast superficial masses of ferruginous clay and contain very rich ore, soft and easy to separate; others are in form of masses enclosed in the hard serpentine. L. Bernheim founded the "Société Le Chrôme," with a capital of 3,800,000 francs, to develop this industry. Today the company owns a mining estate of 60,000 hectares, comprising 6000 hectares of final concessions, 6000 hectares applied for, and 48,000 hectares prospecting areas. This estate consists of three chief groups; one is in the north of the island, and includes the Tiebaghi mine, which is the largest and richest chrome mine known, and which is leased to a London firm. This mine contains a very fine bed, giving ore with as much as 57 per cent. sesquioxide of chromium; production, 4000 to 5000 tons of ore per month. A second group lies 20 km. south of Noumea and comprises the "Lucky Hit," where a vein with maximum thickness of 15 m. is

[1]W. H. Edwards, *Journ.* Can. Min. Inst. Vol. IX.
[2]Notes from an article in *Comptes Rendus* de la Société de l'Industrie Minerale, August, 1906.

mined. The ore, giving 30 to 40 per cent. of chromium sesquioxide, is enriched in a special works, and the mine, the only chrome underground mine in the island, can now produce 12,000 tons annually. A third group in the south is not yet mined. A railway is being constructed to enable work later.

The workings of the Tiebaghi mine are situated on the top of Mount Tiebaghi, some 1800 ft. above sea level. The ore is brought down by overhead cables, at the rate of 200 tons daily, to the tramway. This latter is five miles in length. Punts then are employed to take the ore to the ships in the harbor, another four miles. When there are no ships awaiting loading, the chrome is stored on an island close to the anchorage, a reserve of 15,000 tons being always on hand. The labor employed is a mixture of French, Japanese, Tonquinese, Chinese, Arabs, Indians, Javanese, and Kanakas. The pay-roll amounts to about £500 a week.

Rhodesia.—Chrome ore is now produced from large deposits in the neighborhood of Selukwe. It is shipped to Great Britain and the Continent via Beira, and realizes £3 2s 6d per ton.

Turkey.—Competition from New Caledonia and adverse local conditions have brought about a crisis in the chrome ore industry in Turkey, and the importance of this region is much diminished. Formerly an average of about 40,000 metric tons were exported annually, but now the amount has decreased to a few thousand tons only per year. The transportation of chromite from New Caledonia to English and American ports is comparatively inexpensive, as it is shipped largely as ballast, and this fact makes it hard for Turkey to compete against the prices at which this ore can be sold. Moreover, poor roads within the country make transportation to shipping ports high and taxes are burdensome. The taxes are not levied on the value of the mineral as mined, but upon its value after it has had added to its cost of production the high hauling cost.

THE PRINCIPAL SUPPLIES OF CHROME ORE. (*a*)
(In metric tons.)

	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905	1906
Bosnia........	443	396	458	200	100	505	270	147	279	(*c*)	(*c*)
Canada.......	2,125	2,393	1,,834	1,824	2,119	1,156	817	3,184	5,512	7,781	7,936
Greece........	1,600	563	1,367	4,386	5,600	4,580	11,680	8,478	15,430	(*c*)	(*c*)
New Caledonia (*b*).........	16,018	9,054	14,300	12,480	10,474	17,451	10,281	21,437	42,197	51,374	57 367
Newfoundland	1,031	3,084	657	717	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	(*c*)
New South Wales.......	3,914	3,433	2,145	5,327	3,338	2,523	454	1,982	403	53	*Nil.*
Norway.......	*Nil.*	*Nil.*	*Nil.*	41	165	85	22	*Nil.*	154	*Nil.*	(*c*)
Russia........	6,682	13,433	15,467	19,146	18,233	22,169	19,655	(*c*)	(*c*)	(*c*)	(*c*)
United States.	799	*Nil*	*Nil.*	*Nil.*	142	132	320	152	125	122	183

(*a*) From the official reports of the respective countries. No statistics are available from Turkey. (*b*) Exports. (*c*) Statistics not yet available.

Technology and Uses.

Chromite is the source of the chromium salts which are extensively used in dyeing and tanning, and whose manufacture requires the largest amount of ore. The manufacture of ferro-chrome, chrome brick and other refractory materials consume smaller amounts. The appreciation of the hardness and toughness that small amounts of chromium impart to steel has increased the demand for these special steels which are now largely used for the manufacture of machinery castings, tools, stamps, etc., in which great resistance to shock and abrasion is required. From 0.5 to 2 per cent. of chromium, in connection with about 1 per cent. of carbon, gives a tool extraordinary hardness. In self-hardening steel, chromium is usually used to the extent of 0.25 to 3.5 per cent., together with tungsten. Chromium has great tendency toward the absorption of carbon, and to this fact is due its use in the manufacture of armor plate in which great hardness is required. The ferro-chromium, heretofore on the market, has contained 8 to 12 per cent. of carbon. This high carbon content has prevented the application of chromium in low carbon steels, but improvements in processes now make possible the manufacture of ferro-chromes with as little as 4 or 5 per cent. carbon and consequently the field for their use will be much extended.

Manufacture of Ferro-Chromium Alloys.—Electric furnaces of various kinds aid in making ferro-chromium alloys. In one form of furnace the crucible is one of the electrodes. The furnaces of this type are in cross section partly square, partly circular, and partly rectangular, and consist of iron boxes with thick linings of anode carbon with tar binders, provided with tapping holes. In size they vary from 120 in. wide, 80 in. long and 60 in. high, to 60 in. square and 50 in. high. In some of these furnaces one electrode is constituted by the lining, the other by a vertical bar of carbon in the furnace; others again have two parallel movable electrodes hanging vertically in the crucibles and in such cases the crucibles are detachable from the electric circuit.

Hanging electrodes of this type are made of carbon bars 50 in. long, 4x4 in. cross section, one electrode being formed by two bars side by side, such electrodes being fastened into iron heads, and cooled by water. Through continued use the carbon lining of the furnaces is gradually replaced by melted material, until finally the crucible becomes practically lined throughout in this manner. Seven such furnaces require an alternating current of 22,000 amperes and 100 volts. It is estimated that the energy consumed in producing these ferro-chrome alloys is about 10.6 h.p. hours per kilogram of product obtained.

BIBLIOGRAPHY.

The following special articles supplement the usual statistics of production and notes on the progress of the chrome ore industry. In Vol. II there is a detailed discussion of occurrences of chrome ore both in the United States and in foreign countries, and also an account of the theory and practice of manufacturing chromium salts from the ore. Vol. IV contains some detailed notes on the occurrence of chromite in Quebec and Austria and a 12-page article by S. Peacock upon the manufacture or chromates, describing in detail the material, plant, manipulation, furnace mixtures and by-products. In Vol. VI there are notes on the mining of chromite deposits in Newfoundland and in Turkey. In Vol. VIII the manufacture of metallic chromium is described and notes on the metallurgy of chromium are given. Methods of analysis are likewise described in this volume. In Vol. XIV the state of the chromite mining industry in the United States in 1905 is described and an account is given on the manufacture of ferro-chrome by the electric furnace, together with the process of steel making from chromiferous iron ore and the determination of chromium in steel by volumetric methods.

The following patents relating to chromium and chrome salts were published in 1906:

"Electric Furnace Process." G. Gin, Paris. The inventor prepares in an electric furnace metals such as manganese, chromium and nickel, quite free from carbon. The oxide of the metal is first heated with silica and carbon, so producing a silicide of the metal and carbonic oxide gas; the silicide is heated with a further amount of oxide, so producing pure metal and a silicate of the metal; the silicate is subsequently heated with more oxide and carbon, and silicide and carbonic oxide thus formed. Brit. pat. 8221 of 1905.

"Reduction process." H. Goldschmidt and O. Weil, Essen, Germany. Improvements in the process for reducing chromium and manganese from their ores by means of finely powdered aluminum, consisting in using a mixture of lower oxides of the metal with a few per cent. of the higher oxides. For instance, to every 100 parts of oxide of chromium three to four parts of chromic acid are added and the mixture added to 34 or 35 parts of aluminium powder. Brit. pat. 20,004 of 1905.

COAL AND COKE.

By Floyd W. Parsons.

The United States has produced approximately 6,000,000,000 tons of coal, and the annual output at present is more than 400,000,000 tons which shows an increase of about 100 per cent. in the last 10 years. The production

PRODUCTION OF COAL IN THE UNITED STATES.
(In tons of 2000 lb.)

States.	1905.			1906.		
	Short Tons.	Value at Mines.		Short Tons.	Value at Mines.	
		Total.	Per ton.		Total.	Per ton.
Bituminous.						
Alabama	11,900,153	$14,875,191	$1.25	12 851,775	$17,349,896	$1.35
Arkansas	1,934,673	2,515,075	1.30	1,875,569	2,438,240	1.30
California	48,558	130,065	2.68	(e) 80,000	232,000	2.90
Colorado	8,844,711	11,498,124	1.30	10,308,421	13,916,368	1.35
Georgia and N. Carolina	385,600	482,000	1.25	(a) 363,463	407,247	1.12
Illinois	37,183,374	38,689,858	1.04	(g) 38,317,581	39,467,108	1.03
Indiana	9,772,404	10,261,024	1.05	11,422,000	11,878,880	1.04
Indian Territory (f)	2,970,961	5,398,589	1.82	2,980,600	5,663,140	1.90
Iowa	6,728,000	10,460,000	1.55	7,017,485	10,877,102	1.55
Kansas	6,780,225	10,509,349	1.55	(a) 6,010,858	8,935,195	1.49
Kentucky	8,038,646	7,810,154	0.97	9,740,420	10,714,462	1.10
Maryland	5,108,539	6,641,101	1.30	5,014,995	6,772,243	1.35
Michigan	1,380,307	2,199,207	1.59	1,370,860	2,193,376	1.60
Missouri	4,733,164	7,809,721	1.65	3,860,000	6,176,000	1.60
Montana	1,643,832	2,547,940	1.55	(a) 1,787,934	3,186,620	1.78
New Mexico (f)	1,576,000	2,364,000	1.50	1 973,658	2,960,487	1.50
North Dakota	271,928	339,910	1.25	(a) 300,998	937,894	1.45
Ohio	25,834,657	27,643,083	1.07	27,213,495	29,934,845	1.10
Oregon	109,641	274,103	2.50	(a) 79,731	212,338	2.66
Pennsylvania	119,361,514	127,534,895	1.07	129,532,991	145,076,950	1.12
Tennessee	5,195,200	6,753,760	1.30	6,210,000	7,141,500	1.15
Texas	1,200,684	1,921,094	1.60	1,290,600	2,064,960	1.60
Utah	1,594,943	2,392,415	1.50	1,839,219	2,942,750	1.60
Virginia	2,113,950	7,610,808	1.85	4,546,040	8,501,095	1.87
Washington	2,818,042	5,495,182	1.95	3,293,098	6,421,541	1.95
West Virginia	35,283,382	30,557,938	0.87	46,452,000	44,129,400	0.95
Wyoming	5,446,525	9,531,419	1.75	5,805,322	10,159,314	1.75
Alaska and Nevada	85,000	297,500	3.50	(e) 90,000	360,000	4.00
Total Bituminous	308,344,613	$354,543,505	$1.15	341,629,113	$400,550,951	$1.17
Anthracite.						
Colorado	60,503	181,509	3.00	50,000	155,000	3.10
New Mexico	24,000	78,000	3.25	20,000	70,000	3.50
Pennsylvania	78,647,020	178,528,735	2.27	72,139,566	166,082,002	2.30
Total Anthracite	78,731,523	$178,788,244	$2.27	72,209,566	$166,307,002	$2.30
Total Coal { Sh. Tons	387,076,136	533,331,749	1.38	413,838,679	566,857,953	1.37
Total Coal { Met. Tons	351,120,625	....	1.59	375,397,204	....	1.51

(a) Figures reported by the U. S. Geological Survey. (e) Estimated. (f) Fiscal year ending June 30. (g) Fiscal year ending June 30, 1906.

of anthracite coal in Pennsylvania in 1906 showed a considerable decrease from 1905. This falling off was due principally to a suspension of operations caused by labor difficulties. It is also true that the great prosperity of our country affects the bituminous output more quickly and in greater

COAL PRODUCTION IN THE CHIEF COUNTRIES OF THE WORLD.
(In metric tons.)

Countries.	1902	1903	1904	1905	1906
Asia:					
India	7,543,272	7,557,400	7,682,319	7,921,000	8,875,000
Japan	9,701,682	10,088,845	11,600,000	11,895,000	12,500,000
Australasia:					
New South Wales	6,037,083	6,456,524	6,116,126	6,035,250	7,748,384
New Zealand	1,386,881	1,442,916	1,562,443	1,415,000	1,600,000
Other Australia	931,148	771,536	769,723	805,000	870,000
Europe:					
Austria Hungary (c)	39,479,560	40,160,823	40,334,681	40,725,000	40,850,000
Belgium	22,877,470	23,913,240	23,380,025	21,844,200	23,610,740
France	29,997,470	34,906,418	34,502,289	36,048,264	34,313,645
Germany (c)	150,600,214	162,457,253	169,448,272	173,663,774	193,533,259
Italy	413,810	346,887	359,456	307,500	(e) 300,000
Russia	16,465,836	(f) 17,500,000	19,318,000	17,120,000	16,990,000
Spain (c)	2,807,550	2,800,843	3,123,540	3,199,911	3,284,576
Sweden	304,733	320,390	320,984	331,500	(e) 265,000
United Kingdom	230,728,563	233,419,821	236,147,125	239,888,928	251,050,809
North America:					
Canada—					
Western	1,826,221	1,791,798	2,619,816	3,183,909	3,717,816
Eastern	4,699,396	4,700,645	4,194,939	4,775,802	6,196,360
United States	273,600,961	317,272,110	318,275,920	351,120,625	375,397,204
South Africa (a)	2,213,275	2,957,736	3,015,000	3,218,500	(e) 3,900,000
Other Countries (e)	3,500,000	4,000,000	4,250,000	4,550,000	5,500,000
Totals	804,115,125	882,865,185	867,020,658	928,049,163	990,502,793

(a) Transvaal, Natal and Cape of Good Hope. (c) Includes lignite. (e) Estimated. (f) Estimated by Minister of Finance.

PRODUCTION OF COKE IN THE UNITED STATES.
(In tons of 2000 lb.)

States.	1905.			1906.		
	Short Tons.	Value.		Short Tons.	Value.	
		Total.	Per Ton.		Total.	Per Ton.
Alabama	2,756,698	$6,753,910	$2.45	2,939,090	$7,494,680	$2.55
Colorado	795,650	2,585,863	3.25	1,133,643	2,766,089	2.44
Georgia and N. Carolina	69,200	204,140	2.95	(e) 10,000	29,500	2.95
Indian Territory (a)	41,193	189,488	4.60	59,088	230,443	3.90
Kansas	10,200	31,110	3.05	(e) 12,000	37,200	3.10
Kentucky	65,475	166,961	2.55	(e) 110,000	286,000	2.60
Missouri	3,100	8,525	2.75	(e) 3,000	87,000	2.90
Montana	43.500	282,750	6.50	(e) 55,000	330,000	6.00
New Mexico (a)	76,737	222,537	2.90	(e) 105,000	315,000	3.00
Ohio	112,250	342,363	3.05	(e) 115,000	333,500	2.90
Pennsylvania	18,519,310	45,187,116	2.44	20,681,702	56,874,681	2.75
Tennessee	468,799	1,181,373	2.52	(e) 505,600	1,334,784	2.64
Utah (b)	....	....	....	282,195	234,000	2.60
Virginia	1,203,650	2,837,578	2.35	(e)1,408,300	3,450,335	2.45
Washington	51,072	240,038	4.70	(e) 55,000	247,500	4.50
West Virginia (a)	2,738,777	6,846,942	2.50	3,395,744	7,436,680	2.19
Other States	1,535,000	5,219,000	3.40	(c) 1,820,000	5,400,000	3.60
Total Coke, Short tons	28,490,611	$72,299,694	$2.54	32,690,362	$86,887,392	$2.66
Total Coke, Metric tons	25,846,673	....	2.80	29,653,664	....	2.93

(a) Fiscal year ending June 30. (b) Included with Colorado up to 1906. (e) Estimated. (c) Includes output of by-product coke for Massachusetts, Maryland, Minnesota, New York, Michigan, Illinois, Wisconsin.

proportions than it does the anthracite production. This is due to the fact that bituminous coal is so largely used for steam purposes, while the coke output is consumed in the manufacture of steel; on the other hand

the anthracite production is largely used for domestic purposes. It is probable that our anthracite production has about reached its maximum, although it may show but little decrease during the next decade.

The coke output increased in greater proportion during 1906 than the output of anthracite and bituminous coal. This increased activity was entirely due to the great prosperity in the manufacture of steel. In 1880, there were only 7200 coke ovens, with an output of 2,250,000 short tons, selling at an average price of $1.79, and producing a gross revenue of less than $4,000,000; in 1890 there were 16,000 coke ovens producing 6,500,000 tons, which sold at an average price of $1.94 per ton, producing a gross revenue of $12,500,000; in 1906 there were more than 38,000 coke ovens producing approximately 32,000,000 tons, which sold at an average price of $2.70 per ton, and produced a gross revenue of nearly $87,000,000.

The suspension of mining that occurred in many of the bituminous districts during 1906, pending a settlement of labor difficulties, did not materially affect the coal output for the year. The differences that occurred were anticipated, so that a heavier tonnage preceded and followed the tie-up. The great business prosperity of the year was visibly reflected in all our coal districts, and although many new mines were opened and several virgin fields developed, the production is no further ahead of consumption than it was 10 years ago. The destructive competition that took place three years ago is now generally suppressed. This is due not only to the many large consolidations and better understanding among independent operators, but also to the revised contracts with labor, especially in Indiana and Illinois, which have resulted in establishing a fair and fixed tonnage price for the product. Pennsylvania and West Virginia will continue to furnish a large part of the total output. However, it is expected that southern Illinois and eastern Kentucky will show, perhaps, the greatest new development, with Colorado as a close second. For many years the fear of over-production has worried many coal managers and deterred capitalists, but the same feeling prevailed 20 years ago, and is as far from realization now as then.

The accompanying table gives the production of coke in the United States in 1905 and 1906 by States. Since the manufacture of coke is entirely influenced by the condition of the metallurgical industries, 1906 was the banner year in the coke industry. The supply from the Connellsville region proved inadequate to satisfy the demand of the great steel industries in that territory, so that much activity was manifested in erecting new coke plants in other parts of Pennsylvania and West Virginia. The Connellsville field continues to produce the greater part of the coke output in this country, while West Virginia, Alabama and Virginia are the next largest coke producing States. It is probable that Kentucky and Colorado will show great advancement in the manufacture of coke during

the next decade as both of those States contain large fields of cokable coal.

Imports and Exports.

Exports of coal and coke from the United States for the full year are reported in short tons as follows:

	1904	1905	1906
Anthracite	2,228,392	2,229,983	2,216,969
Bituminous	6,345,126	6,959,265	7,704,850
Total coal	8.573,518	9,189,248	9,921,819
Coke	523,090	599,054	765,190
Total	9,096,608	9,788,302	10,687,009

The exports do not include coal bunkered, or sold to steamships engaged in foreign trade.

The coke exported went chiefly to Mexico and eastern Canada; the distribution of the coal was, in short tons, as follows:

	1904	1905	1906
Canada	6,577,954	6,964,630	7,533,346
Mexico	880,747	927,170	1,084,319
Cuba	519,227	564,385	689,833
Other West Indies	253,585	300,776	319,839
Europe	144,354	101,277	81,734
Other countries	197,651	331,010	212,748
Total	8,573,518	9,189,248	9,921,819

The exports to Europe were chiefly to Italy; those to other countries to South America. The exports to Canada—70.5 per cent. of the total in 1906—were: anthracite, 2,176,342, a decrease of 11,108 from the preceding year, and bituminous, 5,357,004, an increase of 519,824, as compared with 1905.

Imports of coal and coke into the United States are reported as follows in tons of 2000 lb.

	1904	1905	1906
Canada	1,211,304	1,331,292	1,427,731
Great Britain	135,292	94,600	106,771
Australia	235,069	184,426	191,758
Japan	45,429	41,956	11,996
Other countries	759	569	6,251
Total coal	1,628,675	1,652,843	1,744,507
Coke	....	181,376	128,461
Total	1,628,675	1,834,219	1,872,968

Of the coal imported in 1906, there were 32,357 tons classed as anthracite. Some Nova Scotia coal comes to New England ports, but the bulk

of the imports of coal are on the Pacific coast. The coke is chiefly from British Columbia though a little comes from Germany.

Review of Coal Mining By States.

Alabama. (By L. W. Friedman).—A serious railroad-car shortage during the last four months of 1906 interrupted the production of coal in Alabama. The total output for 1906 was 12,851,775 tons, an increase of nearly one million tons over the 1905 production. The car shortage at several of the larger mines, especially where the coal was not used in iron making often caused more than two-thirds time to be lost.

Much development took place during the year. The Louisville & Nashville Railroad built several miles of track on its North Alabama railroad, to coal properties in the western part of Jefferson county and in Walker county. The Central of Georgia railroad built an extension in Jefferson and St. Clair counties. The Southern Railway built a branch line for the Sloss-Sheffield Steel and Iron Company, while the Frisco put down a branch line in Walker county to reach coal properties. There was a demand for every ton of coal that could be mined. The strike of the union coal miners, which began two years ago, was called off in June, 1906, and many of the idle men had no trouble in getting employment at old and new mines. The effect was that, with the new men brought into the district, and the strikers returning to work, more were employed in coal mines in Alabama at the end of 1906 than ever before. State Mine Inspector J. M. Gray states that had the transportation facilities been what was expected by the coal producers the production would have been several hundred tons better.

But one serious accident happened during 1906—that at Piper, where an explosion killed 12 men. Nearly 100 men lost their lives in coal mines in Alabama during 1906, as compared with 185 the year previous. The loss of life in 1905 was due mostly to the explosion in the Virginia City mines, in which 112 men were killed. During 1906 the State Mine Inspector gave notice that the mining laws of the State would be enforced to the letter, and several reforms were introduced by him. He forbade the use of dynamite, and excessively heavy explosives in the mining of coal, and demanded that superintendents and mine bosses caution the miners.

The Sloss-Sheffield Steel and Iron Company during the year began the opening of its Bessie mines in the western part of Jefferson county and it is expected the daily production will run something like 1500 tons.

The coke production was steady. The demand was met sparingly. Several new and large washers were erected by the Tennessee Coal, Iron and Railroad Company and others, looking to an increase in the production and a better grade of coke.

Arkansas.—The coal output in Arkansas has shown a small decrease during each of the last two years. This falling off in production has been due largely to market conditions, and the high charges for transportation, which have made it difficult for the coal producers in this State to compete with other districts lying nearer the large markets. Many tests have shown that the coal in this field is of an excellent quality, about one-half of the product being a semi-anthracite, which gives a high heat and burns without giving off much smoke.

The most important coal district in Arkansas is in Sebastian county, where there are 17 mines. More than half of the State's entire output is produced in this county. Franklin county is the second largest producer in the State, having furnished about one-fifth of the 1906 production. The three mines that are operating in this county are located near Denning postoffice, and employ about 400 men. Johnson county, which produced 229,477 tons of coal last year has five mines and employs 215 men. The town of Spadra is the principal coal center in this county. Pope county with three mines and Logan county with two mines produce a small quantity of coal. The mines in Pope county employ 116 men, while the two mines in Logan county employ about 50 men.

The report of chief mine inspector, Martin Rafter, gives the following interesting figures, with reference to coal operations in Arkansas for the year ending June 30, 1906: Number of tons coal mined for each fatality, 170,506; number of tons of coal mined for each keg of powder used, 25; total number of miners, 3027; total day-men worked in and around the mines, 1384; total number of employes who got employment at the mines, 4411; average number of days worked by each miner, 154; average daily tonnage for each miner, 4.02.

California.—Since the development of the petroleum industry, the mining of coal in California has not been active, and the greater economy in the use of petroleum for steam-making purposes has been the cause of many coal mines closing down or decreasing their output. However, owing to the advance in the price of petroleum during 1906, and the exceptional demand for coal for domestic purposes, the output of coal mines has been increased lately.

Colorado.—This State experienced considerable activity in the coal industry during 1906, and all indications point to a continuance of the present prosperous conditions. The coal output for Colorado in 1904 showed a decrease of more than 1,000,000 short tons, compared with the previous year's production. This falling off, however, was offset by an increase of more than 2,000,000 tons in 1905, with a further increase of nearly 1,500,000 tons in 1906. The new development work that is now taking place, although most actively proceeding in the Trinidad district, is well distributed over the entire State. The main effort has been

to perfect operations and increase the output from the properties already being worked rather than to open new mines.

The largest producing companies in Colorado are the Colorado Fuel and Iron Company, and the Victor Fuel Company; the latter concern having recently passed under the control of John C. Osgood, formerly of the Colorado Fuel and Iron Company, has shown the most aggressive development, with the result that the production from the Victor mines has largely increased.

The principal incident attending the coal industry in Colorado during 1906 was the exploitation of the Yampa coalfield, which although known for many years, has been little prospected, because of its inaccessibility. It is now the intention of the Denver, Northwestern & Pacific railroad to extend its present line from Middle Park through Gore cañon, and thence across to the Yampa valley. When this construction is completed, the Yampa coalfield will become one of the most important sources of coal supply in Colorado. This field (which covers practically 1200 square miles) centers in a broad area of rolling country in the vicinity of Hayden. The Yampa river cuts across the field in a westerly direction, and it is down this valley that the surveys for the railroad have been made.

Investigation has shown that both anthracite and bituminous coal of good quality are found in this district. Three groups of coal beds are present, known as the upper, middle and lower groups. Each group of coal seams has a vertical thickness of 200 to 400 ft., and the several groups are separated by 500 to 1000 ft. of barren shale and sandstone. There are at least three seams of workable thickness contained in each group; the largest bed of coal reported from the field has a thickness of about 20 ft. The anthracite field lies north of the Yampa river. The character of the coal is due to igneous intrusions. At the present time this part of the field is attracting more attention than any other; however, the investigations so far made seem to show that although the coal is of a higher grade, it is of limited geographic extent, and may not be so valuable as many have supposed.

Although Las Animas county is the most important producer in Colorado, the operations carried on in several other localities are of increasing importance. The production from the mines in Gunnison county during 1906 had a value of nearly $6,000,000. The product from this district goes as far west as the Orient, and east to the Missouri river. The operating expenses of the various mines amounted to more than $1,000,000, while employment was given to about 1100 men. The principal coal camps in this region are Crested Butte, Somerset and Mt. Carbon. The Ruby anthracite mines, west of Crested Butte, cover an area of 1400 acres and the coal seam is 3 to 4 ft. in thickness. The annual production is about 6000 tons. The other anthracite mine on Smith Hill, operated

also by the Colorado Fuel and Iron Company, has a production equal to the Ruby.

Idaho. (By Robert N. Bell).—The desposits of steam coal discovered in Fremont county were developed to a limited extent during 1906. One of the mines, operating a 5 ft. vein, produced 20 tons of coal a day for local consumption. This property has 10 distinct seams, varying from 2 to 10 ft. thick and exposed in shallow shafts, cuts and adits, a distance of three miles. The series is somewhat faulted, but indicates an extensive resource of high-grade steam coal. Capital has been attracted to the discovery but unfortunately it is located on unsurveyed land, and the President's withdrawal of Western coal lands from entry, involving the uncertainty of obtaining title, has deterred progress.

Illinois.—Coal mining was prosperous in this State in 1906, but the output was only a little more than in 1905. Great activity was maniifested in the development of new fields, and the opening of new mines. The general tendency was toward a consolidation of the smaller independent concerns, and the forming of larger, stronger corporations. The entire State may be said to be thoroughly unionized, and as a partial result of this condition, the operators are receiving a more uniform price for their product.

One of the difficulties in coal mining throughout Illinois is the attempt on the part of the operators to observe general laws that cannot be fairly applied to all the mines because of the differing conditions. While this is an advantage to some, it is a serious impediment to others. The O'Gara Coal Company, Deering Coal Company, and the Vandalia Coal Company have all pursued an active policy of expansion and aggressive development. The greatest improvement in operating the mines in this State has come from the efforts of these larger companies to abandon the irregular and unsystematic methods that have heretofore been used, and to follow some general plans that are thought to be suited to the mines in question.

Of all the coal States of any importance, Illinois has probably lost more coal through inefficient methods of mining than any other. It is a fact that in many localities the conditions seem to prevent the systematic robbing of pillars; however, it is certainly true that acres of coal that have been lost through squeezes, might have been recovered, had a more systematic plan been adopted.

Last year did not show much advancement so far as coal washeries and methods of treatment are concerned, but in the construction of steel tipples, modern screens, self-dumping cages, and the installation of modern power plants, there was a noteworthy advance.

New railroads are being built, new companies formed, and greater activity than has heretofore been witnessed is taking place. The present

outlook is that Illinois will probably show a greater new development during 1907 than any other coal State.

Indiana.—A review of the coal mining industry in the State of Indiana in 1906 discloses some peculiar and annoying features. During April and May and a portion of June little or no coal was mined in this State on account of the strike that prevailed among the union miners, due to the inability of the miners' and operators' associations to agree upon a wage scale. During this period all the mines of the State with the exception of a few small operations were idle. About the first of June work was commenced in some of the mines, the owners having agreed to pay the demands of the miners. Other mines gradually began operation and by the middle of June all the mines in the State were running again. During the two months of idleness the operators made good use of the time in improving the mines, installing new machinery and equipment, and otherwise preparing for the production of an increased output. In February and March the mines were operated to their fullest capacity because of the impending strike, to be inaugurated April 1. This accumulation was pretty well shipped out when work was resumed June 9. During June and July there were a few strikes at several mines due to a misunderstanding of certain provisions of the agreement. However, these cessations did not materially affect the output.

At the beginning of December conditions became somewhat alarming, especially to the numerous industries that use soft coal as fuel. The storage of coal can be attributed accurately to the lack of cars properly to handle the output. This also was one of several causes leading to the small increase in the output in Indiana during 1906 as compared with 1905. The Indiana Railroad Commission, while besieged with a multitude of complaints from various operators and shippers, intervened only in a friendly way. Better conditions developed after the Railroad Commission filed suit against the principal railroads to compel them to clear up the coal blockade and embargo in the State. For weeks and months it was practically impossible for the operators of southern Indiana coal mines to get their product to the central and northern Indiana markets. The Commission and operators will unite to secure from the legislature a law compelling the railroads to haul the cars a certain distance each day after they are loaded and to return the cars within a given time after they are unloaded. If a stiff penalty is attached for failure to do business on these lines the commissioners and operators believe that the railroads will come to time and cease their quarreling and handle cars more readily. The operators, who have heretofore counseled moderation in dealing with the railroads, are now advocating radical laws.

According to statistics already gathered the production of coal in Indiana in 1906 was approximately 11,422,000 tons, an increase of 1,649,556

tons as compared with 1905. The number of miners employed was about the same as in 1905, viz: 18,811, while the number of employees in offices has decreased owing to the merging of companies. The estimated wages paid to miners during 1906 is $8,160,602. This is a decrease of $135,604, nothwithstanding the increase in the wages paid to miners. There were six mines abandoned and four new mines opened in 1906 against 14 new mines in 1905. The selling price of coal was lower on an average, having run from $1 to $1.25 for run-of-mine and from $1.25 to $1.75 for screened coal. The bulk of the coal at the present time is being mined by machinery and the rate that machines are being introduced is a sure indication of their success. There were 15 mine accidents from various causes during 1906 against 24 accidents in the previous year. The fatalities numbered 25, whereas there were 47 in 1905.

Indian Territory.—According to recent surveys, there are 437,734 acres of workable coal in this Territory. At least one-fourth of this area has already been leased, the Indians receiving on an average about 8c. per ton royalty, which according to statistics, yields them about $400 an acre. If these figures are taken as a basis, the computation would show all the coal lands in the Territory to be worth in royalties alone, $174,000,000. The value of the coal at the mines varies from $1.90 to $2 per ton. If the latter figure is applied to the entire acreage, the total valuation becomes $4,377,000,000.

The mines operated by the coal department of the Missouri, Kansas, & Texas Railway Company are the largest producers of coal in the Indian Territory. These mines are located near Coalgate, while the main office is at Parsons, Kansas. Mines Nos. 9, 10, and 12 are producing, while mine No. 14 is still in the period of early development. The first three mines are shaft mines, while the coal seam in the last operation is reached by slopes. In all the mines of this region, the coal shows a thickness of from 4 to 5 ft. The pitch of the seam is variable; at mine No. 9, the bed dips 14 deg. to the south, while at mine No. 14, the present development tends to show that the seam pitches 15 deg. to the north. The Lehigh vein is the only seam worked. It produces a coal that answers fairly well for steam and domestic purposes.

Iowa.—The State Board of Mine Inspectors reports that the production of coal in Iowa for the year ending June 30, 1906, was 7,017,485 short tons, against 6,728,000 tons in the previous year. The number of employees was: Miners, 11,615; other inside men, 3629; outside men, 1581; total 16,825. This shows an average output of 417 tons for each employee, and 604 tons for each miner. The casualties reported for the year were: Deaths 37, or 2.2 for every 1000 miners employed. The serious injuries numbered 116, or 6.89 per thousand men employed.

Kentucky.—During 1906 there was a material advance in output and greater activity in the development of new districts. It is now becoming more generally understood that the coal seams in eastern Kentucky produce a fuel equal in many respects to the products of the southern West Virginia fields. This fact, combined with the comparatively cheap price asked for the available coal acreage, has induced many capitalists to investigate and secure holdings in the region. The output in 1906 amounted to 9,740,420 short tons, or more than a million tons over the previous year.

One of the most important coal fields in Kentucky is the Cumberland Gap district, which was entirely unknown 16 years ago. It lies in a basin 90 miles long by 15 to 20 miles wide, and contains numerous coal beds from 5 to 7 ft. thick. This field is now connected with Louisville and Cincinnati by the Cumberland Valley Branch of the Louisville & Nashville Railroad, and with Knoxville by the Southern Railway. Since communication has been established with the markets, the coal output of the field has grown to about 1,000,000 tons per year. The development of the coal is confined to the western part of the field, where investigations have shown 13 seams of workable thickness and good quality. Of these seams, eight are commercially valuable. One unfavorable condition, however, is that all of the seams in this district have one or more partings. The coal of the western part of the Cumberland field has been little tested. Some of the coals near Middleboro' are successfully coked, and doubtless most of the product of this area is of the same character, being almost as free from moisture, ash and sulphur, as the best Appalachian coals.

Maryland.—This State has probably reached its pinnacle so far as its position as a coal producer is concerned. The principal region is known as the Georges Creek field and is contained in a deep broad syncline approximately 20 miles long and five miles wide. The exact date of the discovery of coal in the Georges Creek district is not known, but as early as 1825 there was considerable mining, the coal being hauled to Cumberland in wagons and there loaded on flat boats, which carried it down the Potomac. The Baltimore & Ohio Railroad was completed to Cumberland in 1842, which caused increased activity in coal mining throughout the region.

The excellent quality and desirable location of the Pittsburg or "Big Vein" seam, has made it the one most generally worked, although the Conemaugh and Allegheny coals, which underlie the Pittsburg seam, cover much greater areas than the "Big Vein." In the main part of the field, the Bakerton, Upper Freeport, and Lower Kittanning seams are so located that they can be mostly worked by drifting.

Since the Georges Creek coalfield was first opened, more than 125,000,000 tons of coal have been mined. More than 90 per cent. of this has

come from the "Big Vein," which averages about 14 ft. thick, although but 8 ft. of it is worked. Above this workable coal is what is locally called the "rashing" which is left to act as roof for the rooms and entries. Whether this inferior coal will ever be recovered is doubtful, for robbing is extensively going on throughout the field, and the overlying cover is being allowed to come down as the work retreats. There are no washeries in the field, and the coal is of such excellent quality for steam and domestic purposes that in many places it is not even run over a domestic screen, but it is loaded for shipment as run-of-mine. The largest operation in the Georges Creek field is that known as the Consolidation Coal Company. This company is a part of the consolidation which includes the Fairmont Coal Company, the Somerset Coal Company and the Clarksburg Fuel Company.

The miners employed throughout the region are far superior in intelligence to those in most other American mines, and this has resulted in making this field particularly free from rowdyism and labor troubles. The system of mining here carried on is ideal for the conditions present, and results in extracting probably 95 per cent. of the entire seam. At the Ocean No. 7 mine, the largest in the State, the method adopted is so planned that some of the rooms are from 600 to 1000 ft. in length. Nearly 14,000 long tons of coal have been extracted from one acre of the "Big Vein," which shows the immense production available. Of this large tonnage, the coal mined from the entries and rooms is in better blocks and contains less slack than the coal broken by the weight of the cover in robbing pillars. The production in 1905 showed a small increase over that of 1904, but the output in 1906 shows a small falling off, which signifies that the field has reached its maximum production. One fact that is particularly deserving of mention concerning the Georges Creek coal field is the absence of serious accidents. This may in part be attributed to the fact that few of the mines contain any considerable quantity of explosive gases, but also it goes to prove that great care is exercised by the operators in carefully timbering, and in various ways eliminating many of the dangers that are attendant upon coal mining.

Michigan.—This State contains a workable coal area of more than 6000 square miles. The coal seams are reached by shafts, and at present only those veins lying within 250 ft. of the surface are being worked. Michigan produced less than 1,500,000 short tons of coal in 1906; however, conservative estimates place the available coal of the State at more than 15,000,000,000 tons.

Missouri.—The coal output of Missouri in 1906 showed a considerable decrease, as did each of the three years immediately preceding. This falling off in production is due mainly to the fact that the large markets are supplied with coal from other fields at a lower cost for transportation

than the Missouri operators can secure. It is probable that Missouri will in time become more important as a coal-producing State, since the workable areas are considerable in extent, and the quality of the coal is equal to that in adjoining fields. Colorado is now producing a sufficient quantity of coal to supply many of the western markets that formerly used the Missouri product, so that the present output of Missouri is principally used for local consumption.

Montana.—The coal output of Montana in 1906 showed little increase over the substantial output of 1905, which was much greater than the production of any preceding year. The coalfields in this State are widely distributed and much remains to be learned concerning their extent and quality. It is claimed that Montana possesses a coal area of 1300 square miles and ranks ninth as compared with the other States in this regard. Most of the important coal operations in Montana are controlled either by the large copper producing companies of the Butte district, or the large railroads that traverse the State. Much of the fuel in Montana is lignite, and the product from all of the seams requires careful sorting or washing before it is commercially valuable. This is due to the partings that occur in the various beds and to the high ash content. In several instances, there have been strong companies formed in Montana to operate coal mines, and these concerns have spent large sums of money erecting coal washeries and building modern plants; so far, such investments have not proved remunerative.

New Mexico. (By C. R. Keyes).—Activity in coal developments was marked in 1906. The output increased, being about 2,000,000 tons, and progress in the construction of railroads to new fields will in the course of another year or two result in an immense advance in the production. In the northeast the Raton range coalfield received much attention, and a number of large mines were opened. Three lines of railroad are under construction, which will afford access to markets not now reached by the Atchison, Topeka & Santa Fe Railroad, which has long controlled this territory. The coal rights on the Maxwell land grant, embracing over 300,000 acres, have been recently sold to a syndicate which is also building a railroad eastward into Kansas, and whose present expenditures involve upward of $15,000,000. Phelps, Dodge & Co. are also surveying a new line, about 200 miles long, to tap their Dawson fields, already reached by their road from Tucumcari.

In the northwest, in the San Juan valley, great activity was manifested during 1906. At present the region is greatly hampered by inadequate transportation facilities, but these conditions will not long continue, as the Southern Pacific, Denver & Rio Grande, and other railroads are pushing into this territory. The San Juan field is in many respects the most remarkable in the country.

At the present time the Gallup field continues to be one of the principal producers and new developments here have been in progress.

A feature of special note has been the reopening, on a large scale, of the Carthage coalfield, near the Rio Grande in central Socorro county. The railroad from San Antonio has been entirely rebuilt, and the output is shipped directly over the Santa Fe.

Still further south, and near the site of the Elephant Butte dam, on the Rio Grande, some good coal has been opened up in the Mescal cañon. The seam is about 4 ft. thick, and the field is probably one of considerable extent. This is the most southerly of the known coal deposits of New Mexico. A large power-plant has here been erected and power transmitted over the wires will supply a number of mines and mills within a radius of 10 miles. In this locality are six other seams in the coal-bearing section. One bed of bituminous shale is 400 ft. in thickness.

Ohio.—The output of coal in Ohio during 1906 was approximately 27,000,000 short tons, continuing this State as the fourth largest producer. The increase was chiefly due to the greater activity by the larger operating companies. The principal producers are at present the Sunday Creek Coal Company, and the New Pittsburg Coal Company, both concerns carrying on their operations in the Hocking Valley region. Coal mining in eastern Ohio showed considerable advancement during 1906, but the greater part of the new development took place in the Hocking Valley. Ohio is the premier State in the use of mining machines, about three-fourths of its entire output being machine-mined coal. One peculiar circumstance with respect to coal mining in Ohio is the small number of tons produced by each employee when compared with the output per man in other coalfields. Only two States, viz., Iowa and Missouri, report a less tonnage produced for each employee than Ohio.

Pennsylvania.—Coal mining in the anthracite and bituminous districts of Pennsylvania was carried on with much activity during 1906. The labor troubles that occurred were of short duration and, if anything, were a blessing to the industry. It is probable that had no suspension of work occurred, the operators would have been compelled to have occasional temporary shut-downs to prevent the supply from exceeding the demand. The resumption of work on the terms accepted resulted in no particular victory for either side and tended to show that neither operators nor men have forgotten the awful experiences of the last great strike.

The shortage of cars was a trouble much heralded in the papers, but it may as well be realized at once that this difficulty was not without its redeeming features. If all of our mines were constantly to have plenty of cars for shipping their product it would not be long before many of them would not be working full time. This condition is an old trick

that the operators often take advantage of, and many times such car trouble is to the mine manager a welcome misfortune.

Much has been said concerning the employment of boys under the minimum age in our mines and on the breakers. The responsibility for this serious fault does not rest particularly with the operators or mine inspectors, but rather may be laid at the door of the parents who secure certificates for their children by swearing to a false age. Only by strict investigation and a careful enforcement of the laws will this trouble be eliminated.

Exact figures as to the amount of coal produced by compressed air and electric power are not yet available. Preliminary estimates seem to indicate that about half of the coal produced is from the use of compressed air, while the production by electric power amounts to nearly one-quarter of the total output. This is not due to any decrease in efficiency derived from electric machines, but rather to the fact that the law prohibits electric wires and the conduction of electric current in all mines or parts of mines that are considered gassy and are worked with safety lamps. The use of electricity for driving all of the jigs, screens and rolls in the newer anthracite breakers is an innovation that is meeting with much success.

Tennessee.—Although 1906 does not show the coal production in Tennessee to have made so great an increase as in 1905, the year was gratifying in its commercial results. The output of more than 6,000,000 tons gives this State the tenth rank among the coal producers, and all indications at present point to a much greater tonnage in the near future. The quality of the coal is excellent, as the following average analysis from one of the mines in Campbell county will show: Fixed carbon, 60.87 per cent.; volatile matter, 34.63; ash, 2.43; moisture, 1.62; sulphur, 0.43. The coal produced in 1906 had a value of about $6,700,000, while the coke production was valued at about $1,250,000, or an increase in value of approximately 10 per cent. over 1905. So far as accidents are concerned, the fatality rate in Tennessee has dropped back to its natural condition. The average for all coal mines in the United States is now about 3.45 per 1000 employees, whereas the rate in Tennessee, which was 2.69 in 1903, has dropped to below 2.60 for each one thousand men employed in 1906.

Virginia. (By Thomas L. Watson).—No shipments of coal were made from the mines of the Richmond coal basin during 1906. Actual developments in this basin were in progress at only one operation, namely, the mines of the James River Coal Corporation, at Midlothian, in Chesterfield county. The developments made by this company comprise the completion of an inclined slope at an angle of 33 deg. to a depth of 1022 ft., to coal 8 ft. thick. The coal in this shaft has been actually proved for considerable distances, so that the future of the company seems especially

good. The building of $1\frac{1}{4}$ miles of trackage, and the erection of a tipple are under way, and on their completion, in 1907, the shipping of coal will begin.

Rapid advances were made during 1906 in developing the Montgomery-Pulaski counties anthracite coal area. In the Montgomery portion of the field, the Merrimac mines of the Virginia Anthracite Coal Company, the largest operation in the district, produced steadily throughout the year with an output of 100 tons per day. Two smaller operations on the same ridge, Price mountain, and three on the nearby north ridge, known as Brush mountain, in the vicinity of Blacksburg, were producers, supplying strictly a local market.

In the Pulaski portion of the field, the developments were more extensive than in Montgomery county. The Parrot mine, formerly known as the Kimball, of the American Coal Company, produced on an average about 75 tons of coal per day. Extensive improvements were made during the year in the tipple of this mine. At the mines of the Belle Hampton Coal Company extensive development work, largely in the nature of deepening the mine slopes and extending the entries was carried on. During the latter part of the year the shipments from this mine were estimated at an average of about 25 tons of coal per day, taken out principally to meet the cost of the development work. About three miles northeast of the Altoona coal mines of the Bertha Mineral Company a slope was put down during the year to a depth of about 600 ft. with an excellent showing of coal.

The Pocahontas Collieries Company, at Pocahontas, carried on much new development work during 1906; not only were the older mines of this company greatly improved as to equipment and efficiency of methods, but a new mine was opened at Boissevain, in the heart of the Pocahontas field, and this new operation will likely have a production of about 3000 short tons per day.

The great impediment to increased operations in Virginia is the scarcity of labor, which condition has now become a serious problem.

Washington. (By R. P. Tarr).—This State contains coal measures which will furnish fuel not only for its own demands, but also for those of western Montana, Idaho and Oregon. Within this State are the only coal areas of importance in the Northwest. There are undeveloped and almost unknown deposits in Whatcom, Skagit, Snohomish, Chehalis, Pacific, Lewis, Cowlitz, Clarke and Skamania counties, and other parts of Kittitas than those where operations are now carried on.

When the earlier efforts were made to open the coal deposits of Washington, the serious mistakes occurred which produce adverse conditions now. Other than the knowledge of the existence of coal in certain parts of the State, little was known concerning any area until after the inves-

tigation and report of Bailey Willis, of the U. S. Geological Survey. His admirable report, practically confined to the Palmer-Wilkeson fields, directed attention there. Many mines were opened, operated temporarily, and then abandoned, remaining as monuments of the errors of those who were unacquainted with coal mining problems. These failures produced a tremendous reaction which put a quietus on future development.

The railroad and steamship companies attacked the problem of mining systematically, and success has attended their labors. Those railroads now contemplating entering the Sound country will do likewise. But there is another problem confronting the State. Along with the railroad development comes increase in population and many additional industries. Therefore the commercial side of coal development must be immediately considered. The railroads cannot take care of this; outside capital must. At present 30 mines are in operation, about half of which are of importance. The remainder have small outputs and are undergoing every sort of struggle to stagger on. Several of these are reopenings of abandoned properties and owing to former mistakes in mining methods should stay closed rather than be opened.

In 1906 the total coal output in this State was 3,293,098 tons. This was an increase of 475,056 tons over 1905. Of the 30 mines, five are operated by the Northern Pacific Railway Company and produce 44 per cent. of the entire output, eight are operated by the Pacific Coast Steamship Company and produce 37 per cent.

The accompanying analyses show the composition of the best types of coal mined for various uses:

COMPOSITION OF WASHINGTON COALS.

	Roslyn, Steam, etc	Burnett, Gas.	Wilkeson, Blacksmith.	Fairfax, Coke.
Moisture	1.91%	1.44%	1.38%	1.30%
Volatile matter	34.30	40.90	21.69	19.22
Fixed carbon	51.11	49.24	66.96	64.22
Ash	12.68	8.42	9.97	15.26

The coal is mined from three principal fields, viz.: Palmer (King county), producing 52 per cent.; Rosyln (Kittitas county), producing 41 per cent.; Wilkeson (Pierce county), producing 7 per cent. Of the entire output 66 per cent. is consumed by railroad and steamship companies. The remaining 34 per cent. is used commercially in Washington and tributary territory having a population of 1,250,000 inhabitants.

The Cascade uplift was very intense, but not to the degree of much metamorphism of the coal measures. During the uplift four active vol-

canoes were formed, namely: Baker, Rainier, Adams and St. Helens. Dikes from these have considerably broken the measures and aided in the production of faults. Near some of these a slight metamorphism produced coking portions. (This is graphitic coke and not anthracite as some assert.) The seams are of intense dip in many localities and are worked at dips varying from 5 to 90 deg. In those portions where the dip is less intense the amount of moisture runs from 10 per cent. to 15 per cent. Most of the coals come under the characterization of lignites. While far superior to the brown lignites of Dakota and Montana, these moist Washington coals do not stand transportation well and cannot be stored at all.

All Washington coals show a decided increase in ash, have a duller luster and lighter texture than eastern coals. Having been laid down during Cretaceous-Tertiary times, when the amount of carbon dioxide was greatly diminished, there is thus good reason for these different and inferior characteristics. While many seams are of extreme thickness, there are few over 6 ft. that are not streaked and exceedingly dirty. The containing walls of the coal seams are generally of shale and poor, but in a country plentiful in timber, this is not so serious a problem as elsewhere. Cleaning by the dry process will do much to benefit the quality. The glacial deposits and forest coverings have done much to render prospecting difficult, and to find us to-day with only a scant knowledge of the coal resources. However, nothing will contribute more to the permanent greatness of Washington than the proper development of the virgin portions of these coalfields. Altogether the promise is good, but greater enterprise must attack this problem.

West Virginia.—In the larger fields of West Virginia there was practically no check in the wonderful activity in coal mining that occurred in 1906, and as a consequence the production showed a substantial increase over previous years. There are some who will claim that West Virginia is no longer a State full of opportunities, but rather has become like Pennsylvania and some of the older districts where most of the valuable coal lands have already been accumulated. This is true only in part, for although some of the fields are closely held there are still thousands of acres of good coal land back from the railroads that is for sale at a reasonable price. There are also large land companies which are ready and willing to lease valuable tracts to competent men who can prove that they have the ability to open and successfully operate a mine.

So far as the industry as a whole is concerned, the most important happenings of 1906 were the completion and the successful operation of the Coal and Coke Railroad, and also the rapid development of the Deepwater road, which will carry coal to tidewater. Both of these railways are projected into new territories, and will consequently be the means of opening new fields.

For many years coal mining in West Virginia was unprofitable because of the destructive competition. This condition has been remedied by the consolidation of many mines in the several districts. There are still many independent operators, but they market their coal through one of the larger selling agencies. In northern West Virginia the largest operator is the Fairmont Coal Company, which controls nearly all of the mines in what is known as the Fairmont and Clarksburg fields. This company, by virtue of its wonderfully organized selling department, can dispose of all the coal its mines will produce; last year was one of unprecedented prosperity in this district. Although the Fairmont company is opening and equipping a large mine near Clarksburg, W. Va., the policy of this corporation has been to perfect and increase the output at the mines already opened, rather than to develop new properties.

Many new mines have been opened along the Coal and Coke Railroad, and the field opened by it will undoubtedly become a large producer in the near future. The coal in this region is not as good for steam purposes as the New River and Pocahontas product, but it is a first-class smithing coal, for which reason it has become famous.

The largest operators in the New River field are the New River company which was recently organized with Sam. Dixon (one of the veteran operators of the field) as president, and the New River Smokeless Coal Company. The New River company is perhaps carrying on new development work more aggressively than any other West Virginia corporation. The coal produced by it is unsurpassed for steam purposes, and because of its low volatile matter is considered almost smokeless. None of the coal is washed and a large part of the production is shipped directly as run-of-mine.

Operations in the Kanawha and Pocahontas fields were vigorously carried on. The largest operations in the Pocahontas field are controlled by the Pocahontas Collieries Company and the United States Coal and Coke Company. The main offices of the former are at Pocahontas, Va., while the United States Coal and Coke Company has its headquarters at Gary, W. Va. Both of these companies have large plants and convert a considerable part of their output into coke. The Pocahontas company is opening up a new mine at Boissevain, which, after completion, will be followed by other new development work.

The great problem throughout West Virginia and especially in the southern part is that of a deficient supply of labor. The scarcity of men has caused the operators considerable trouble and great efforts have been made to import miners into this field. Wages are above the average and the climate is particularly healthy. However, the region is somewhat removed from large towns and legitimate amusement is scarce. It is probable that West Virginia will continue to hold her place as the second

largest producer of coke, and that before many years it will have a soft-coal output rivaling that of Pennsylvania.

With regard to the tonnage produced for each employee engaged in the production of coal in the various States, it is interesting to note that West Virginia leads all the other States, the number of tons per man being approximately 800, or 59 tons more than Colorado, which ranks second. Maryland shows 734 tons per man, and Pennsylvania 720 tons. Alabama, with a production of 691 tons per miner, showed a decline of 60 tons, while Illinois with a production of 628 tons per man, showed a decrease of 49 tons.

Coal in Foreign Countries.

Australia.—The coal raised in New South Wales during 1906 was 7,626,362 long tons, valued at £2,337,227, exceeding the output in 1905 by 994,224 tons and £333,766 in value. The output by districts was as follows: Northern, 5,336,188 tons (£718,178); Southern, 1,783,395 tons (£494,-871); and Western 506,779 tons (£124,178). The coal exported during the year amounted to 4,901,760 tons valued at £2,076,223. This is much the largest quantity ever despatched from the State and shows an increase of 1,183,707 tons and £592,245 in value over that for the previous year. The number of persons engaged in the mining for coal in 1906 was 14,929, an increase of 910 compared with 1905. The value of coke manufactured during 1906 was £110,607, being an increase of £10,301 over the previous year, the industry having benefited largely by the greater activity in metalliferous mining.

In a paper by Professor David, read before the Engineering Society of Sydney University, it was stated that assuming 4000 ft. to be the limit down to which coal can be profitably worked, there is a total of 150,000,000,-000 tons of coal still available in the Palæozoic coal area of New South Wales, and a like amount in Queensland, which, valued at 6s. 8d. per ton, is equivalent to £100,000,000,000. One of the most important of the Palæozoic fields is that of the Dawson river, situated north of Rockhampton, in Queensland. The seam shows about 11 ft. in thickness, the coal being clean and of great economic value. The Dawson river coal has proved to be the finest steam coal yet discovered in Australia, though there is in New South Wales, near Inverell, a seam of coal which is thought to be a continuation of the Dawson seam, and is almost equal to it in quality. Tests made on this coal by the Admiralty have proved highly satisfactory, as the coal is nearly smokeless. Among the remarkable deposits of coal of Cenozoic age in Australia is that of the great Morwell coalfield, in Victoria. This is a brown coal, and one of the thickest masses yet discovered in the world, consisting of two seams, one 265 ft. thick, and the other 227 ft.

Belgium.—Coal production in Belgium for the year 1905 was 21,844,-200 metric tons, while 23,610,740 tons were produced in 1906; an increase of 1,766,540 tons last year. This increase in activity has brought Belgium back to the position it held as a coal producer in 1903 and 1904, and with a continuance of the present business activity, it is probable that 1907 will witness a greater production of coal in Belgium than ever before recorded. The largest part of the exportation of coal from Belgium in 1906 was made to France.

Brazil.—The coal formation of Brazil begins in northern São Paulo, near the border of Minas Geraes, and extends southwestward across the States of Paraná, Santa Catharina, and Rio Grande do Sul, probably passing into Uruguay and the Argentine. It was formerly supposed that the coal beds of Brazil existed only in narrow and separate geological basins which had never been connected, but it has been demonstrated by Doctor White, commissioner appointed by the Brazilian Government to report on the subject, that the coal formation is continuous from the State of São Paulo into and across Rio Grande do Sul. There is one main coal horizon near the top of the coal-bearing series above described, which appears to be continuous from Santa Catharina into Rio Grande do Sul, and it is possibly the same coal horizon which extends northward through Paraná and São Paulo. This main bed is thin (25 to 30 cm.) in São Paulo, at least where it is exposed at the surface, but thickens to 50 cm. in Paraná, to nearly 1 m. in Santa Catharina and to nearly 2 m. in Rio Grande do Sul, thus exhibiting a progressive increase southward.

The coal from the upper or main bed is practically of the same quality from São Paulo to Rio Grande do Sul, and always contains much sulphur and slaty material. The coal is mined for commercial purposes, having 25 to 35 per cent. of ash, of which 2 to 5 per cent. is sulphur in the form of nuggets of iron pyrites, and hence the coal in its crude state is too impure for general commercial purposes. Several tons of the coal were sent to Germany for experimental purposes and the results of the tests made by the Humboldt Engineering Works at Kalk, near Cologne, were encouraging, since it was found possible to remove practically all of the sulphur and to reduce the ash to 12 or 13 per cent. in 33 per cent. of the coal, while in 42 per cent. of the remainder the ash was reduced to 26 per cent. and the sulphur was practically eliminated. The coal, like all of the beds in the later Carboniferous series, contains more combined water than the earlier formed coals, and hence its effective heat of combustion is thereby decreased below that of Cardiff coal, the single determination of a briquette made from the coal of San Jeronymo in Rio Grande do Sul having given 12,500 B. t. u. as its heating value when the ash is reduced to 13 per cent.

Canada.—The production of coal in Canada in 1906 shows a small

increase over the output for 1905. Of the total production in 1906 approximately 1,500,000 tons were exported. Nova Scotia and British Columbia are the chief centers of production. Imports from the United States during 1906 were 2,246,168 metric tons of anthracite and 5,556,658 tons of bituminous, a total of 7,812,826 as against 6,739,619 tons in 1905 and 6,294,880 tons in 1904.

China.—The Belgian Consul at Tientsin reports that the first Manchurian coal mines worked by the Russians were those at Yentai, but no very satisfactory results were achieved and these were abandoned in favor of the mines at Fuchung, situated to the east of Mukden, on the Hung-Ho, which in the opinion of Mr. Tanaka, the director of the Tokio Bureau of Mines, are very rich in coal. From borings carried out by the Japanese it appears that in one part of the Fuchung district the seams of coal are 100 ft. thick. Moreover, the quality of the coal is extremely good. A sample of inferior Manchurian coal, recently analyzed at Tokio, was declared to be superior to first-quality Japanese coal.

France.—The production in 1906 was 34,313,645 metric tons, which shows a decrease of 1,734,619 tons from 1905. This was due largely to serious accidents that curtailed the output from some of the largest producers. The loss in tonnage will likely be regained if no further calamities occur to cripple production in 1907. The preliminary report for the Nord and the Pas-du-Calais, the two more important coal-mining districts in France, is as follows, in metric tons:

	1904	1905	1906
Nord	....	6,729,840	6,243,086
Pas-du-Calais	....	17,543,215	15,828,688
Total coal	21,718,269	24,273,055	22,071,774
Coke made	....	1,772,793	1,678,823
Briquets made	....	972,129	956,333

The heaviest producer in the Nord in 1906 was the Anzin company, with 3,102,465 tons; in the Pas-du-Calais the Lens, with 3,030,258 tons.

FOREIGN COAL TRADE OF FRANCE.
(In metric tons.)

	Exports.			Imports.		
	1904	1905	1906	1904	1905	1906
Coal	1,120,153	1,658,680	1,372,870	10,884,468	10,513,920	14,354,320
Coke	160,580	242,040	178,380	1,656,361	1,632,710	2,257,860
Briquets	66,788	88,940	130,510	528,107	399,390	547,250
Total	1,347,521	1,989,660	1,681,760	13,068,936	12,546,020	17,159,430

After careful soundings it has been proved that the coal-beds of French Lorraine are extensive and of much importance. At a depth of over 1000 m. there are several important seams of coal of over 2 m. in thickness. At 896 m. depth two seams have been found with a thickness of 2.65 m. (8 ft. 6 in.). As coal is considered workable down to a depth of 1000 m. at least, it is believed that the Lorraine coal basin will now be considered by the least sanguine as a workable scheme. Similar coal deposits have been located on the German side of the frontier and German companies are under formation for the exploitation of the territory.

Germany.—Great activity was shown in the various coalfields of Germany during 1906. The increase in production amounted to 19,722,590 tons, which is the largest advance made since 1900. Deducting the excess of exports over imports in 1906, the approximate consumption of coal for the year, in all forms, was 191,613,907 tons. The activity of the iron trade and of most manufacturing industries explains the large gain.

COAL PRODUCTION OF GERMANY.
(In metric tons.)

	1904	1905	1906
Coal	120,948,050	121,298,607	137,117,926
Brown coal	48,500,222	52,512,062	56,415,333
Total mined	169,448,272	173,810,669	193,533,259
Coke made	....	16,491,427	20,260,572
Briquets made	....	13,074,682	14,500,851

Indo-China.—The most important coal mines in Indo-China are those at Hongay, owned by the Société Française des Charbonnages du Tonkin, which employs about 3000 workmen, Chinese and Annamites. The financial position of the Société is flourishing. During 1906 the coal extracted was approximately 230,980 tons, of which 106,289 tons were manufactured into briquettes, some of which were shipped to Hong Kong. Another mine, the Kebao, is worked by Chinese and Annamite contractors. The output of the Kebao mine is about 6000 tons yearly. The Schoedelin mine has been worked by contract since October, 1902, by the Heritiers Sarran. The approximate output is 5000 tons a year. The coal is of fair quality, but friable, and will not stand handling or storage.

Mexico. (By C. A. Bohn).—In the Mexican coal fields several large companies have been formed, the principal of which are the Mexican Coal and Coke Company, at Esperanzas, Coahuila, producing 50,000 tons of bituminous coal and 8000 tons of coke; and the Sabinas Coal Mining Company, at San Juan de Sabinas, just north of Esperanzas, organized in the early part of 1906, which is developing and opening up its 30,000

acres of coal lands. It is awaiting the completion of the railroad from Baroteran before placing its bituminous coal on the market. For the opening of the semi-anthracite beds in Puebla, between Tulcaningo and Guachinango, the National Railroad has ordered a survey to be made, but another year will have passed before much can be done there. On the other deposits known to exist in the States of Sonora, Vera Cruz, Oaxaca, Michoacan and Guerrero little or nothing has been done because of the lack of transportation facilities.

New Caledonia.—In order to encourage the development of the coal resources of New Caledonia, no export duty is to be levied on coal, and coal lands will be subject to the payment of only 50c. per hectare, whether worked or not. A company has begun operations with some success at Nondoue, about 15 miles from Noumea, and about five miles from the present terminus of the railway, with which it is to be connected by a branch. The present company, however, hardly possesses sufficient capital to work on a large scale, and the great problem is whether any adequate demand or outlet can be found for the coal.

Panama.—Coal seams are found in many parts of this republic. In the province of Bocas del Toro a very good grade of bituminous coal was discovered, but it was so far back in the interior that it was not a paying investment. In the Isthmian Canal Zone coal veins were found by the French during their occupancy. This statement seems to be verified by the reopening of an old coal seam on the Negrito river. Recently two other places on the same property, but on a lower level, were bored, and it was reported that a coal seam 3 ft. thick was discovered.

South Africa.—At the present time the carboniferous deposits of South Africa are known to have a combined area of over 56,000 square miles, which estimate will probably be greatly increased, for coal is found practically all over South Africa above a certain altitude. The known deposits lie for the most part at a depth of not more than 500 ft. from the surface, and are easily worked, while the seams range from 10 ft. to 20 ft. in thickness. The most important coal producer of the South African colonies is the Transvaal. In this colony the coal is worked at shallow depths.

The measures are generally free from faults and dislocations, but are subject to intrusion of igneous dikes (whinstone and dolerite), sometimes 1000 to 1500 ft. wide, but not perhaps to the same extent as the seams in Natal. The igneous dikes have no perceptible effect upon the general dip of the seams, but the quality of the coal changes within a certain radius. An average analysis of the coal in this field shows 62 per cent. fixed carbon; 15.1 per cent. volatile matter; 0.6 per cent. moisture; 0.3 per cent. sulphur; and 22 per cent. ash. The coal is dull, of splinty appearance, weathers badly, and is liable to spontaneous combustion.

Little or no timber is required, hardly any water is encountered, and fire-damp is practically unknown.

To sum up briefly the coal situation in South Africa it may be stated that the seams vary from 30 in. to 40 ft. in thickness and from 5 to 25 per cent. in ash. There are 22 collieries in active operation with a nominal capital of nearly $25,000,000. Out of the 14 producing companies, four are paying dividends.

The new coaling appliances at Durban, Natal, were expected to be ready for use about the end of March, 1907. The plant will load into vessels at the rate of 400 tons per hour, either direct from trucks or from storage bins of a capacity of 10,000 tons. The coal will be either dumped into ships' holds from trucks, lifted bodily and tipped sideways, or handled by transporters, which carry drop-bottom buckets of six tons' capacity. The dumper can deal with loads of nearly 80 tons. Loading and trimming will be done by the harbor department at the rate of 1s. 3d. (30c.), per ton for bunkers, and 10d. (20c). per ton for holds. Two steamers can lie alongside and coal at the same time.

Large deposits of coal are found in the eastern part of Cape Colony, and are thought to be an extension of the Natal and Zululand coal fields. Although the quality of the coal is inferior, the demand for it is considerable, due principally to the fact that other coals have to be brought such a great distance. Since 1899 the output has been practically stationary, amounting to 207,403 tons in 1903, bringing an average price of $4.14 per ton. About 93 per cent. of the labor employed is native, each man producing on an average 95.7 tons of coal during the year.

COAL PRODUCTION OF NATAL. (a)
(In tons of 2240 lb.)

	1904	1905	1906
Bunker	383,000	563,000	488,000
Exports by sea	12,000	43,000	215,000
Consumed in Natal or exported overland	462,000	523,000	527,000
Total	857,000	1,129,000	1,230,000

Spain.—The outputs of the various provinces in 1906 were as follows:—Oviedo (Asturias), 1,817,147 tons; Cordova, 432,962 tons; Ciudad Real, 295,646 tons; Leon, 244,700 tons; Seville, 165,000; Valencia, 112,281 tons; Gerona, 12,000 tons. The greatest increase was in Leon, 41,825 tons; the output of the Asturias shows a decline of 24,960 tons as compared with 1905. The chief producers in 1906 were the Sociedad Metalurgica Duro Felguera (529,407 tons); Sociedad Hullera Espanola (375,000), and Sociedad Fabrica de Mieres (256,841 tons). The production of anthra-

cite in 1906 was 159,519 tons, as compared with 163,275 tons in 1905, of which 96,080 tons were raised from the mines of the Sociedad de Penarroya in Cordova, and 41,300 tons from the Minas de Santa Lucia (Leon). The chief producing center of lignite is the province of Teruel (74,240 tons), and Barcelona (66,500 tons). The consumption of coke amounted to 888,028 tons, as against 821,100 tons in 1905, comprising 449,927 tons made at collieries or ironworks (as against 465,812 tons in 1905), 210,000 tons made at gasworks, and 228,101 tons imported.

COAL STATISTICS OF SPAIN.
(In metric tons.)

	1905	1906		1905	1906
Production of coal	3,075,741	3,079,736	Equivalent of coke	207,761	326,184
Production of lignite	168,994	204,840	Coal exported	2,171	4,398
Imports of coal	2,208,398	2,199,091	Consumption	5,656,723	5,805,453

Turkey.—Of the numerous coal deposits in both European and Asiatic Turkey, only a very few have been carefully investigated, and no accurate geological chart exists even of the important coal basin at Heraclea on the Black Sea. This district extends along the southern shore of the Black Sea for a distance of about 95 miles from southwest to northeast, between Heraclea and Amasry, the breadth of the deposit being nearly four miles in some places. The first European report of this field was issued by Schlehan about 50 years ago, and it has since been examined by French and Belgian engineers. Its extent and the conditions of the seams are, however, only imperfectly known, but there appears to be a number of workable seams of good coal, ranging in thickness up to about 13 ft. The quality has been approved by different naval experts, but it will not stand a long voyage. It is bituminous in character, and the quality improves as the distance from the coast increases. Analyses, made some years ago, show that the percentage of volatile constituents varies from 27.6 to 45, the ash from 4 to 11, and the fixed carbon from 51 to 64. The coal obtained at greater depths is said to be well adapted for metallurgical purposes, coking and steam raising; coke is shipped to the Laurium smelting works (Greece) and to foundries at Marseilles, France.

United Kingdom.—The production of coal in the United Kingdom, showed a greater advance during 1906 than has been recorded in any recent year. The total output amounted to 251,050,809 metric tons, or an increase of more than 11,000,000 tons over the production during 1905. The exportation, including coal sent abroad for the use of steamships engaged in foreign trade, is given in the accompanying table.

BRITISH COAL EXPORTS.
(In tons of 2240 lb.)

	1904	1905	1906
Coal	46,255,547	47,476,707	55,599,771
Coke	756,949	774,110	815,224
Briquets	1,237,784	1,108,445	1,377,209
Total exports	48,250,280	49,359,272	57,792,204
Steamer coal	17,190,900	17,396,146	18,590,213
Total	65,441,180	66,755,418	76,382,417

The Coal Markets in 1906.

Bituminous.

Chicago. (By E. Morrison).—In general, 1906 was similar to previous years in the history of the Chicago coal trade. There was the same oversupply of coal from Illinois and Indiana mines, prices continued low and the last four months of the year saw the same difficulty in getting supplies because of car shortage on the railroads. The year, however, contained one notable variation from the normal in its record for the months of April and May, when mining in the western fields was almost wholly suspended because of labor troubles. It was shown in these two months that stores of bituminous coal could be provided for at least two months, without serious losses from slacking or inconvenience to the users of coal. The prices of western bituminous coal rose 20 to 80 per cent., but nearly all the large consumers had laid in stores sufficient for two or three months' needs, and the amount of coal sold at the speculative prices was small. In the other months of the year prices were about the same as in 1905 for the leading grades of bituminous. Anthracite, after the end of labor troubles at the mines, sold at the same prices as in 1905.

Undoubtedly, too many bituminous mines have been worked in Illinois and Indiana, in the last two or three years. The annual car-shortage troubles of the autumn are in one respect helpful to the coal trade of Chicago; they keep back from the city shipments that under normal conditions flood the market with coal that must be quickly disposed of to avoid demurrage charges. Chicago is the natural market-place for coal not sold when it leaves the mines of the two States in question, and some arrangement that would regulate shipments would greatly benefit both operators and wholesale dealers.

Approximately 11,000,000 tons of bituminous and anthracite coal are received by the Chicago market yearly, of which about 3,000,000 tons are reshipped to points outside the city. A large amount of coal that never reaches the city is, of course, handled by local dealers. About 15 per cent. of the total received in Chicago is anthracite and 60 to 65 per cent. is bituminous from the Illinois and Indiana fields, the rest being

from West Virginia, Pennsylvania and Ohio. Receipts of anthracite by lake were 208,420 tons less than in 1905.

The range of average car prices of Illinois and Indiana bituminous, for the year, is shown in the accompanying table.

PRICE OF COAL AT CHICAGO.

Month.	Lump and Egg.		Run-of-Mine.		Month.	Lump and Egg.		Run-of-Mine.		Month.	Lump and Egg.		Run-of-Mine	
Jan......	$2.00	2.75	$1.30	1.75	May......	$2.50	3.00	$2.25	3.00	Sept......	$2.00	2.50	$1.50	1.90
Feb......	2.10	2.85	1.30	1.75	June......	2.00	2.30	1.70	2.25	Oct......	2.00	2.85	1.50	2.25
March....	2.20	3.00	2.00	2.50	July......	1.75	2.50	1.50	2.15	Nov......	2.20	3.00	1.75	2.25
April......	2.75	3.00	2.50	3.00	Aug......	1.60	2.40	1.30	1.85	Dec......	2.20	3.25	1.75	2.25

Smokeless coal ranged $3.25@3.50 for mine-run, with the standard quotations for the last half of the year $3.40 for run-of-mine and $4.30 for lump and egg, these prices applying to Pocahontas and New River. Hocking valley coal ranged $3.15@3.40 for run-of-mine, with lump running as high as $3.75 toward the close of the year. Coals from east of Indiana in general were scarce in the last half of the year, because of the car shortage. Anthracite sold at the standard prices of $6.25 for grate and $6.50 for egg, stove and chestnut, subject to discounts of 30c. for June, 20c. for July and 10c. for August. The customary discounts of 40c. for May and 50c. for April were not announced, owing to the labor troubles at the mines.

Cleveland. (By G. W. Cushing).—The coal trade in the Lake region for 1906 presents a study of the development of forces which have lain dormant, but apparent, for several years. For one thing the year was conspicuous for the development of productive capacity of mines out of proportion with the demand and beyond the capacity of the railroads to take care of it. For another thing it brought into stronger relief the possibility of new fields competing with Ohio, West Virginia and Pennsylvania in the markets those States have dominated for many years. The crux of the situation naturally rested upon the strike of the bituminous coal miners in April.

When the year opened there were already stong premonitions that a coal strike was ahead. This brought about buying on the part of consumers to collect a stock which would last for several weeks. This made a good market for coal, which was fully up to the expectations of the operators and the ability of the railroads to furnish cars. At first there was talk that the anthracite and bituminous miners would join in a general strike in Pennsylvania, Ohio, Indiana and Illinois. West Virginia and Kentucky were considered out of this combination, being largely non-union. The Mine Workers' officials evidently arrived at the conclusion that public sentiment would not support a strike of such general

nature, and in consequence the struggle was confined to the bituminous mines. The men wanted 10c. a ton advance. West Virginia and Kentucky hoped this would be granted, since the differential on coal rates is only that amount in favor of the Ohio and Pennsylvania fields, and this addition to the cost of production would put those two States on an equal footing with Ohio and Pennsylvania. But the Pittsburg Coal Company had contracts for the delivery of a specified amount of coal per week to the Steel Corporation and had to yield to the demands of the miners, forcing the independent mines in Pennsylvania to the same position. Part of the Ohio mines held out until June and some until the middle of July. When an agreement was reached to resume operations it was found that half of the season of navigation was gone and yet there were contracts, in the closed districts, to ship an aggregate of nearly 4,000,000 tons to the head of the Lakes, to say nothing of supplying the strictly local demand. The shippers by Lake immediately went into the market to tie up vessel capacity in quantity for the last half of the season and succeeded in getting about all the boats they needed. Within six weeks, however, there set in a shortage of railroad cars which increased as the regular fall trade opened, and the business in iron and steel kept enlarging. By October the car shortage was serious and by the middle of November it was admitted that shippers by Lake would not be able to fill their contracts. This was made finally impossible when winter set in about Nov. 15, and brought the season of navigation to a close. The car shortage crippled West Virginia and Kentucky as well as Ohio and Pennsylvania.

The early demand of the consumers to provide a supply against a strike kept prices steady until the strike was declared. Then there was a rapid rise, which continued only long enough to determine that most consumers had a supply which would last for at least six weeks and in most cases three months or more. Then prices dropped rapidly and the market was quiet. When the car shortage set in in September prices began to move up again and by the middle of October mine-run steam coal was selling at $1.50 at the mines and slack at $1. This was due to the fact that the supply of cars at no time permitted mines to run more than one-third their capacity while in some instances production was only 20 to 25 per cent. of normal. Prices were also influenced at one time—principally in midsummer—by the fact that many consumers showed no discrimination between the various grades of coal, having improved their furnaces to use either fine or lump coal. This change promises to have even a greater influence on prices in the future.

The Hepburn law had its effects also. In Ohio it brought about a ruling by the State Railroad Commission that the old practice of reserving certain points, local to certain railroads, as markets for mines along those

roads, is illegal, thereby broadening the market for the product of all mines. In Kentucky it had the effect of bringing the railroads to terms. In that State the railroads had a peculiar grasp of the coal situation, refusing to put switches in operation when those led to mines that competed with the railroad-controlled mines. The new law gave the operator redress for such grievances, and brought about the opening of some new mines. The Kentucky fields are extensive, and the absence of unionism there gives the Kentucky operator a chance in the markets north of the Ohio which he is almost sure to develop, if the car situation will ever permit. In the Lake region many shippers filled their contracts, while others failed. The shipments were equal to those of former years, but not up to the demand in 1906, largely because the market has broadened, owing to developments in northwestern Canada. The total movement of bituminous cargo coal through the south shore ports of Lake Erie for 1906 was 13,360,000 tons, compared with 10,455,000 tons for the season of 1905. To this total Pittsburg contributed 8,250,000 tons in 1906, against 6,475,000 tons in 1905. West Virginia territory contributed 2,690,000 tons in 1906, against 2,075,000 tons in 1905. The Ohio fields contributed 2,410,000 tons in 1906, against 1,900,000 tons in 1905.

Pittsburg. (By S. F. Luty).—The production of coal in the Pittsburg district in 1906 was the greatest in its history, but the returns to stockholders of the large companies and to independent interests were not entirely satisfactory. This was due to the fact that many large annual contracts were taken in January and February at low prices, as it was expected the mining rate would be reduced. Instead, the United Mine Workers of America forced an advance of about 6 per cent., dating from April 1, and the cost of production showed a decided increase over previous years.

A general strike of miners was threatened early in the year unless a substantial advance was conceded, but operators did not seem to regard the matter seriously and continued to quote low rates for deliveries extending through the year, the minimum price being 90c. for mine-run coal, and many contracts were made at $1 a ton in January, after the national convention of the United Mine Workers at Indianapolis had made a demand for an advance of 12 per cent. Iron and steel manufacturers, fearing a suspension, began to store coal and all the mines in the district were unusually active in the first three months.

When a general tie-up of the mines of the country appeared certain President Roosevelt intervened by writing to Francis L. Robbins, chairman of the Interstate Coal Operators' Association, and John Mitchell, president of the United Mine Workers, urging them to endeavor to reach an agreement. It was due to the efforts of these men that a compromise was made at a reconvened conference held on March 19 at Indian-

apolis. The miners withdrew the demand for an increase of 12 per cent. for the year and agreed to accept a raise of 6 per cent. and make a contract for a period of two years, dating from April 1, 1906.

Western operators objected, but Mr. Robbins signed the agreement for his company and its subsidiary interests, including the Monongahela River Consolidated Coal and Coke Company, and all the mines of these concerns and a number of independents continued in operation. On April 6 the other independents gave up the fight and before April 10 every mine in the Pittsburg district was in full operation. The suspension in other parts of the country continued for a few weeks longer, when it was seen that Pittsburg operators were getting the business and there was a general resumption.

A shortage of railroad cars prevented the full operation of the mines throughout the year, but despite this handicap the production for 1906 was between 46,000,000 and 47,000,000 tons, compared with 42,000,000 tons for 1905, 31,000,000 for 1904, and 18,000,000 tons in 1897. The Pittsburg Coal Company, the leading interest, produced 16,000,000 tons, and the Monongahela River Consolidated Coal and Coke Company, the large river interest, 6,000,000 tons.

Shipments to the Northwestern and Southern markets from this district were greater than in any previous year. The shipments to the lake ports during the season amounted to 9,200,000 tons. In 1905, 7,460,000 tons were shipped and in 1904, 6,058,000 tons. The shipments to Southern ports aggregated about 3,500,000 tons, of which 2,500,000 tons were sent out by the Monongahela River Consolidated Coal and Coke Company. The rest of the tonnage of this big company went to the mills along the Monongahela and Allegheny rivers and to other consumers in the Pittsburg district. John H. Jones, president of the Pittsburg-Buffalo Company, declares that if the railroads had been able to furnish necessary transportation facilities the production in the Pittsburg district would have been fully 3,000,000 tons greater in 1906.

An unusually large number of big contracts were entered into early in 1906 for extended deliveries. One by the Pittsburg Coal Company with the Pittsburg Steel Company called for 1000 tons daily for a period of six years. The Monongahela River company received a contract from the Midland Steel Company for 250,000 tons annually for five years. All contracts made by the big river interest for the year with consumers at Southern ports were at low prices, as an advance in the mining rate had not been taken into consideration. In December the company gave notice that all annual contracts to be made in January, 1907, would be at an advance which will range from 10 to 20 per cent.

Many improvements were made during the year and there are now but few mines operated by old methods. The Pittsburg Coal Company

installed modern machinery and equipment at nearly all of its mines. The Pittsburg-Buffalo Company in May began sinking two large shafts on a 14,000-acre tract in Washington county, which is included in the Pittsburg district. These shafts are to be the finest in the world, when completed in the spring of 1907. Each shaft will have an actual working capacity of 1250 tons an hour. The coal will be used exclusively for coking and a number of ovens are to be built. The Pittsburg Coal Company greatly increased its coke production in 1906, and is still at work on extensions. In February this company sold 262 acres of coal land to the W. J. Rainey Company at $1222 an acre, having paid $280 an acre for the property a few years ago. The company also disposed of its interest in the Western Coal and Dock Company and the Whitnall Coal Company.

Preparations are being made by producers for a large output in 1907. There will be no labor troubles, as the present scale will continue in force for another year, dating from April 1. It is the same scale as was in force in 1903 when the pick-mining rate was 90c. a ton. A reduction to 85c., or 5.5 per cent., was made for a period of two years dating from April 1, 1904, but the rate of 1903 was restored by the new agreement.

PRODUCTION OF PITTSBURG COAL COMPANY
(In tons of 2000 lb)

	1904	1905	1906
Pittsburgh district	12,783,067	12,926,954	17,628,396
Hocking district	1,349,428	1,371,120	1,415,920
Total coal	14,132,495	14,298,074	19,044,316
Coke made	206,005	355,873	429,076

Anthracite.

Seaboard.—Like the anthracite trade, the seaboard bituminous trade was fairly uniform throughout 1906. The summer dullness prevailed somewhat longer than usual, consumers in the East being apparently unwilling to lay in stocks early, as all possibility of strikes had passed over, and they felt sure of getting coal when they needed it. This was followed in the fall by a rush to buy, and the closing of the year was an active period. The strike in central and western Pennsylvania did not seriously affect the trade. In the last quarter of the year there was a good deal of trouble from shortage of cars and delays in transportation. This did not affect the trade in the East to the same extent as in the West, though it was sufficiently annoying. The coastwise traffic, which supplies a large part of eastern New York and most of New England, showed few changes. Vessels were in good supply nearly all the year, and the fall season was comparatively free from severe storms, while

ice did not form until very late in the shoal-water ports of the East. The sailing vessels and barges, in which this traffic is mainly carried, had generally a good year.

The decrease of nearly 4,000,000 tons in the production of anthracite coal during 1906 does not signify any decrease in consumption. The shipments of 1905 were swollen by large quantities of coal from the mines in November and December, which were stored at various points along the lines, in anticipation of a strike. The suspension after April 1, pending negotiations between operators, brought the shipments to market down to 488,439 tons in April, and 3,276,310 tons in May; but at no time was there any shortage in supplies at consuming points, the stored coal being sufficient to meet the demand until full production was resumed. The movement of the miners failed, the settlement finally effected being a continuation of the terms arranged by the Anthracite Coal Commission three years before, with a very few minor modifications.

If exact figures could be compiled, it would probably be found that the consumption of domestic coal was slightly less than in 1905. The demand for steam sizes was remarkably steady during 1906 with an increasing demand at its close. Of the steam sizes, pea coal is finding an increasing use for domestic purposes. In November the Philadelphia Coal Exchange decided that pea coal should hereafter be considered a domestic size, selling at 25c. below chestnut. This action has not been followed in New York or other cities.

The tidewater prices of anthracite continued unchanged throughout 1906 at the list figures of $4.75 for lump, $5 for egg, stove and chestnut. The usual discounts of 50c. in April and 40c. in May were not given, owing to the suspension. The discounts of 30c., 20c. and 10c. for June, July and August, respectively, were given. The tidewater prices of steam sizes were fairly uniform, closing at $2.80@3 for pea; $2.25@2.50 for buckwheat; $1.45@1.50 for rice, and $1.30@1.35 for barley.

SHIPMENTS OF ANTHRACITE.
(Tons of 2000 lb.)

	1904		1905		1906	
	Tons.	Per ct.	Tons.	Per ct.	Tons.	Per ct.
Reading	11,399,362	19.8	12,574,502	20.5	11,258,295	20.2
Lehigh Valley	9,611,426	16.7	10,072,120	16.4	8,536,254	15.4
N. J. Central	7,201,276	12.5	7,983,274	13.0	6,983,217	12.5
Lackawanna	9,333,069	16.3	9,554,046	15.6	9,201,875	16.5
Delaware & Hudson	5,276,797	9.2	5,640,628	9.2	5,346,695	9.6
Pennsylvania	4,765,963	8.3	4,890,635	8.0	4,856,004	8.8
Erie	5,711,173	9.9	6,225,622	10.1	5,636,537	10.2
N. Y., Ontario & Western	2,646,730	4.6	2,864,096	4.6	2,444,273	4.4
Delaware, Susquehanna & Schuylkill	1,546,476	2.7	1,605,378	2.6	1,435,445	2.4
Total	57,492,522	100.00	61,410,291	100.00	55,698,595	100.0

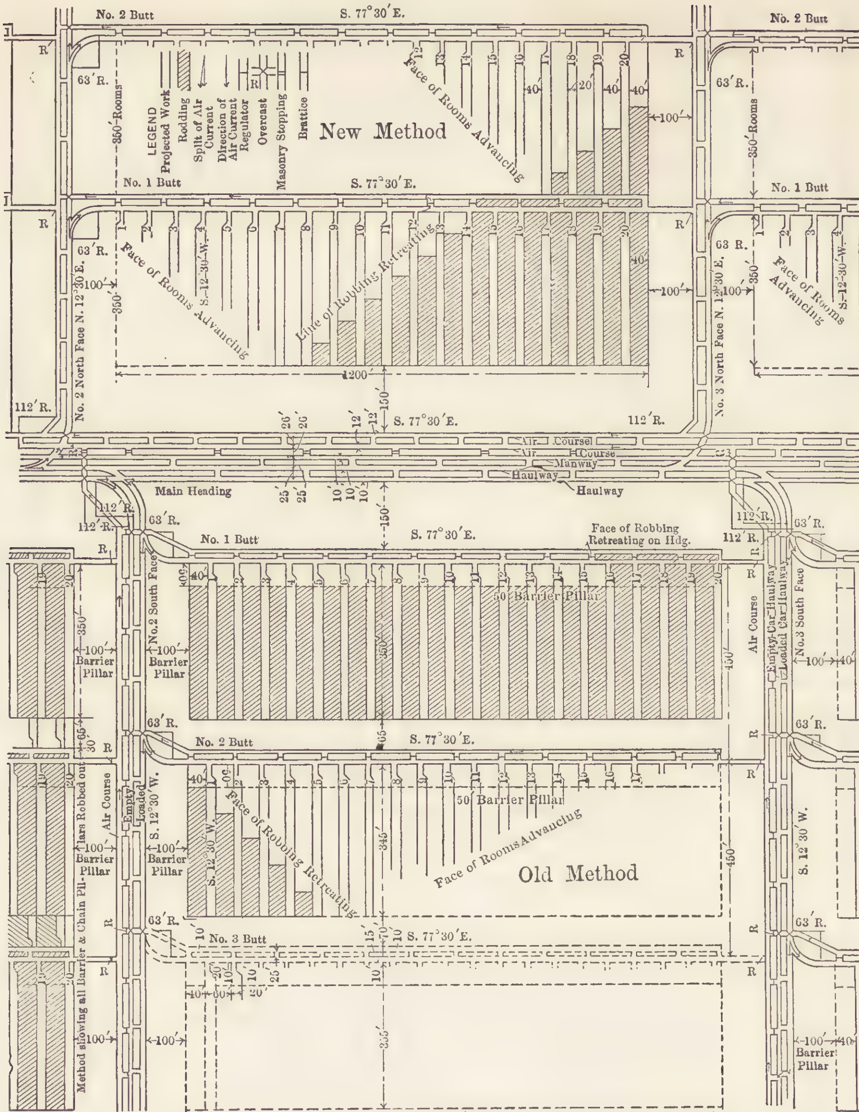
Recent Practice in Coal Mining.

Great progress was shown during 1906 in the methods of mining used by some of the larger companies. It is no longer the practice to try to reduce the first cost of mining by sacrificing all regularity. Instead of driving entries water level to prevent any expense from cutting or filling, the scheme is to now drive all headings and rooms on carefully placed centers, with certain limits as to the distance of driving and the width of the room or entry.

Some bituminous mines are now procuring from 85 to 95 per cent. of all the coal contained in each acre of land. This is being accomplished by working the seam on what is known as the panel system with 15 to 25 rooms to each panel, and by then systematically robbing all the pillars that have been left. The accompanying maps show two ideal systems as carried on in West Virginia and Ohio. The Fairmont system is adapted for good roof, where pillars will be robbed on the retreat; the Hocking system is best used where the roof is bad and where systematic robbing is prevented by an overlying stratum of quicksand that would come in and fill the workings.

In the anthracite coalfield the problems of mining are more numerous, and since the various companies are now turning their attention to mining thin seams of inferior coal, a greater number of difficulties are encountered. Anthracite mining will not admit of such a regular system of working as can be inaugurated in most bituminous districts; however, the development is made to conform as nearly as possible to the general plans laid out. Last year, in the anthracite field, for every ton of coal produced, 8.5c. was spent for timber and mine props. This immense drain on the available timber supply has caused the operators in this field to turn their attention to methods of preserving the timber that is used. The coming year will probably witness many successful experiments that will lead to modifications and better results in methods of preserving mine timbers.

Since much success has attended the practice of washing and selling the culm that was formerly considered nothing more than waste coal, the operators are now turning their attention to the fine dust that results from this culm washing. The practice of flushing this fine dust into the mine is likely to be entirely discontinued, especially if it is found that this fine coal can be utilized profitably in making briquets. One other practice that is meeting with general disapproval is that of connecting the underground workings of adjacent mines. Not only is this dangerous, but on closer study it is said to be unattended by the increased economy which was thought to result from the sinking of fewer shafts. It is to be hoped that legislative action will prevent this evil practice from being further extended.



PLAN OF MINE DEVELOPMENT AS CARRIED ON BY THE FAIRMONT COAL COMPANY IN WEST VIRGINIA.

In fighting the mine fire at the Warrior Run colliery, the anthracite managers have for the first time made an effort to analyze carefully all the gases besides taking daily temperatures and other data that may throw light on the action of such fires, and furnish information that will be useful in future efforts along this line.

Last year witnessed a continuation of the struggle for supremacy between electricity and compressed air for driving coal-mining machines. The advocates of both of these systems are ready to prove the advantages of their favorite apparatus.

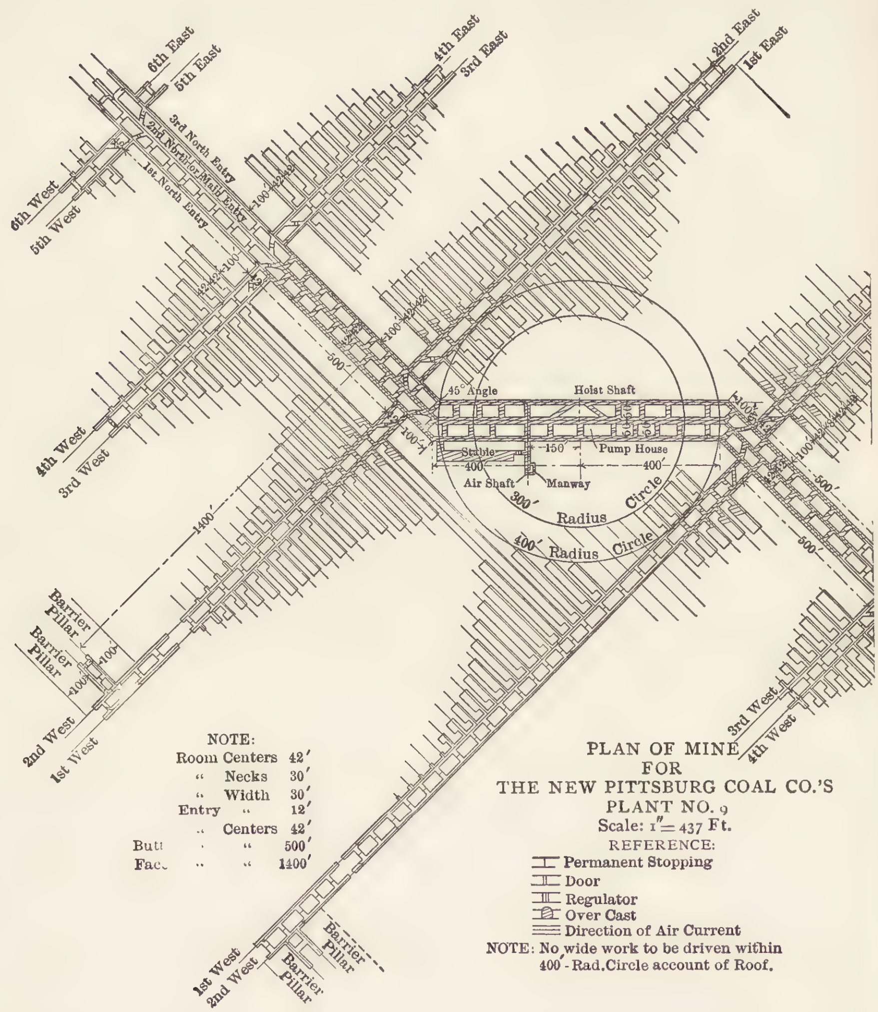
Haulage and Hoisting.—For hauling coal, electric motors seem to be meeting with the greater favor. Many of the new installations have been of the third-rail type, while the greatest advancement along this line has taken place in what are known as gathering motors, which carry their cable on a self-coiling reel, and since they are compact, often standing no higher than 30 in. above the rail, they are particularly useful in collecting cars from the face of rooms. Where the haulways in a mine are on a grade of from 8 to 14 per cent., rack-rail haulage has proved wonderfully successful, and the future will probably witness the installation of many such haulage systems.

One of the newer innovations is the steel mine car. The manufacturers of these cars are very optimistic, and stand ready to prove to operators that the day of the wooden mine car is past. One point in this connection is worthy of mention, and that is the bad practice of spragging mine cars; progressive managers now prefer to have all such vehicles supplied with a steel brake.

In European coal mines, a number of operators have installed electric hoists which seem to be meeting with much success; in this country, however, the hoisting power still continues to be furnished by steam. Successful winding drums are now built so as to combine the advantage of both the spiral and the cylindrical forms. By having the first four or five revolutions on a gradually increasing diameter, the engine is enabled to get under speed quickly, and the retardation at the end of the hoist is considerably lessened. The objection to drums that are entirely of the conical shape is that the increased width introduces more inertia to be overcome.

In connection with mine cars and other machinery, it is indeed well that superintendents are giving more attention to proper lubrication. Only recently when visiting a large and supposedly modern mine, I watched the men drop the loaded cars, that had just been hauled out of the mine, upon the tipple. The approach to the tipple was 200 ft. long and was built on a 1.75 per cent. grade which had in previous years been sufficiently steep to cause the loaded cars to run by gravity upon the tipple. The fact that a man now had to push each car almost all the way to the dumping point attracted my attention, and subsequent investigation proved that the whole trouble was caused by the cars not being properly lubricated. It would be interesting to know what amount of power is being needlessly expended in hauling these rickety, poorly oiled cars through that mine every day.

Timbering.—In supporting the roof of an inclined seam, the props should neither be set vertically, nor at right angles to the dip of the vein; but rather at an angle slightly less than at right angles to the dip of the seam. This method is preferable because the tendency of the roof is to slide down hill, and if the prop is set as here suggested, the slipping of



the roof will tend to drive the support into a line at right angles to the dip, which causes the timber to tighten and afford greater support.

In a mine where considerable timber is used, the economical gain from the systematic drawing of props after the coal has been mined, not only results from the fact that such timber can be used again, but has also

the further advantage of causing the roof of the seam to settle more steadily and regularly on the gob. The practice of carelessly neglecting to withdraw timber may cause a sudden settling which sometimes results in a squeeze, or has some other disastrous effect.

Where considerable timbering is necessary, and many props are used to support the roof, it is found advisable to taper the bottom end of a prop before setting. The pointed end of the prop is then set in a hole cut in the floor. The advantage of this method lies in the fact that the weakest point of the prop is near the floor, and consequently, when the weight of the roof comes on, the point of the prop crushes or spreads, instead of the timber breaking in the center. The prop can now be removed, and after cutting off the bottom, can be re-pointed and used where the seam is lower. It is well to remember, however, that tapered props cannot be used satisfactorily when the bottom is soft, since the pointed end, instead of crushing, sinks further into the strata. The one disadvantage of tapered props is the difficulty in withdrawing the timber.

SYNOPSIS OF BITUMINOUS MINING METHODS.

Locality.	Width Main Entry	Width Cross Entries.	Room Centers.	Room Neck Width and Length.	Room Width.	Room Length.	Pillar Between Main Entry and its Air Course.	Pillar Between Cross Entry and its Air Course.	Distance from Cross Entry to Cross Entry.	Distance Main Entry to Parallel Entry.	Barrier Pillar Protecting Main Entries.
Fairmont, W. Va	12	10	60	10x20	20	350	26	25	1,400	450	150
Southern W. Va.	10 & 12	10	54	8x24	30	250	24	20	540	1,000 to 1,500	75 to 150
Hocking Valley. Ohio.........	12	12	42	8x30	30	244	42 & 30	30	500	1,400	100
Western Penna.	8 to 10	8	42 to 52	10x24	12	300	52	25 to 50	1,000	400	60
Alabama.......	8 to 16	8	42	8x21	25	270	30	15 to 30	300	800 to 1,400	20 to 60
Indiana........	8	8	30 to 50	8x20	20 to 40	260	18	12 to 20	550	1,000	12
Illinois.........	9	8	32	9x18	26	146	20	12 to 18	300	1,000	40
Maryland.......	8	8	65	None	13	100 to 800	40	30 to 40	1,000	....	250
Colorado.......	8	9	40	9x20	20	277	40	30	600	1,400	100

Ventilation.—In ventilating our mines, no radical improvements have been introduced; the tendency, however, is to increase the number of intake and return airways. Not long ago the general system was to have one main intake and a parallel return airway; but present practice favors at least two intake and two return airways, and in some instances five or six have been driven and are being successfully used.

Modern practice now requires that stone or heavy brick brattices be built to close the crosscuts and separate main airways. Overcasts cost about $48, and are much preferred to doors. Furthermore, when the cost of a trapper-boy is considered, the overcast is certainly cheaper and much safer. Modern managers also prefer to split the current into all cross-

entries, and are specifying that their ventilating fans shall be readily reversible. In slope or drift mines the fan is generally run as an exhaust; while in shaft mines the blower system is preferred on account of hoisting during the winter, when, if the hoisting shaft is used for an intake, the collection of ice on the guide-rods and sides of the shaft during the cold months often causes serious trouble.

One other feature in ventilation has attracted some attention. This consists in the use of booster fans in remote parts of large workings, where from leaky brattices and friction, the air current has lost some of its efficiency.

It is especially important in ventilating an inclined seam, to see that the intake air, which is generally cooler than the return, is conducted at once to the lowest portion of the workings, from which point it gradually absorbs heat and tends to ascend to the higher workings. This system creates a motive column which naturally assists the entire ventilation of the mine.

Among the many advantages resulting from splitting the air current, may be mentioned the fact that by this method a greater quantity of air can be propelled by a given power. When splitting the air is resorted to, the rubbing surface is increased, and as a consequence, the pressure, which varies directly as the rubbing surface, is also slightly increased; on the other hand, splitting the current will reduce the velocity with the same pressure, and since the pressure varies in proportion to the square of the velocity, the reduction in this latter element more than balances the effect of the increased surfaces of contact.

In planning the ventilating system each main split should begin as near the downcast shaft as possible; it is also best to have the lengths of the different splits as nearly equal as possible and to provide each main split with a separate return as far as practicable. In a general way, it may be stated that the limit to the advantageous splitting of an air current is reached when the extra power required to overcome the increased shaft resistance is greater than the power saved by the decreased resistance of the underground roads. It is also inadvisable to split the current again when the velocity of the air at the face is reduced to 300 ft. per minute.

Surface Arrangements.—The improvements that have taken place in our colliery surface plants are too numerous to be dealt with at length. The tendency has been to do away with all outside labor, by substituting mechanical devices, and to concentrate the outside plants as much as possible, thus preventing the loss that results from having to carry steam or other power long distances. Much attention has been directed toward having the intake at a sufficient distance from the tipple or dumping point, so that whatever dust is raised in handling and loading the output will

not be distributed and drawn back into the mine. At one operation, an expenditure of less than $5000 enabled the company to do away with four men who had been employed on the tipple and received from $1.75 to $2.50 per day. This result was accomplished by the installation of self-dumping cages, and by having a heavy pan-conveyor carry the coal that was hoisted from the top of the shaft to the head of the gravity screen.

Pumping.—In coal washeries and other places where water is to be lifted to a hight less than 100 ft., practice is now favoring the installation of turbine pumps instead of the old-fashioned steam pump. Three-throw pumps have met with considerable favor and are now largely used in mine work. They consist of three ram pumps placed side by side on one bed and driven by connecting rods from three cranks, which are arranged on one shaft and set at an angle of 120 deg. with each other. Among the advantages claimed for this style of pump, the strain on the driving shaft is kept almost uniform, and the delivery of the water nearly constant. It is also possible to drive these pumps either by electricity or by wire rope.

Machine Mining.—The use of coal-cutting machines in bituminous mining largely increased during 1906. The State of Ohio may be considered the leading exponent of machine mining, since approximately 75 per cent. of its entire output is produced by coal cutters. Each coal cutting machine in Ohio is now producing, on an average, nearly 8000 tons annually, which record is excellent considering the number of days worked. Much is said about machines producing an excessive quantity of slack coal, but a careful examination of available data seems to show that on an average there is as much slack and coal of the smaller sizes in mines where pick mining is used. One objection, however, is certainly true, viz., machines often mix considerable of the bottom rock or clay with the coal, thus causing the product to run higher in ash.

Beside the less cost of undercutting when mining machines are used, there also results an increased production from a specified length of working face and the employment of fewer men. One advantage thus gained is that a smaller amount of entry has to be constructed and maintained for any given output; furthermore, a mine can be developed more quickly.

To install a 15-machine plant, capable of producing from 600 to 800 tons of coal in eight hours, the cost for the power plant, mining machine, pipes and all other accessories, including the foundation, boiler-setting, and complete installation is about $14,000. The total expense for maintaining and operating such a plant is about $5500 per year as a maximum figure. As to operation, practical results in the Pennsylvania fields show that, on an average, each mining machine will produce about 50 tons of coal per eight hours, cutting 65 ft. of face to a depth of 5 ft.

As to the advisability and economy of using machines in coal mining, much depends upon local conditions at the mine. Where the roof of the coal seam is good and requires little timber, there is no doubt that after considering the additional initial cost, machines are to be recommended. When the roof is bad, the difficulties attendant upon machine mining are: First, the difficulty of moving the machine about in a narrow place where many props are used; second, the danger of miners being caught under a fall of roof, because the noise made by the machines makes it difficult for the workmen to hear the warning sounds that occur previous to a fall of top. For this reason, it is generally best occasionally to stop the machines and listen for any sound that would indicate the working of the overlying strata.

Explosives and Shot-firing.—So far as the selection and proper use of explosives are concerned, the European countries are further advanced than Americans. In Great Britain and on the Continent much attention is paid to the composition of the explosive; this has resulted in compelling the manufacturers to use the maximum care in preparing their product; otherwise it would be debarred from the permitted list. In the United States we seem only concerned with the quantity of powder that a miner shall be allowed to take into a mine.

Before placing a shot, the coal face should be carefully inspected to avoid locating the hole near any crack in the coal or near another bore hole. If these precautions are not observed, a windy shot may result and serious damage occur. Blown-out shots are a source of great danger to coal miners, and numerous precautions should be observed to eliminate the possibility of such an accident. It is advisable, wherever possible, to see that the drill hole does not exceed the depth of under-cut. It is also best to have the hole pointed slightly upward, because all charges when fired have a tendency to rise, and a hole drilled in this manner will not require so much tamping as otherwise. It is also advisable when shots are fired to have the room or entry in the vicinity thoroughly wet so that dry dust will not be raised and ignited.

In driving a rock-heading, the best results will be obtained if the holes are so placed that the line of least resistance is three-fourths of the depth of the hole; also when the powder fills the hole to about one-third its depth. If a 3-in. hole is drilled 8 ft. deep, with its line of least resistance 6 ft., the hole should be loaded to a depth of about 31 in., which would need about 7 lb. of powder. In this connection it may be mentioned that 1 lb. of blasting powder will occupy about 28 cu. in.

Coal Washing.—In the satisfactory operation of a coal washery it is necessary to feed the jigs slowly and regularly. It is also important to have the material jigged to a uniform size, if the separation of the refuse from the coal is to be complete. In breaking coal through rolls, experience has

shown that the waste may be increased as much as 5 per cent. by over-feeding. It has also been proved that the amount of waste is less when the coal is not broken at once into small sizes, but is passed through two sets of rolls, the second set being closer together than the first.

Other experiments in the preparation of coal have shown that by washing that product which has gone over a $\frac{3}{4}$-in. screen, the washed coal contains 40 per cent. less ash, 15 per cent. less sulphur and 5 per cent. more fixed carbon than the unwashed coal. The amount of water used in washing in one instance was about 36 gal. per ton of coal, and the cost of the operation was 2.5c. per ton, for a plant whose capacity is more than 350 tons per day.

Coke Manufacture.—We are on the eve of a great awakening to the immense losses that have resulted from making coke in bee-hive ovens and allowing all gases and by-products to be wasted. If the power that has already been lost in the Connellsville region, through the waste of gases in the manufacture of coke, could be made available at the present time, it is fair to say that the factories of our country would not have to purchase coal to produce power for many years to come. This great and inexcusable waste has been one of the most shocking things in our industrial history.

By-product ovens are coming into greater use, and where operators persist in retaining the principles of bee-hive ovens, they are in many cases modifying the style by building their ovens 25 to 30 ft. long and 6 ft. wide. The by-product oven does not produce a coke with a silvery luster like the products that come from the bee-hive ovens; however, the structure and quality of the by-product coke is not far inferior to that of the old-fashioned product. In a number of instances where ovens have recently been constructed, the arrangements have included a long main flue running behind the ovens and connected to each one by smaller flues, thus catching the gases that have heretofore been lost. These gases are carried to the power house and there burned in gas engines, and used to manufacture steam. A considerable saving with little extra expense is thus effected.

The advantages derived from compressing coal before putting it into an oven to manufacture coke are: A 10 per cent. net increase in the coke output; a firmer, denser, stronger coke, with a higher specific gravity; elimination of the spongy appearance of the top coke; less breeze and a more even size of product. The disadvantages of this method are an increase of about 20 per cent. in the coking time, because of the water present in the compressed coal, and the additional labor cost and repairs to the machinery.

In 1890, statistics showed that on an average in our country, 158 tons of coal were required to manufacture 100 tons of coke; this quantity of coal necessary has steadily decreased, until during 1905 less than 148 tons

of coal were required to manufacture 100 tons of coke. This speaks well for the progress of the industry and the efficiency of operation.

Theories, Problems and Innovations.

Explosions.—The problem of greatest importance that has presented itself to managers and engineers during the last few years is the prevention of explosions. The last 12 months was a period of great effort toward the solution of this problem. Whether the increased loss of lives during 1906 caused this sudden awakening and made us seriously face the conditions as they actually exist, I do not know. The fact remains that the general mining public, if not also our State and Federal governments, is evincing unusual interest in this vital problem.

A few years ago the engineers who believed that an explosion could occur with coal dust as the sole factor were in the minority, while at the present time such an opinion is general. We are now considering whether an explosion of dust can take place without the presence of any flame, and many competent authorities agree that such is the case. We are all ready to acknowledge that where a quantity of gas is present in a mine, the temperature of the gas-containing atmosphere may be so increased by sudden compression, due to a fall of roof or other like cause, that the temperature of ignition may be realized, and the gas exploded. We will also agree that such a pressure may react to a distant part of the workings, and there ignite an entirely separate body of gas. Without any gas present in a mine, there is no doubt that various dust explosions have been started by a blown-out shot, but whether a dust explosion can originate from such compression of the mine air as may be caused by a fall of roof without the presence of some gas, is a theory that we believe has yet to be proved. Dynamite or powder might be so exploded, but coal dust would not answer so quickly to such action.

Prevention.—In providing a remedy for dust explosions, several unique plans are being considered, and some of these arrangements have already been successfully installed. At some mines a reservoir has been placed at an elevation above the workings so as to give a considerable head to water that flows through pipes, which are extended along the main entries and into all partings where dust is likely to accumulate. This plan of sprinkling the mine and wetting the intake air has met with only partial success. It is well known that when a layer of coal-dust is sprinkled with water, the latter collects on top, while the dust underneath remains perfectly dry.

A second method is to haul a large part of the dust to the surface, where it is dumped, and to have parts of the mine absolutely free from dust. This is accomplished by arching certain sections with brick, or by frequently wetting a stretch of entry. By this method, a dust explosion will generally

be confined to some one locality, and will not extend through the workings. Other plans are to have each car of coal pass under an automatic shower before it starts on the main haulway, while still another scheme is to have the fine coal loaded into separate cars which are covered before being moved.

Those managers who have overcome the dust problem by sprinkling salt or the chloride of calcium along the different haulways have been subjected to considerable initial expense, but the result desired has been attained. One ton of calcium chloride, costing $12, will treat 1500 ft. of entry. Salt, costing only $3.50 per ton, will cover almost as great an area.

Safety Chambers.—In attempting to provide greater safety for miners in case of an explosion, much attention has been directed to this phase of mining, and numerous suggestions have been advanced. Some of the larger mines of Austria have prepared a so-called safety chamber underground, which is a strongly constructed retreat having a single entrance with a door that can be made tight against air and water by means of rubber fittings. Each chamber has three cylinders of oxygen, sufficient to enable 30 men to breathe three days. Respiratory apparatus is here at hand for parties leaving the chamber to reconnoiter or signal. In the safety chamber there are foods, medicines, and first-aid appliances. The standard room as here built will accommodate 24 men for a reasonable time while awaiting rescue.

Barometric Pressure.—Recent investigation has gone to prove that barometric pressure has a probable effect upon gases in a mine, and consequently exerts considerable influence on mine explosions. The humidity of the air, which is also closely allied to barometric pressure, has an undoubted effect upon the conditions that govern and make mine explosions possible. In this connection it is likely that the Government will investigate this hypothesis during the coming year, and be of much service to the mining industry. There is no doubt that if through our weather bureau mine managers could know of the movements of areas of high and low barometric pressure, certain precautions might be taken that would result in a reduction of accidents and a material saving of life.

Legislative Action.—The mine inspectors of the larger coal States are urging their respective legislatures to revise the present laws, so that coal-mine development will hereafter be governed by more stringent rules, which will insure greater safety to the men, and as a final result, less worry and expense to the operator. At no place is military discipline so necessary as in the operation of a mine. It would not be bad policy to have our mines policed, and see that all workmen are arrested for such infractions of rules as smoking, or having matches in a gassy mine; or for carrying powder, entering a place marked "danger," neglecting to report the strength of brattices, etc.

The organization of rescue teams is no longer an idle dream, as is proved by a recent meeting held at Scranton, Penn. All the mines of one large company were represented in this event, and the work done by the different teams showed the result of conscientious training, and promises well for the future of this important work.

Considerable discussion in Pennsylvania is now being directed toward a recently proposed law to make any person holding a mine foreman's certificate eligible to the nomination and election for mine inspector. It has always been the custom for anyone desiring to become a mine inspector, first to pass a regular inspector's examination, which examination is much more complete and difficult than the examination for a mine foreman's certificate. This fact makes it evident that the proposed law would lower the standards of efficiency and preparation now required of mine inspectors.

The former practice of having the Governor appoint each inspector from among those who had successfully passed the examination has been replaced by a less satisfactory system where each political party selects a candidate from among those eligible and the office is then filled by public election. Since many of the public at large vote along party lines, or from personal feeling, the position has become a sort of political plum, to be given to the most popular, rather than the most capable.

The President's policy in withdrawing public coal lands, although bitterly opposed in many quarters, is undoubtedly a step in the right direction, and is destined to prevent much of the wholesale robbery that has taken place.

Formation of Coal.—A great many theories have been evolved dealing with the methods and forces that have operated in forming our coal deposits. Of the many ideas recently presented, the one most favorably commented upon ignores the conclusion that our coal beds are the remains of great primeval forests. More careful thought and examination leads us to believe that all of our coal seams have been formed from minute vegetable matter that probably lived and flourished at the bottom of inland lakes, being deposited year after year for a long period, and then covered with layers of sand and other material, while later ages furnished the pressure and heat necessary to complete the transformation. The principal argument in favor of this theory is that the parting between coal seams and the underlying and overlying strata is perfectly uniform and even. The strata forming the floor for beds of coal do not show any traces of the roots or stumps of large trees, nor indeed any evidences of large vegetable growth.

The claim is also made by some geologists that coal is produced by the decay under fresh water of plants belonging to the conifer, fern and

palm varieties; these grew during the Carboniferous age. The density of the vegetation may be taken at 100 tons per acre per century, and if the vegetable matter thus formed were to be compressed to the density of coal, it would give only 0.6 inch-acre of coal; however, four-fifths of this is lost as gaseous product, leaving only one-eight inch per acre, or 1 ft. of coal in 10,000 years.

Ankylostomiasis.—Much interest has recently been aroused by the investigations respecting ankylostomiasis, or miner's worm disease. European countries have studied the subject in great detail, with the result that several laws have been enacted which make it compulsory on the part of both managers and miners to observe certain hygenic rules. The malady resembles very closely progressive anemia. In the initial stages the patients suffer from various digestive troubles. Later they complain of oppression and pain in the epigastric region. The anemic symptoms succeed one another more or less rapidly and the victims become more and more pale. The patient also suffers from alternating vomiting and diarrhœa. After a certain time loss of strength ensues with palpitation of the heart and a feeling of suffocation, while in addition to this there is giddiness and singing in the ears.

During the last few years, researches have developed the fact that the disease is primarily caused by the penetration of ankylostomiasis germs through the skin. This discovery has not only upset all the previously existing ideas respecting the actions of the parasites, but also has definitely proved that the formulas heretofore used for the treatment of the disease require considerable modification. It may be stated generally that 50 per cent. of all the coal mines are infected to a more or less extent, the germs being able to live in the coal dust and mud in the workings. The germs never, however, arrive at the mature state in the mines; it is in the human body only that they can reach the adult stage. The two common ways in which the miners are affected are: (1) when covered with dirt into which the parasites have found their way, the workmen eat their meals and swallow possibly some of the germs which find their way to the intestines; (2) when the miners are at work the germs in the mud and dust with which the men are covered find their way through the pores of the skin, thence to the veins, and are carried in the blood to the heart, afterward passing to the lungs and finally to the intestines where they thrive and grow into adult worms.

The laws that have been enacted to eliminate the disease require that properly equipped closets, to the number of one to every 25 workmen, be provided at the surface, and that portable buckets with watertight linings fitted with hermetically closed covers, and having a receptacle to contain deodorising powder, be provided underground. Where such provisions to combat the disease have been taken, and the miners have been encour-

aged to adopt more cleanly habits, the results toward improvement have been marked.

Effect of Sulphur.—An interesting statement based on recent investigations, and coming from an excellent source, tends to contradict many of the long accepted theories dealing with the subject of coal. It has generally been believed that the amount of sulphur in a coal is an indication of the tendency of the ash to clinker. Now comes the emphatic statement that in some of the Illinois seams, where the sulphur content is highest, the ash produces the least clinkering, and that as a consequence, conclusions regarding the behavior of the ash, when based on the amount of sulphur in the coal, are not justified.

One further problem that has puzzled many mining men is the continued upheaval of the bottom of some of the gangways in the mines of many districts. At one anthracite mine, for several months past, the mine officials have been confronted by the problem of having the floor in one of the haulways continually being forced or squeezed up, the rise often amounting to several inches during a night. It has generally been thought that the upheaval of bottom in such cases is due to the pressure of the overlying strata on the surrounding pillars, or to the action of water; careful investigation has led to the belief that in many instances this bulging or raising of the entry floor is due to the great pressure of the gases contained in the strata underlying the coal measures.

Accidents.—Because of the great number of accidents that are continually occurring in our coal mines, there is a move on foot in many States to enact certain laws that will make it possible to hold the company officials, or a careless individual, responsible for any loss of life that may occur. Agitation of this sort is advisable and will probably result in making both mine officials and miners themselves more thoughtful and careful. Considering coal mining throughout the entire United States, it is found that there are more than five times as many lives lost by falls of roof as from explosions of gas and dust. It is also interesting to point out that about 60 per cent. of all the fatal accidents in or around coal mines are due to negligence, recklessness, or ignorance on the part of the victims. The larger proportion of those injured in these mine disasters are non-English speaking employees; this condition is undoubtedly due to the fact that many miners do not have a sufficient knowledge of the English language to understand orders, and consequently to protect themselves in the performance of their duties.

A few years ago it became the practice in several districts to have a regular and experienced corps of men who went through the mines and did nothing but set props. This system has met with considerable success and may prove worthy of general adoption. If the men performing this work are conscientious and careful, the number of accidents from roof

falls will surely decrease. It seems unnatural that the miners themselves should be careless where their own lives may pay the penalty; nevertheless it is true, and is due to the fact that the miner, being paid by the car or ton, considers all time lost that is occupied in doing anything other than mining coal.

One other practice that has recently become a law in certain States is that of having specially appointed shot-firers, and forbidding any miners to shoot the coal. This system is likely to become a disastrous failure, as the miners before leaving in the afternoon, carelessly charge and tamp their holes, knowing that they will not fire them, or be in the vicinity at the time. Several fatal accidents have resulted in this way.

Labor Conditions.—It seems at present that the labor difficulties, which have been a constant menace to the industry as a whole, are to be largely eliminated hereafter, because of a realization on the part of both capital and labor that all misunderstandings can be better settled, and with less loss to all, by an impartial commission or court. There is no doubt that the life of a coal miner is attended by considerable hardship and much danger; however, recent statistics show that during 1906 hard coal miners received on an average $690.34 each. This rate of pay to coal miners is higher than the annual wages received by the skilled and unskilled employees of nearly all other manufacturing industries. Like all other organizations, the miners' union is composed of men both good and bad. It would not only be unusual, but also impossible for conditions to be otherwise, and it is through the desire of the few unscrupulous to further selfish ends that a large part of the trouble that has attended the industry has occurred.

Co-operative Mining.—A review of the work accomplished during 1906 would be incomplete without a reference to co-operative mining. Two conditions are absolutely necessary in creating efficient workmen: The first is encouragement; the second is enthusiasm. The latter will not be present if the first is not exercised, and encouragement is the cheapest gift in the bestowal of a mine manager. One other factor may be mentioned as a basis for enthusiasm, and that is a thorough and intelligent understanding of the work in hand. A man who can read nothing but English will not be likely to grow enthusiastic over an article in a German paper; neither will a miner who is unfamiliar with the principles of ventilation or the effect of mine gases be sufficiently interested in building his brattices or hanging a curtain. The foregoing statements may be advanced as the fundamental principles on which successful co-operative mining is based. This leads us to a consideration of the plan which is successfully carried on at several different localities in America. One operation of such a character is at Saginaw, Mich. This mine is owned entirely by the workmen who operate it. They establish prices, make contracts, and

go underground to dig out the product. There are no strikes here, for every man is personally interested in the welfare of the operation.

The company was first organized with 100 men, and a capital stock of $50,000. After a year of success, however, it has been decided to increase the capital to $250,000, and the company to 500 miners. Some of the men had no money, and arranged to pay their part in labor.

When trouble was expected in Ohio in 1906 the Caledonia Company, as this co-operative mine is called, was flooded with orders, due to the fact that consumers knew there would be no shut-down at this property.

The men at the Caledonia mine choose their own superintendent, who is responsible to a board of managers elected by the miners. Other men, noting the success of this new venture, have formed companies along these lines, and as every employee is interested in seeing that the product turned out is of the best, the mines have succeeded in establishing a reputation for themselves, as well as surviving the fiercest kind of competition. There have been many instances where co-operative work of this sort has utterly failed, while, on the other hand, the advocates of the system can point to a number of examples where success has attended the efforts of the men thus engaged.

Conclusion.—I remember during the summer of 1902, when the whole country was agitated because of the terrible coal strike that was taking place, causing the price of coal to be raised until its purchase by the common people was almost impossible, a coal operator (whose mines were working because he employed unorganized labor) rubbed his hands and smilingly told me that there was no doubt but that coal was king. So long as things run smoothly the general public does not realize this fact, but let the wheels that govern the production of coal, whether the time be winter or summer, stop for a short period, and see what alarm prevails and suffering ensues.

As to the efficiency of our men and methods, we have only to remember that every miner now employed in our coal mines produces on an average nearly twice as much coal per year as the miner of any other country. We as a people are also consuming more than twice as much coal for making steam than the United Kingdom, our nearest competitor. Coal mining during 1906 did not follow the great wave of prosperity that swept over the country, but rather led the whole procession, and present prospects seem to indicate that it will continue in that exalted position.

By-product Coke Ovens.

By C. G. Atwater.

The last article in The Mineral Industry treating specifically of the by-product coke oven appeared in the issue for 1902, Vol. XI. Since that was written the progress of the by-product coking process, as recorded

in the actual number of ovens built, has been less rapid than in the years preceding. Table I gives the plants now in existence, together with their location, size, and the uses to which the coke and gas are put. Of the ovens enumerated therein, 376 are not completed, and 75 were not operated during 1906 for various reasons. Of the remainder nearly all were operated throughout the year, and in most cases were pushed to their maximum capacity.

TABLE I. BY-PRODUCT COKE OVENS IN THE UNITED STATES AND CANADA IN 1906.

Company.	Location.	No. of ovens.	Use of Coke.	Use of Gas
Otto-Hoffmann and United-Otto ovens.				
Cambria Steel Co.	Johnstown, Pa.	260	Blast furnace	Fuel
Cambria Steel Co. (a)	Johnstown, Pa.	112	Blast furnace	Fuel
Pittsburgh Gas and Coke Co.	Glassport, Pa.	120	Blast furnace and domestic	Illum. and fuel
New England Gas and Coke Co.	Everett, Mass.	400	Domestic and locomotive	Illuminating
Dominion Iron and Steel Co.	Sydney, C. B.	400	Blast furnace	Fuel
Hamilton Otto Coke Co.	Hamilton, Ohio	50	Foundry and domestic	Illuminating
Lackawanna Iron and Steel Co.	Lebanon, Pa.	232	Blast furnace	Fuel
Lackawanna Iron and Steel Co.	Buffalo, N. Y.	188	Blast furnace	Fuel
Lackawanna Iron and Steel Co. (a)	Buffalo, N. Y.	376	Blast furnace	Fuel
South Jersey Gas, Electric & Traction Co.	Camden, N. J.	150	Foundry and domestic	Illum.and power
Maryland Steel Co.	Sparrows Point, Md.	200	Blast furnace	Illuminating
Michigan Alkali Co.	Wyandotte, Mich.	30	Lime kilns	Fuel
Sharon Coke Co.	Sharon, Pa.	212	Blast furnace	Fuel
Zenith Furnace Co.	Duluth, Minn.	50	Blast furnace	Illuminating
Semet-Solvay ovens.				
Solvay Process Co.	Syracuse, N. Y.	40	Lime kilns	Fuel and power
Semet-Solvay Co.	Dunbar, Pa.	110	Blast furnace	Fuel
National Tube Co.	Benwood, W. Va.	120	Blast furnace	Fuel
Carnegie Steel Co. (b) (c)	Sharon, Pa.	25	Blast furnace	Fuel
Semet-Solvay Co.	Ensley, Ala.	240	Blast furnace	Fuel
Peoples Heat and Light Co. (b)	Halifax, N S.	10	Domestic	Illum. and fuel
Solvay Process Co.	Delray, Mich.	120	Lime kilns	Illum. and fuel
Philadelphia Suburban Gas Co.(b)	Chester, Pa.	40	Blast furnace	Illuminating
Empire Coke Co.	Geneva, N. Y.	30	Foundry and domestic	Illuminating
Central Iron and Coal Co. (d)	Tuscaloosa, Ala.	40	Blast furnace	Fuel
Pennsylvania Steel Co.	Lebanon, Pa.	90	Blast furnace	Fuel
Pennsylvania Steel Co.	Steelton, Pa.	120	Blast furnace	Fuel
Milwaukee Coke & Gas Co.	Milwaukee, Wis.	160	Blast furnace and foundry	Illuminating
By-products Coke Corporation	So. Chicago, Ill.	160	Blast furnace, foundry and domestic	Illum. and fuel
Rothberg ovens				
Lackawanna Steel Co.	Buffalo, N. Y.	141	Blast furnace	Fuel
Retort Coke Oven Co.	Cleveland, Ohio	80	Blast furnace	Fuel
		4 306		

(a) Not completed. (b) Not operated in 1906. (c) Omitted from table in Vol. XI, p. 153. (d) 80 ovens as given in previous table, but only 40 actually built.

In comparing this list with the one in the previous article, above referred to, it will be noted that the retort ovens at Cleveland have been replaced by the Rothberg ovens, the number being increased to 80, and also that the Newton-Chambers ovens at Pocahontas, Va., have not been included in the later list. The last mentioned plant has been reported as out of operation for some time past, and the oven itself differs from the true by-product type, operating with a closed retort, inasmuch as it requires the admission of air into the coking chamber to support combustion and

heat the ovens, instead of burning the cleaned gas in separate heating flues surrounding the coking retort. They have, therefore, been omitted from the enumeration.

Allowing for the difference indicated by the foot-notes to the table, an addition of 642 ovens has been made to the number existing in 1903, counting those that will be completed by the date of issue of this article. Considering the previous progress in the introduction of the by-product oven and the great demand for coke that has existed during most of the time in question, the number of additions is not large. It would not be proper nor indeed in accord with the real conditions, however, to assign this lull in by-product oven construction to any lack of faith in its ability to do what its advocates claim for it, nor to doubt as to its ultimate prevalence in the field. The large number of ovens built in Germany and England during this time, and the generally successful performance of those operated in the United States make any such doubt untenable. The real reasons lie rather in this very success, and in the boom conditions that have prevailed in the iron business.

The production of tar and ammonia in the country has been greatly increased by the amount of these products recovered in the by-product ovens, and this increase has gone hand in hand with a considerable increase from the former sources. Thus, the ammonia supply has practically doubled since 1902, and the tar production has increased in nearly the same ratio. Nearly half of the present coal tar production of the United States and two-thirds of the ammonia are due to the by-product oven.

The market for the different forms of ammonia has apparently suffered no depression from this increase, in spite of the apprehension formerly expressed in some quarters, but that for tar has not responded so readily. According to one authority[1] 28 per cent. of the tar produced in 1905 was burned as fuel and the price per U. S. gallon as given in the official reports[2] has declined from 3.5c. in 1902 to 2.7c. in 1905. While there is no doubt that the extension of the present uses for tar, and the development of new ones, other than as fuel, will soon absorb the present surplus and take care of reasonable increase, the effect of this slight uncertainty has not been to encourage immediate new construction.

As to the effect of the boom conditions in the iron trade, the demand for coke has been so great that the profit of the bee-hive operation has been satisfactory, and in many cases where extension was imperative to obtain coke already contracted for or necessary to take advantage of profitable opportunities, the comparatively short time required to build and place in operation a bee-hive plant as compared with that required for a by-product installation has proved the deciding factor. The iron and steel

[1]Bulletin 65, Bureau of the Census, p. 19.
[2]"Gas, Coke, Tar and Ammonia," United States Geological Survey, 1902 and 1905.

business offers notoriously large returns, even for more or less antiquated plants, when the market is on the boom. It often requires the pinch of falling prices to bring out clearly the need of operating economies. Further, the investment in a by-product plant is heavy, and to gain the maximum returns from such a plant requires entrance into manufacturing lines hitherto foreign to the iron business. The recovery of the surplus gas in form suitable for illuminating purposes, which is essentially an American innovation, has added a new and very profitable element to the process, but the necessary connection with large centers of population to afford a proper market for this gas is not always attainable. Its German substitutes, the recovery of the benzol from the gas and the use of the de-benzolized gas for power generation in the gas engine, are not as yet generally practiced in this country. Benzol is recovered at several plants, but the use of the high-heat-unit coke-oven gas in gas engines has lagged far behind German practice.

The introduction of by-product coking plants operating independently of blast furnaces, relying on the foundry coke and domestic fuel demand for their coke disposal, and supplying illuminating gas to cities, has been successful, and an almost unlimited field is opening to the by-product oven process in this direction. The pioneer plant at Everett, Mass., has succeeded in developing a market for its coke as an industrial, domestic and locomotive fuel, and its gas has been continuously disposed of to advantage. The later plants in the same industrial field, using a coal from which foundry coke as well can be made, have created a market for their product at a remunerative price and have also found a ready sale for their gas. The transportation of the gas through mains under pressure to considerable distances has also been resorted to in several instances, and where gas of suitable composition has been used has met with unqualified success.

Both in the plants producing furnace coke and those making the foundry and domestic article, it has frequently been found advantageous to mix two or more coals, either to better the quality of the coke or that of the gas, or in order to use a coal of lower cost. This is a common practice abroad. The proportions in the mixture, and the methods of making it, vary considerably in different plants. Much attention has been given in the last year or two to this matter, and special apparatus for this purpose has been developed.

The process of compressing the raw coal into a cake before charging into the oven has been practically abandoned, in this country at least. At the Dominion Iron and Steel Company's works the compressors installed were operated for a time, the result being entirely satisfactory as regards the quality of the coke produced, but not so fortunate from the standpoint of cost. The apparatus was subsequently dismantled. Simi-

larly the apparatus installed by the Lackawanna Steel Company at the Lebanon and Buffalo plants was operated for a while, but this method of charging has now been definitely given up, and the generally accepted plan of charging loose coal through openings in the oven top has been adopted. The difficulty in this case seems also to have been in the cost of operation, the labor expense and the loss from spillage due to broken cakes being very heavy, especially in winter. The beneficial effects of compression do not appear to be important when coals low in volatile matter are used, as these naturally make a coke of compact cell structure and do not incline to sponginess. With those coals of moderate or high volatile contents, which are apt to produce spongy coke, particularly in the upper portion of the oven charge, compression has proved decidedly beneficial, diminishing or entirely eliminating the spongy portion, and decreasing the percentage of breeze. The defect of brittleness, which is apt to accompany an inclination to sponginess, is not particularly helped by compression, as it depends on the inherent qualities of the fused coal forming the cell walls. The mechanical operation of compressing the coal may aid by altering the size of the cells, but it cannot be expected to affect the strength of the cell-wall material, which will still retain its tendency towards brittleness. Under present conditions the use of such coals does not seem to offer sufficient economy to compensate for the additional operating charges. The last two plants above mentioned are the only ones at which compression has been regularly resorted to in the United States though the process is still in use in Germany and elsewhere with certain high volatile coals.

Coke Production.—The amount of coke produced from by-product ovens in the United States for each year since their introduction is shown in table II, together with the figures for the total coke production.

TABLE II. COKE PRODUCTION OF THE UNITED STATES.
(Tons of 2000 lb.)

Year.	By-product	Total.	Per Cent. of Total	Year	By-product	Total.	Per Cent. of Total.
1893	12,850	9,447,580	0.13	1900	1,075,727	20,533,348	5.35
1894	16,500	9,203,632	0.18	1901	1,179,900	21,795,883	5.4
1895	18,521	13,333,714	0.14	1902	1,403,488	25,401,730	5.5
1896	83,038	11,788,733	0.7	1903	1,882,394	25,274,281	7.4
1897	261,912	13,288,984	2.0	1904	2,608,229	23,661,106	11.1
1898	294,445	16,047,209	1.8	1905	3,462,348	32,231,129	10.7
1899	906,534	19,668,569	4.6	1906	4,482,179	(a)32,700,000	15.9

(a) Estimated.

These figures show that while the total coke production has increased greatly, generally speaking, there have been years when the output was

[1]These figures and the others in this section are based on the data given in the "Mineral Resources of the United States," except those for the year 1906 in Table II, which are made up from reports by the plants in operation. In the case of the by-product coke, all of the operating companies have made returns.

less than in the preceding year. In the by-product oven production the increase has been continuous, though at times it has been small. The periods of small progress have been followed by large increases in a single year, the additions coming as it were in waves. As the total number of ovens in operation increases, the effect of these increments on the total is proportionally less.

The proportion of the total coke produced in the by-product oven, starting with a small fraction of 1 per cent., has increased until it exceeded 11 per cent. in 1904. This advance has not been made without various halts, and at times slight retrogressions; the latter are due, however, to variations in the total output of coke rather than in that from the by-product oven itself. Such, for instance, is the case in 1905, in which the proportional figure, 10.7 per cent., falls below that of 1904, not because there was not a normal increase in the by-product coke output, amounting indeed to 33 per cent., but because the total coke output increased at the same time by 36 per cent.

The figures for 1906 show that the by-product coke production has increased by 29.4 per cent. over that of the preceding year, and that it now forms about 13.7 per cent. of the total coke output. The gain in both directions is therefore a substantial one.

If we omit from the total number of by-product ovens given in Table I those in Canada and those not completed or not operated in 1906, we have a total of 3343 by-product ovens that contributed to the coke production in that year. Using this figure and that for the production in 1906, Table II, we find that the output of coke per oven for the year was 1340.8 net tons. The official figures for previous years show that the production was 1158.8 short tons in 1905, 896 in 1904 and 962.4 in 1903. The increase in these figures is due to the larger capacity of the newer ovens constructed and to the increasing efficiency with which the plants are operated. The corresponding figure for the bee-hive ovens in 1905 was 365.8 tons per oven per year, or in other words, a by-product oven produces over three times as much coke as a bee-hive oven. The later type of by-product oven, coking 8 to 11 tons of coal in 24 hours, may be counted on for 2100 to 2600 tons of coke a year, or four to five times as much as the maximum capacity of a bee-hive oven.

The actual amount of coal coked in by-product ovens in 1905 was 4,628,891 tons and the coke yield was 74.8 per cent. In 1904 the coal coked was 3,572,949 tons, and the yield in coke was 73.1 per cent., while for 1903 the figures were 2,605,453 tons and 72.25 per cent. respectively. Previous to this the authority from which these figures are taken makes no separation of the coal coked, except for 1898, in which year the coal coked in by-product ovens was 402,297 net tons and the coke yield 73.2 per cent. These coke yields are all much higher than the figures for bee-

hive practice. The yield from both types of oven is given as an average of 64 per cent., but there is reason to believe that the coal weights on which these yields are based, as far as the bee-hive output is concerned, are largely estimates, and are taken too low, so that it is doubtful if the yield of coke averages over 60 per cent. for the whole industry. This does not apply to the by-product ovens, as the coal and coke are generally weighed at these plants. The superior yield of the by-product oven is due to the exclusion of the air from the coking chamber, so that none of the fixed carbon in the original coal is consumed while coking, the loss being practically confined to the volatile hydrocarbons and moisture driven off. This is in sharp contrast to the results of the bee-hive oven, particularly on low volatile coals. With such coals the volatile matter is insufficient to coke them properly under the bee-hive conditions, which necessitates the combustion of a portion of the fixed carbon as well, and reduces the yield to 50 or 60 per cent. Some of these coals yield as high as 81-82 per cent. of coke in the by-product oven.

Gas Production.—The production of illuminating gas from by-product coke ovens has been made successful through the resort to fractional distillation, or separation of gases. This principle was first tried on a large scale in this country at the Everett plant. Without going into the details of the method, which have been elsewhere described,[1] it may be stated that it consists in separating the gas into two portions, viz.: one containing the larger proportion of the light giving hydrocarbons, for illuminating purposes; and the other comprising the residue of the gas, for use in heating the ovens. The separation is accomplished at the ovens, by the use of separate gas collecting mains and the gas is treated in separate condensing systems. By this method it has proved possible to obtain gas of excellent illuminating value, entirely fitted for city distribution purposes, without resorting to the use of enrichers, at the same time making a metallurgical coke from coal comparatively low in volatile matter that would not be considered fitted for use in the coal gas retort.

Figure 1 is a diagram showing the candle power, calorific value and volume of the gas given off during each hour of the coking period from such a coal. The average illuminating value of the surplus gas, as shown by the upper line on the diagram, was 18 candle power, a figure which is fully up to the usual practice in coal gas works. The point at which the division of the gases took place is indicated by the vertical line. The portion of the candle power line to the right of this indicates the candle power of the oven-heating or fuel gas. The middle line of the diagram shows the calorific value per cubic foot of the gas generated for each

[1]See article on "The Manufacture of Coke in the United States with Special Reference to the Market for By-Products," by Dr. F. Schniewind, *The Mineral Industry*, Vol. X, p. 135.

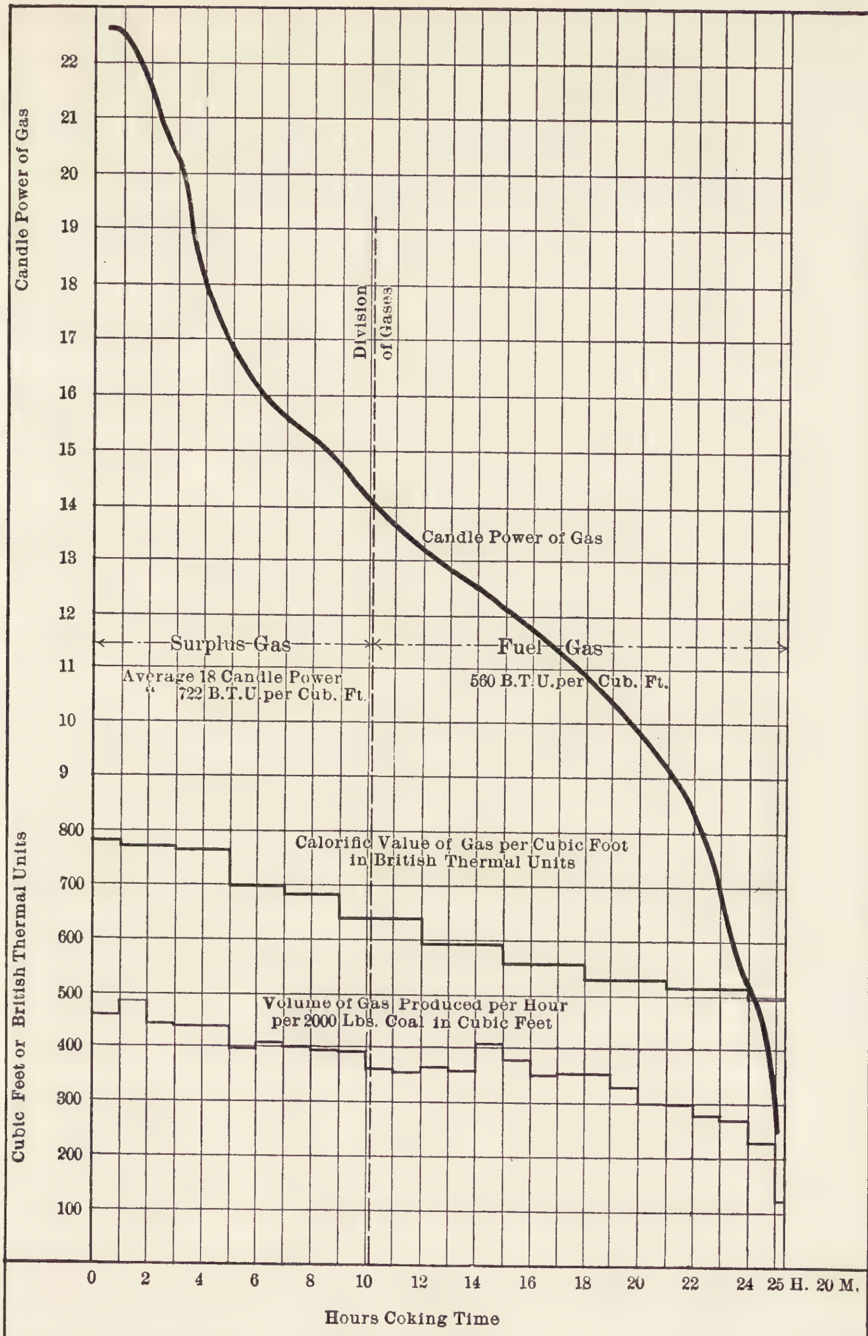


FIG. 1.—DIAGRAM SHOWING CANDLE POWER, CALORIFIC VALUE AND VOLUME OF COKE OVEN GAS.

hour, and the lower line shows the volume of the gas generated each hour from 2000 lb. of coal. The surplus gas in this case amounted to 4300 cu.ft. or 46 per cent. of the total gas evolved, while calorifically considered it contained 52 per cent. of the total heat value of the gas.

From a net ton of coal averaging 35 per cent. volatile matter over 5000 cu.ft. of 19 candle-power gas has been produced in by-product ovens of the United-Otto type, in regular operation, without enrichment. This gas was, of course, in excess of that required for heating the ovens. This is no mean achievement from the standpoint of heat economy. The fuel gas, together with other small losses, represents the only part of the original heat in the coal not recovered in some form in the operation, and comprises less than 10 per cent. of the total heat value. The loss in converting the raw coal to coke and by-products is therefore but 10 to 11 per cent., a loss at least as low as that of any other method of gasifying coal. This high efficiency is due in part to the use of the regenerative system of waste heat recovery.

The gas produced from by-product ovens has already become a considerable factor in the illuminating gas industry. This is shown by Table III, compiled from data given in *Mineral Resources*.

TABLE III. COAL GAS PRODUCED AND SOLD FROM COAL GAS WORKS AND BY-PRODUCT COKE OVENS IN THE UNITED STATES.

	1904	1905
Gas works, cubic feet	30,109,449,125	30,722,278,832
By-product coke ovens, cubic feet	4,705,542,148	9,731,936,300
Total cubic feet	34,814,991,273	40,454,215,132
Percentage from by-product ovens	13.5	24

It is apparent from the figures in that table that the proportion of oven gas is increasing rapidly and that the amount of gas produced and sold from by-product ovens is increasing more rapidly than that from the coal gas retort. The increase in the latter as reported is but two per cent., while in by-product oven production the increase was 107 per cent., the gas from the latter source forming nearly one-fourth of the total coal gas made and sold. If to the 40,454,215,132 cu.ft. of coal gas we add the 77,412,024,591 cu.ft. of oil and water gas made and sold in 1905, as given by the same authority, we have a total of 117,866,239,723 cu.ft. of gas made and sold for the year. Of this amount the oven gas formed nearly 8.3 per cent. As considerable oven capacity making illuminating gas has come into operation since these figures were compiled, it may be expected that the ensuing years will show an increase in this percentage. The higher cost of the oil used in the manufacture of carburetted water gas, and the increased favor with which gas manufacturers regard coal

gas, because of its lower cost, both tend to hasten this movement. On the other hand the conservative attitude adopted, perhaps naturally, by the vested gas interests toward so radical an innovation, tends toward delay, and this is increased by the lethargy of the people at large, as regards the use of coke for the suppression of the smoke nuisance. This is particularly true in the larger cities of the Middle West, where in many cases the disposal of coke for other than blast furnace use would have to be considered, if a plant were built. The success of the plants already built has gone a long way towards overcoming the obstacles in the way of future ones.

Coal Tar Production.—The amount of coal tar produced in by-product coke ovens in 1905 was 36,379,854 U. S. gallons and in 1904 it was 27,771,115 gal.[1] Previous to this the official figures are not given separately. The total coal tar produced in gas works and by-product ovens for the years since 1902 is as follows: 53,099,508 gal. in 1902; 62,964,393 in 1903; 69,498,085 in 1904; and 80,022,043 in 1905. The production of the by-product ovens, therefore, was 45 per cent. of the total in 1905, and 40 per cent. in 1904. In 1898, the only previous year in which an attempt was made to collect such figures by the Geological Survey, the coal tar production of the gas works was 24,384,798 gal. and that from by-product ovens was estimated as 10 gal. of tar per 2000 lb. of coal coked, or 4,022,970 gal. The total coal tar produced was therefore about 28,407,768 gal., of which the by-product oven production was only 14 per cent. These figures show that the tar output of the United States has been nearly trebled in seven years, and that the major part of the increase is due to the by-product coke oven. In addition to this, considerable water-gas tar is produced and consumed in the country.

The uses to which coal tar is put are the same as have hitherto prevailed in this country, namely, the saturation of felt and distillation into soft pitch for use in water-proofing, roofing, etc., the pitch being also extensively used in paving. Tar is also used for coating pipes and ironwork and in the preparation of paints and varnishes. A more recent use is its application to ordinary macadam roads as a surface dressing, to allay dust and increase durability, for which purposes it has had considerable success. A certain amount has been burned as a fuel, both as tar and in the form of pitch in fuel briquets. In the manufacture of roofing and waterproofing products the industry is well organized and equipped, and its products compare favorably with those of any other nation, but the manufacture of coal tar chemicals and colors has not as yet been extensively undertaken.

Ammonia.—The production of ammonia in its different forms from by-product coke ovens and other sources has been treated in a separate article in an earlier part of this volume, to which reference may be made.

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[1] "Production of Gas, Coke, Tar and Ammonia" for 1905, by E. W. Parker.

United-Otto Ovens.—This designation applies to the ovens built by the United Coke and Gas Company, and includes the ovens of the earlier Otto-Hoffmann type as first built in this country. The distinguishing characteristics of this oven are the vertical position of the heating flues in the oven wall and the recovery of the waste heat by the use of regenerative chambers. To these have been added an improved system of heat distribution in the oven, which makes it possible to heat properly ovens of larger size, and methods of oven construction designed to permit the production of illuminating gas. Ovens of the United-Otto type are now built to take a coal charge of 9.5 tons (of 2000 lb.), coking in 20 hours, whereas the earlier ovens coked a charge of five to six tons in about 30 hours. During the year 1906 the 50-oven addition to the Camden plant and the 15-oven addition to the Wyandotte plant were completed and

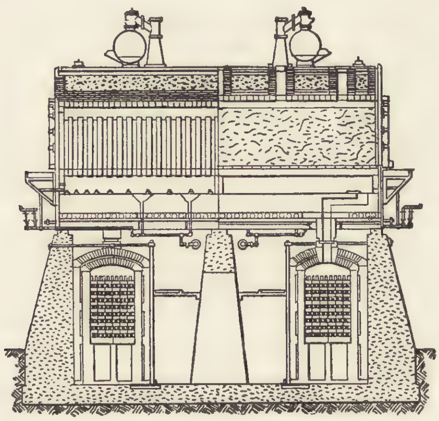


FIG. 2.—LONGITUDINAL SECTION OF LATEST TYPE OF UNITED-OTTO BY-PRODUCT OVEN.

placed in operation. There were, therefore, 1892 ovens of the United-Otto types in operation in 1906. Of the 112-oven addition to the Johnstown plant one battery was completed and placed in operation early in 1907, and the other is expected to follow in a short time.

A longitudinal section of the latest type of United-Otto oven is shown in Fig. 2, the left-hand portion being taken through the oven wall, and the right-hand through the oven. On the left is seen the arrangement of heating burners, there being one at the end of the oven and two entering from beneath. The right side shows the location of the charging holes and the downtake communicating with the regenerative chamber below. The sub-structure of the oven consists of three piers of masonry or concrete, upon which are supported steel I-beams carrying the oven masonry proper. The regenerative chambers are placed next to the outer supporting walls, and are so designed that expansion and contraction occuring in them can

have no effect on the oven structure and walls. There is also the additional advantage that every portion of the oven structure is easily accessible for inspection and incidental repairs if need should arise.

The general arrangement of a plant of 100 United-Otto ovens arranged for extension to 200 ovens is shown in Figs. 3 and 4 in cross-section and plan. Fig. 3 shows the coal-receiving tracks at the left, the coal being dumped from standard hopper-bottom cars to the crusher and carried by the elevator to the oven coal bin. Thence it is drawn by gravity through the chutes to the coal larry, which runs on tracks along the oven batteries, and discharges the coal through chutes corresponding with openings in the oven top. The uneven piles of coal in the oven are then levelled by an electrically operated levelling bar, the oven is sealed up and the coking operation is proceeded with. The coke charge is then pushed out by the

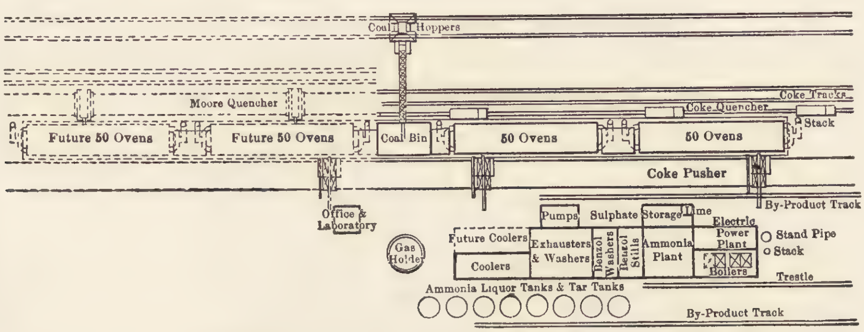


FIG. 3.—GENERAL ARRANGEMENT OF 100 UNITED-OTTO OVEN PLANT WITH PROVISION FOR EXTENSION TO 200 OVENS. GROUND PLAN.

pusher on the right of the oven, and the coke is received and quenched in the car on the left of the oven. This car may be either of the inclined type, in which the coke falls from the oven and is distributed in a shallow layer on the inclined platform, being there quenched by water from a hose, or it may be received by a later type of apparatus known as the Moore quencher. This is shown in dotted lines in Fig. 3. It consists essentially of a chamber of about the same internal dimensions as the oven, formed of cast-iron sections and with cellular walls that permit of water cooling. The bottom is formed by a chain conveyor which is operated by an electric motor, and the ends are closed by doors, the end next the oven being also provided with extension sides to connect with the oven. Means are provided for deluging the coke inside the quenching chamber with water, and there are stacks through which the steam generated may escape. The whole is mounted upon a carriage driven by electric motors and is movable on rails parallel with the face of the oven batteries, so that each oven in turn may be served. The coke charge is pushed intact

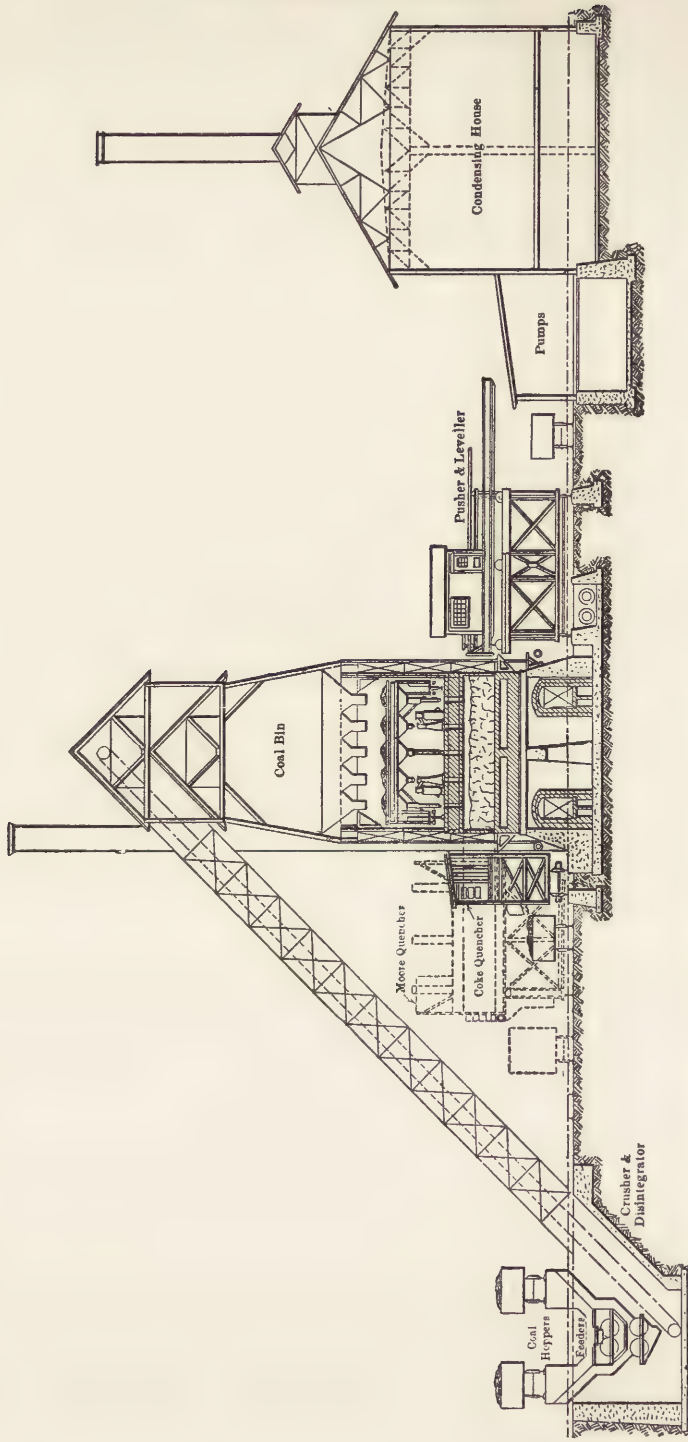


FIG. 4.—GENERAL ARRANGEMENT OF 100 UNITED-OTTO OVEN PLANT WITH PROVISION FOR EXTENSION TO 200 OVENS. CROSS-SECTION.

from an oven directly into the quencher chamber, aided by the moving conveyor on the bottom, and when the charge is quenched, the door on the outer end is opened and the coke is ejected by the conveyor, falling into a car placed on the track alongside. The advantages of this method

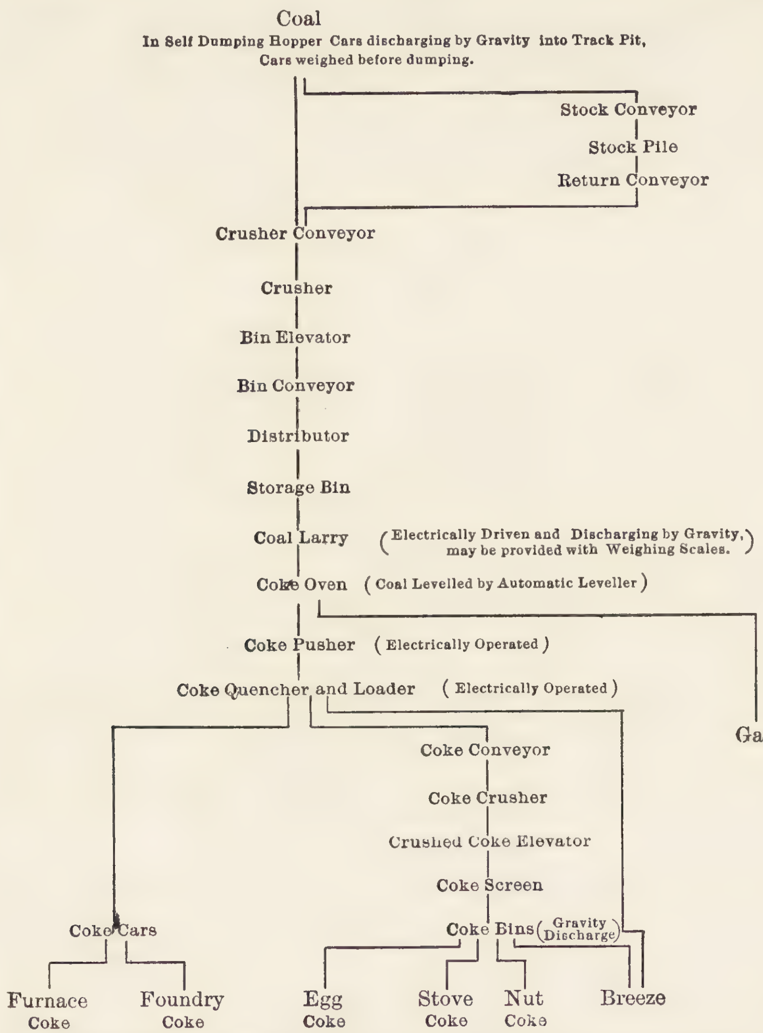


FIG. 5.—DIAGRAM SHOWING MECHANICAL HANDLING OF COAL AND COKE IN BY-PRODUCT OVENS

of quenching are that the coke charge is kept on the oven floor level and thus can be discharged into steel cars of the usual hight without having to depress the coke tracks below the yard level; and that the coke is dropped only once, with a minimum of breakage and the least possible

amount of operating labor. The color of the coke is also brighter, as the quenching is done practically in an atmosphere of steam, and the method of watering allows the residual moisture in the coke to be kept down to the lowest figures. Both of these points are regarded as important by blast furnace men.

The sequence of operations in the treatment of coal and coke and the extent to which mechanical means are resorted to to the exclusion of hand labor is shown by Fig. 5. The general arrangement of the condensing and by-product plant is shown in section in Fig. 3, but to better advantage in the ground plan, Fig. 4. The gas is collected in mains running parallel with the oven batteries, and enters the condensing house at the end nearest the coal bin, passing in succession through the various stages of

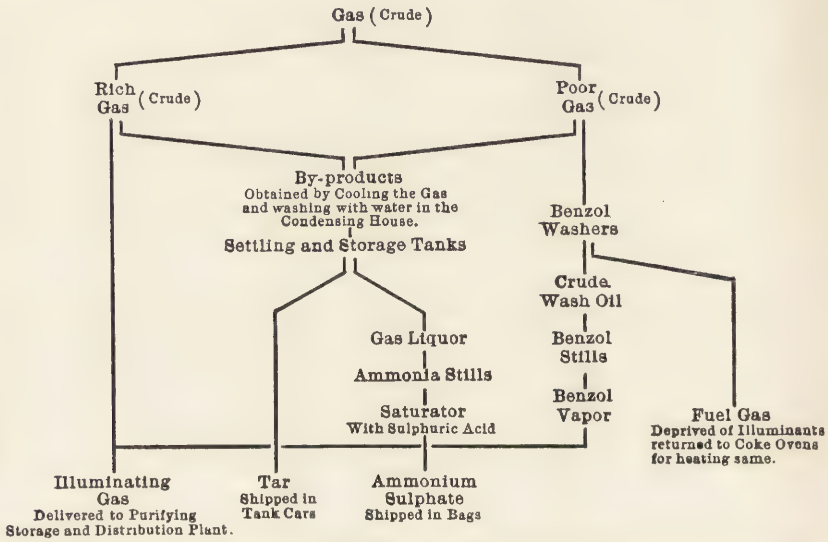


FIG. 6.—DIAGRAM SHOWING TREATMENT OF GAS AND BY-PRODUCTS WITH GAS SEPARATION AND BENZOL TRANSFER.

the treatment. The sequence of the operations and the disposal of the products recovered is shown in Fig. 6. This diagram also indicates the method resorted to in order to recover the illuminating value from the fuel gas, as shown in Fig. 1, and transfer it to the illuminating gas fraction, this being known as the method of benzol transfer. It is of particular importance with low volatile coals, which produce gas of low candle power. This arrangement has resulted in an enrichment of over four candle power for a daily output of 2,500,000 to 3,000,000 cu. ft., the enrichment being permanent even when the gas was pumped 38 miles, under 10 lb. pressure, in winter weather. This method transfers the benzol in the form of a vapor directly from the absorbing oil to the gas

to be enriched, without the necessity of an intervening condensation and rectification of the benzol. The elimination of these processes simplifies the operation and reduces its cost. A detailed description of this and of benzol enrichment in general is given in a paper entitled "Benzol Enrichment with Reference to Coke Oven Gas," by Dr. F. Schniewind in *Progressive Age*, Nos. 10, 11 and 12, 1905.

Semet-Solvay Ovens.—The distinguishing characteristics of these ovens are the use of a separate system of horizontal heating flues for each oven, through which the heating gases pass continuously in the same direction, the separate division walls between the ovens which entirely sustain the oven superstructure, and the use of the steam boiler to save waste heat from the out-going gases.

The original type of construction of this oven has been closely adhered to, except as to the dimensions, which have been largely increased. In 1893 the standard oven was 30 ft. long, 5.5 ft. high and contained 4.4 short tons of coal at a charge. This oven was built with three horizontal flues on each side, one above the other, and 25 ovens constituted a block, the capacity of which was 110 tons of coal. In 1903 the oven was built with four horizontal flues on each side, and of somewhat increased length, taking from seven to nine tons of coal at a charge. The standard battery was then 40 ovens, with a total maximum charging capacity of 360 net tons. In 1905 the length of the standard oven was increased to 35 ft., its hight to 9 ft., there being five horizontal flues on a side and the capacity was nine tons of coal. The standard block was 80 ovens, with a total charging capacity of 720 tons of coal. These ovens are known as the three-, four-, or five-high ovens respectively. In spite of these changes in length and hight, however, the width of the oven has been maintained at an average of 16.5 in. Greater widths (up to 20 in.) have been tried, but did not give as large an output of coke per day as the standard dimension. The coking time, which was 26 hours in 1903, has been reduced to 24, and more recently to 18 hours. This reduction may be partly attributed to the use of higher heats and partly to the use of machinery in charging and discharging the ovens. The pushing, charging and sealing up of an oven is now done in 15 minutes.[1]

During 1906 the plants at Chicago, Tuscaloosa and Steelton, together with an addition of 80 ovens at the Milwaukee plant, were completed and placed in operation. There was, therefore, 1230 ovens of this type in operation during the year.

The longitudinal and cross sections of the five-high Semet-Solvay oven are shown in Fig. 7. As is there shown, each of the five horizontal flues communicates at one end with the flue next below, the lower flues on each side of the oven discharging into a common sole flue beneath the

[1]Bulletin No. 65. Census of Manufacturers, 1905, Coke, p. 30.

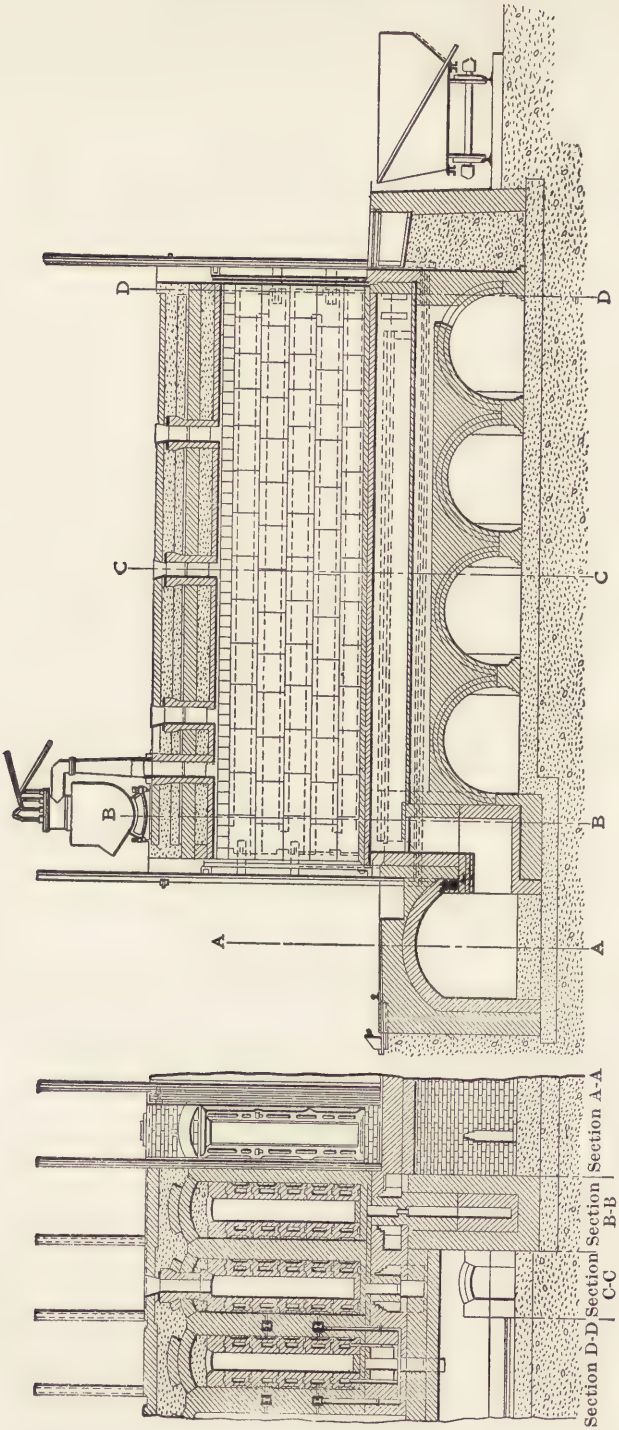


FIG. 7.—IMPROVED SEMET-SOLVAY OVEN.

oven floor which in turn leads through a downtake to the stack flue common to all the ovens of a battery. The course of the heating gases is therefore through the whole length of each flue in succession, beginning at the top. The four upper flues on each side are provided with gas and air inlets by which they are heated, the hot gases from them serving to heat the lower flues and the sole flue. The air for combustion is heated by passing through flues which lie beneath and alongside of the sole flue, and absorb heat by transmission through its walls from the outgoing gases. The air is then led upward through suitable flues to the point of admission to the burners. The temperature of this air is stated to be from 200 to 500 deg. C. The stack flue leads the waste gases to the waste heat boilers, which absorb sufficient heat from them to supply a considerable quantity of steam, the gases passing thence to a stack, which furnishes draft for the system.

The construction of the horizontal heating flues characteristic of this oven is such that they are entirely independent of the oven masonry above them, and carry only the weight of the oven lining itself. The division walls which separate each flue system from the adjacent one are relied upon to sustain the load of the superstructure, and are also said to serve a useful purpose as reservoirs of heat, thus preventing any chilling of the oven while emptying or charging.

In the first Semet-Solvay ovens built in this country the horizontal flues were constructed of monolithic fire-clay tile of large dimensions, each tile fitting into the next one with a bell and spigot joint, and each course of tile forming a complete horizontal flue. As the walls of the tile were not called upon to sustain the weight of the ovens, their thickness could be made less than would have otherwise been necessary, thus facilitating the transmission of the heat to the coal charge. Later small brick having a special form of joint have been successfully used to build up the horizontal flues.

The sequence of operations in the Semet-Solvay ovens is not essentially different from that outlined in Fig. 5. The coal larries and coke pusher are operated by electrical motors instead of hand and steam power as formerly. The quenching of the coke is usually done in an inclined car, the end view of which is shown in Fig. 7, the length being sufficient to allow it to take a full oven charge distributed in a thin layer. Specially arranged water-pipes having large orifices, and placed above the oven door, discharge water on the coke as it comes from the oven. Coke quenched in this way can, with proper attention, be made with as low a percentage of water as the best bee-hive article.

In by-product recovery considerable progress has been made, principally in the direction of greater efficiency and increased yields. The distillation apparatus for treating ammonia liquor has been much improved in its

efficiency, while at the same time it produces concentrated liquor up to 25 per cent. NH_3, thus making a saving in transportation charges over less highly concentrated liquor. The manufacture of aqua ammonia has also received considerable attention, and an article of high commercial quality is now made at several plants. Considerable progress has also been made in the recovery of lighter tar oils and in their use for making and enriching illuminating gas.

The Solvay Process Company's plant at Delray has been delivering gas from by-product coke ovens, enriched with benzol to 18 candle power or over since 1902, to the Detroit City Gas Company, and illuminating gas has since been regularly supplied for city distribution from the plants at Geneva, Chester, Milwaukee and elsewhere. In a paper read by F. B. Wheeler before the Twenty-Eighth Annual Meeting of the Western Gas Association entitled "The Convenience and Advantages of the Practical Enrichment of Low Candle-Power Coal-Gas with Benzol" the use of benzol is treated at length. The results given as to benzol consumption for three plants are summarized in table IV.

TABLE IV. BENZOL CONSUMPTION FOR GAS ENRICHMENT.

	Candle Power of Gas.		Cubic feet of gas enriched 1 c.p. by 1 gal. benzol.
	before enriching	after enriching	
First plant,			
average for 30 days	13.48	18.55	24,491
Second plant,			
average for 6 days	13.86	20.49	20,000
average for 8 days	14.22	20.34	21,739
average for 4 days (a)	6.94	18.42	22,000
Third plant	8.15	17.37	24,039

(a) The gas in this case is stated to have contained as high as 2 per cent. oxygen.

In the 6-day and 8-day tests above mentioned the gas was transported two miles through mains under a pressure of 2 lb., and at temperatures at times as low as 37 deg. F, with an average loss in illuminating power of 0.28 and 0.24 candles respectively. The small extent of this loss under these trying conditions emphasizes the stable character of benzol enrichment, as applied to coal gas.

COPPER.

By W. R. Ingalls.

The production of copper by domestic mines in the United States in 1906 was 917,620,000 ℔. against 875,241,741 ℔. in 1905. The details are given in the following tables. These statistics are based on reports received directly from the producers of Lake and blister copper. The distribution by States is based on reports from the smelters as to the origin of the material treated by them, together with reports from some of the important mines that do not smelt their own ore. However, it is impossible to allocate the production by States with absolute precision, and the distribution given in these statistics is to be regarded only as a close approximation, although in many cases it is precise.[1]

COPPER STATISTICS IN THE UNITED STATES.

(In pounds.)

	1901.	1902.	1903.	1904.	1905.	1906.
Alaska	(a)	(a)	(a)	2,043,586	4,703,609	8,700,000
Arizona	126,183,744	119,841,285	153,591,417	191,602,958	222,866,020	263,200,000
California	33,667,456	25,038,724	19,113,861	29,974,154	16,697,486	24,421,000
Colorado	7,872,529	8,463,938	7,809,920	9,401,913	9,854,174	9,565,000
Idaho	480,511	(a)	(a)	5,422,007	6,500,005	9,493,000
Michigan	155,511,513	170,194,996	192,299,485	208,329,248	218,999,759	224,071,000
Montana	229,870,415	266,500,000	272,555,854	298,314,804	319,179,880	299,850,000
New Mexico	9,629,884	(a)	(a)	5,368,666	5,638,843	6,262,000
Utah	20,116,979	23,939,901	38,302,602	47,062,889	51,950,782	49,712,000
Wyoming	2,698,712	(a)	(a)	3,565,629	2,393,201	146,000
Southern States	6,860,039	13,599,047	13,855,612	15,211,086	14,907,982	18,821,000
Other States	4,551,430	9,218,490	10,846,477	1,418,065	1,550,000	3,379,000
Total production	597,443,212	636,796,381	708,375,228	817,715,005	875,241,741	917,620,000
Stock, January 1	93,050,230	209,587,698	162,935,439	230,111,792	208,376,672	132,587,496
Imports	176,472,369	161,551,040	167,161,720	182,292,205	210,724,685	225,843,281
Total supply	866,965,811	1,007,935,119	1,038,472,387	1,230,119,002	1,294,343,098	1,276,050,777
Deduct exports	227,194,184	376,298,726	312,822,627	555,638,552	548,772,403	467,839,041
Deduct consumption	440,913,929	468,700,954	495,537,968	466,103,778	612,983,199	668,576,336
Stock, December 31	197,857,698	162,935,439	230,111,792	208,376,672	132,587,496	139,635,400

(a) Included in "Other States."

The actual production of copper in the United States is much greater than the production of the domestic mines, a large quantity of ore, matte, and pig copper being imported for refining in this country. The total production of American refiners in 1906 was about 1,136,000,000 ℔. of which 224,071,000 ℔. were marketed as Lake copper (a comparatively small portion of which was electrolytically refined), about 865,000,000 ℔. as

[1] For a more detailed account of the method of compiling these statistics see *Eng. and Min. Journ.*, Apr. 20, 1907, p. 772.

electrolytic copper, and the remainder either as casting or as pig copper. The production of refined copper in the United States according to class is given in the accompanying table:

PRODUCTION OF COPPER ACCORDING TO CLASS.
(In pounds.)

Year.	Total Domestic.	Total Foreign.	Grand Total.	Lake.	Electrolytic.
1897	501,370,295	26,938,254	528,308,549	145,839,749	250,000,000
1898	535,900,232	36,055,352	571,955,584	156,669,098	314,107,776
1899	581,319,091	40,659,868	621,978,959	155,845,786	386,410,356
1900	600,832,505	62,484,290	663,316,795	144,227,340	466,092,663
1901	597,443,212	....	....	155,570,465	(a)485,016,400
1902	636,796,381	....	....	170,194,996	(a)606,270,500
1903	708,375,228	....	....	192,299,485	(a)617,293,600
1904	817,715,005	....	....	208,329,248	705,478,400
1905	875,241,741	183,252,259	1,058,494,000	219,000,000	(b)760,000,000
1906	917,620,000	247,549,000	1,165,169,000	224,071,000	(b)865,000,000

(a) As estimated by the Metallgesellschaft, Frankfurt am Main. (b) Partly estimated.

A somewhat more minute classification is given in the following table:

Year.	Prime Lake.	Arsenical Lake.	Electrolytic.	Casting.	Pig Copper. (a)	Total.
1904	171,284,430	37,044,818	(b)710,000,000	(b)55,000,000	(b)44,408,000	1,017,737,248
1905	175,457,000	43,542,000	(c)760,000,000	46,000,000	33,495,000	(c)1,058,494,000
1906	180,273,426	43,797,574	(c)865,000,000	47,000,000	29,098,000	(c)1,165,169,000

(a) Exported. (b) As estimated by Aron Hirsch & Sohn, Halberstadt, Germany. (c) Partly estimated.

The above tables do not take into account under the caption "electrolytic" the comparatively small quantity of Lake copper which is refined in that way by the Quincy and Calumet & Hecla companies. They refine thus only selected material, especially high in silver, the separation of which more than offsets the slight deterioration in grade which is suffered by the conversion of "Lake" into "electrolytic."

Refining Capacity.—There are in the United States nine important electrolytic refineries (exclusive of the two which refine Lake copper) of which the largest are situated in the vicinity of New York. These are the works of the Nichols Copper Company, at Laurel Hill, Long Island, the United States Metals Refining Company's works, at Chrome (formerly Carteret), N. J., the Raritan Works at Perth Amboy, N. J., the works of the American Smelting and Refining Company, at Perth Amboy, and the Balbach works, at Newark, N. J. Outside of this district there are works at Baltimore, Md., Great Falls, Mont., Tacoma, Wash., and Oakland, Cal. The works at Baltimore, Md., and Tacoma, Wash., are now controlled by the American Smelting and Refining Company. The aggregate capacity of these works, at the end of 1905, was between

750,000,000 and 775,000,000 lb. of refined copper per annum. In 1906 their capacity was increased to about 1,120,000,000 lb.

ELECTROLYTIC COPPER REFINERIES OF THE UNITED STATES.
(Approximate annual capacity at end of 1906.)

Works.	Location.	Capacity, Pounds.
Nichols Copper Company..........	Laurel Hill, N. Y...	288,000,000
Raritan Works....................	Perth Amboy, N. J.	288,000,000
American Smg. and Ref. Company.	Perth Amboy, N. J.	144,000,000
U. S. Metals Refining Company...	Chrome, N. J.......	144,000,000
Baltimore........................	Baltimore, Md......	130,000,000
Balbach Smelting and Refining Co.	Newark, N. J.......	75,000,000
Boston & Montana Copper Co.....	Great Falls, Mont..	30,000,000
Tacoma Smelting Co.............	Tacoma, Wash.....	18,000,000
Mountain Copper Co..............	Oakland, Cal.......	3,000,000

Prices.—The average price of Lake copper at New York in 1905 was 19.616c.; of electrolytic 19.278c. The monthly fluctuations, and comparisons with previous years, are given in the tables presented further on in this article. The increase in 1906 in the difference between the prices of the two grades is explained by the fact that during the latter part of the year the producers of electrolytic sold a large part of their output three and four months ahead, while the nearer deliveries commanded more or less premium, of which the Lake companies, not generally selling so far ahead, were able to take advantage. This condition is reflected in the reports of the Lake companies, the actual receipts of which are shown in the following table:

RECEIPTS OF LAKE COPPER COMPANIES.

Company.	1905.			1906.		
	Pounds.	Value.	Per lb.	Pounds.	Value.	Per lb.
Adventure..................	1,606,208	$252,572	15.73c.	1,552,628	$292,823	18.860c.
Allouez......................	(a)	....	....	3,486,900	667,567	19.145
Atlantic......................	4,049,731	642,306	15.86	(a)	....	....
Baltic........................	14,384,684	2,266,416	15.62	14,397,557	2,742,403	19.05
Centennial...................	1,446,584	230,129	15.91	2,253,015	439,516	19.51
Champion....................	15,707,426	2,444,554	15.56	16,954,986	3,231,329	19.06
Isle Royale..................	2,973,761	461,687	15.53	2,937,098	552,468	18.81
Mass Cons..................	....	....	....	2,106,739	411,235	19.52
Michigan....................	2,891,796	453,683	15.61	2,875,341	568.399	19.77
Mohawk.....................	9,387,614	1,457,588	15.53	12,723,515	2,493,809	19.60
Osceola......................	18,938,965	2,942,239	15.54	18,588,451	3,511,358	18.89
Quincy.......................	18,827,557	2,981,121	15.83	16,194,838	3,157,993	19.50
Tamarack...................	15,824,008	2,448,240	15.47	9,832,644	1,853,453	18.85
Trimountain................	10,476,462	1,620,893	15.47	9,507,933	1,791,715	18.85
Totals....................	116,514,796	18,201,428	15.621	113,411,645	21,714,068	19.146

(a) Did not produce throughout the year.

Imports and Exports.—The imports of copper into the United States in 1906 were large; the greater part of these came from Mexico in the form of blister copper; some also came from Canada and some was received from South America, importations from the last source being on

the increase. Exports to Europe fell off decidedly. The demand from China which was so important a feature in 1905, was insignificant in 1906.

EXPORTS OF COPPER FROM THE UNITED STATES. (a)

Ore matte and regulus stated in tons of 2240 lb. Ingots, etc., in pounds.

Country.	1903.	1904.	1905.	1906.
Ore, matte, and regulus				
Exported to:				
United Kingdom	318	164	50	206
Germany	....	102	....	59
British North America	....	3,486	24,690	36,700
Mexico	10,667	15,175	12,948	10.600
Other countries	1,306	....	....	54
Total	512,291	18,927	37,688	47,619
Ingots, bars, plates, and scrap				
Exported to:				
United Kingdom	47,140,717	112,224,871	60,945,794	55,097,670
Belgium	4,209,720	9,365,791	4,997,206	6,475,054
France	53,745,221	99,888,455	74,604,455	80,703,723
Germany	71,130,077	103,825,445	104,575,864	96,629,040
Italy	7,774,016	15,297,091	15,800,967	19,777,296
Netherlands	96,927,346	147,678,581	130,675,386	151,650,293
Russia	10,411,679	22,333,578	18,418,982	9,523,992
Other Europe	16,516,663	29,064,494	25,279,162	25,260,807
British North America	2,644,831	3,472,614	3,019,450	4,176,135
Mexico	165,283	191,429	290,763	263,319
China	....	10,403,034	79,940,250	4,932,128
Other Countries	63,971	804,647	16,359,751	262,561
Total	310,729,524	554,550,030	534,907,619	454,752,018

(a) The imports of ore, matte and regulus are reported as gross weight, the copper contents not being stated.

IMPORTS OF COPPER INTO THE UNITED STATES.

(In pounds.)

Country.	1903.	1904.	1905.	1906.
Ore, matte and regulus				
Imported from				
British North America	(a)243,918	15,046,131	15,403,429	10,329,955
Mexico	(a) 39,261	20,803,961	28,890,239	31,690,058
South America	(a) 77	91,509	1,503,427	4,390,589
Other countries	(a) 1,656	3,006,121	4,308,205	2,874,289
Total	(a)284,912	38,947,722	50,105,300	49,284,891
Pigs, bars, ingots, plates, and scrap				
Imported from:				
United Kingdom	18,788,558	19,172,854	26,284,302	22,549,321
France	1,926,279	22,075	1,549,138	3,202,168
Germany	1,600,766	875,329	2,945,441	5,503,712
Other Europe	240,689	16,943	1,955,358	5,649,689
British North America	15,923,760	17,690,656	23,636,843	30,398,369
Mexico	89,361,100	97,965,593	102,646,343	85,595,359
Cuba	467,832	368,634	433,440	513,240
West Indies and Bermuda	317,112	373,743	278,502	399,569
Japan	3,604,643	80	....	6,752,468
Other countries	4,477,256	5,858,535	890,018	16,194,477
Total	136,707,995	142,344,433	160,619,385	176,558,390

(a) Tons of 2240 lb. copper content not given. The imports reported for 1904–1906 are the copper contents of ore, matte and regulus.

The large exports of copper from the United States to China, which began to attract attention during the early part of 1905, remained mysterious for a long time. It was early reported that China was requiring

copper for coinage purposes, but it was difficult to believe that this explanation, which later became generally accepted, could account for the very large amount of copper that was being taken by China; and even up to the end of 1905, especially when Chinese copper began to be resold to Europe and America, suspicion continued rife that a speculative movement was in some way involved in the transactions.

Some consular reports, American and British, published in 1906, threw much light upon the matter. Mr. Cloud, the American vice-consul at Hankow, reported that the copper coinage industry in Chekiang, "which for many months has proved a veritable gold mine for the group of mandarins in charge of it," had recently all but collapsed and the local mints had shut down, at least for the present. The alleged reason for this suspension was that they were waiting for the new uniform stamp adopted by the Imperial Government for all the provinces, but investigation disclosed that the merchants generally had instituted a boycott against the new coins, refusing to accept them in amounts exceeding 20c. Mexican.

Mr. Frazer, the British Consul General at Hankow, in a report dated in March, 1906, stated that the abnormal importation of copper for currency purposes, commented upon in his last previous annual report, was insignificant in comparison with the figures for 1905, in which year the copper imports represented one-third of the total foreign imports for the year. Ten-cash tokens poured from the mints in Wuchang, and the Hanyang arsenal set up nearly a score of presses for stamping the disks. The numbers struck at the three mints during the Chinese year are given as 1,864,000,000, 1,524,000,000, and 483,000,000, a total of 3,871,000,000. Late in the autumn, the Central Government, either alarmed by the rapid depreciation in the market value of these tokens, or jealous of the enormous profit netted by the Provincial Government, forbade the importation of ready-made disks. This check found Hupei with a stock of about 2000 tons of copper, which had cost on an average at least £80, and was not salable at much over £79. About 800 tons of ingots were resold to Europe before the Chinese new year (Jan. 25).

The exportation of copper from the United States to China in 1905 was almost an even 40,000 tons. The new ten-cash piece contains from 206 to 222 grains of pure copper, or say 33 cash to the avoirdupois pound. On this basis and the statistics of the mintage reported by Mr. Frazer, about 59,000 tons of copper was actually coined in 1905. China obtained some copper from the United States during 1904, in which year the American exports to China were not reported separately, and also obtained supplies from Australia and Japan. Complete statistics of the importation of copper into China in 1904 and 1905 are not yet available. It is, however, quite evident that the large purchases of copper by China

during 1905 were really for coinage purposes, and for nothing else; and, moreover, that the resales of Chinese copper at the end of 1905 were comparatively small in amount and were due to the necessity for realizing upon some uncoined copper, compelled by the late action of the Chinese Government.

Consumption.—The domestic consumption of copper in 1906 was far in excess of anything previously known in the United States, the average being about 55,660,000 lb. per month against about 51,000,000 per month in 1905. All the mills manufacturing sheet, wire and other finished forms of copper, all the brass mills and all the manufacturers of machinery into which copper and brass enter in part, were extremely busy throughout the year. The great extension of electrical work of all kinds, the construction of new electric roads and the application of electricity on existing railroads were largely responsible for the demand, which shows every sign of continuance through 1907. The capital stock of the American Brass Company, the largest consumer of copper, was increased from $12,500,000 to $15,000,000, on Feb. 26, 1907. About 15,000 men were engaged in the brass industry at Waterbury, Conn., in 1906. There were considerable additions made to the various manufacturing plants, including those of competitors as well as subsidiaries of the American Brass Company. In some of the shops the men worked on relays and in the casting mill they were employed for 24 hours daily. The Waterbury Brass Company erected a new rolling mill.

Stocks.—The stocks estimated do not indicate that at the close of the year there were 66,000 tons of copper available for immediate delivery. As production is reported from the mines, and periods varying from 20 to 75 days for different districts are required to transport and refine the copper, there must necessarily be a large amount of copper nominally included in the stocks, irrespective of any quantities of refined metal which may remain unsold. Our estimate of this is based on the proportions of copper marketed as Lake, electrolytic, casting and pig, and the relative time required for transportation and refining. Careful inquiry has made it evident that the unsold stock of copper in marketable form at the close of the year was probably less than 9,000,000 lb. No account is taken of stock of "mineral" at Lake smelting works, or of refined copper in consumers' warehouses.

Copper Mining in the United States.

Alaska.—The copper production of this Territory showed a large increase over 1905. There is no question that the copper resources of Alaska, near the sea-coast and easily workable, are large, and an important increase in production from this source is to be looked for in the near future. The Guggenheim Exploration Company has made large investments on

Copper river, and is preparing to build a railway to bring out the product, which according to all reports will be large. The erection of the new smeltery at Point San Bruno, San Francisco, will afford the necessary increased capacity for production. A good deal of the ore and matte produced in Alaska now goes to smelters on Vancouver Island and figures in their reports of blister copper. In our statistics this copper is deducted from the reports of those smelters and is credited to Alaska, where it originated.

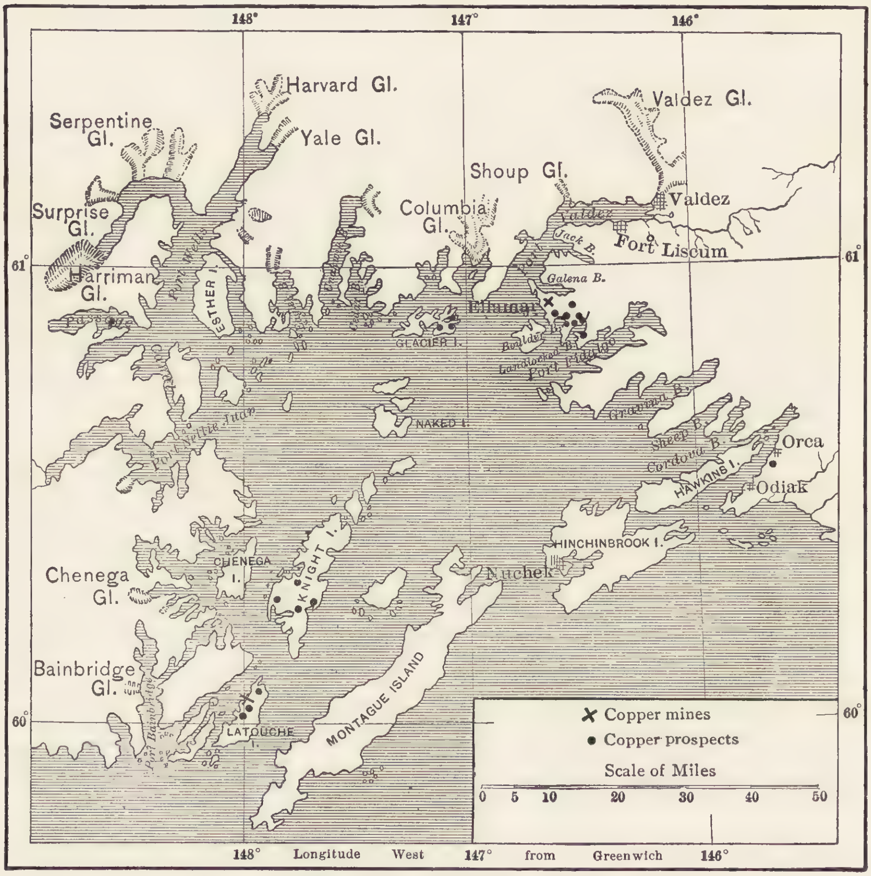
The list of producing mines in the Ketchikan district is gradually becoming larger. Nearly all are situated on Prince of Wales Island. Included among the shippers in 1906 were the Mt. Andrews, Uncle Sam, Mamie, and Stevenstown, all on Kasaan bay; Alaska Copper Company's Rush-Brown property on Karta bay, an inlet from Kasaan bay; Omar Mining Company's Khayyam on McKenzie inlet, a southern branch of Skowl arm; Niblack Copper Company's mine at Niblack anchorage; Cmyru Copper Company's Cymru mine on Moira Sound; Alaska Copper Company's Coppermount on Hetta inlet, and the Jumbo near Sulzer on the same inlet; and two west coast properties—Alaska Metal Company's claim (formerly known as Corbin's) at Bruce, and the Tyman's Red Wing. The entire group of islands of the Ketchikan district has been described by H. W. Turner as being fairly seamed with mineralized lodes, the ores being rich in sulphides and adapted to smelting operations. At present there are two smelters treating copper ores—one at Hadley on Kasaan bay on the eastern side of Prince of Wales Island, and the other at Coppermount on Hetta inlet in the southern part of the island.

The Alaska Copper Company, owning a group of copper properties situated on a high mountain at Hetta inlet, Prince of Wales Island, and a smelting plant at Coppermount, on the inlet, employed about 140 men at its mines and smelter. The smelter treated about 200 tons of copper ore per diem. The matte was shipped to Tacoma, Wash., for converting into blister copper.

The Alaska Smelting and Refining Company, of Hadley, Prince of Wales Island, received its ore supply from the Brown-Alaska company's Mamie mine, the Hadley Consolidated company's Stevenson mine, the Cracker Jack mine—all three situated near Hadley—and the Ellamar mine on Prince William sound. Beside these local ores, silicious copper ore, for fluxing purposes, was obtained from the Britannia mine, at Howe Sound, B. C., and the matte produced at Hadley was shipped to the Britannia works, at Crofton, B. C., for conversion into blister copper.

A considerable number of prospectors were at work in Prince William Sound, Alaska, during the summer of 1905, and previous to that time other prospects had been explored. In none of the places is machinery used, with the one exception of the workings of the Reynolds-Alaska

Development Company, at Boulder bay. The chief center of interest for prospectors is the vicinity of Copper mountain. This is a ragged-crested mountain, rising nearly 4000 ft. above sea level and situated about four miles southeast of Ellamar. Boulder bay is at the west base of this mountain and Landlocked bay at its south base. The rock of Copper mountain is greenstone, with a small amount of sediments (slate, graywacke, and quartzite).[1]



COPPER PROSPECTS NEAR PRINCE WILLIAM SOUND.

Two mines on Prince William Sound have demonstrated that copper ore of good grade occurs in this district and that it can be produced at a profit, notwithstanding the fact that the ore is shipped, at an expense of \$2.50 to \$3 per ton, to Tacoma, Wash., before it is smelted. Up to the present time nearly the entire output has been from the Gladhaugh mine. At

[1]From Bulletin No. 284, U. S. Geological Survey, 1906, "Copper and Other Mineral Resources of Prince William Sound, Alaska," by U. S. Grant.

the Bonanza mine existing developments warrant the prediction that there will be an early increase in production and that a large body of ore will be found available. None of the prospects, as developed in the summer of 1905, showed indications of as large an orebody as is known at either of the two mines.

Erosion in very recent time has been general throughout the Prince William Sound region, so that no considerable secondary concentration of ores exists. The ores of possible commercial importance have all the characteristics of primary deposits and are a phase of a general sulphide deposition along certain channels or zones. In general there is no reason to expect that stringers of ore on the surface will develop in depth to payable veins or that veins of considerable width at the surface will continue with unvaried dimensions and richness to great depths. On the contrary, it is known that orebodies pinch out in individual cases, and, on the whole, irregularity of form is to be expected. Developments of prospects should be confined to the following of ore. Running long cross-cuts to catch stringers or veins in depth is bad practice, since experience has shown that the continuation of the deposits is by no means assured.

Throughout the district much of the development work has been misdirected and nowhere except on Latouche island and at Virgin bay have excavations gone far enough to prove definitely the presence of workable orebodies. However, at a number of prospects the copper and gold contents of the ore are sufficiently high for profitable mining and these places are worthy of further prospecting. These facts, coupled with the location of many of the veins at or very close to tide water and the present demand for ores of this character for furnace mixtures, give reason to expect an increase of copper mining outside of the two mines already in operation. Should the future see the establishment on Prince William Sound of plants for smelting the copper ores of the Copper river district, for which purpose the coal of the Bering river or Matanuska field could be utilized, the prospect for mining on the sound would be still brighter.[1]

Arizona.—Of the chief copper-producing States, Arizona made the largest increase in 1906. The increase would have been larger had it not been for the shortage in labor, which was a difficulty all through the year, and moreover a shortage in the fuel supply which became a difficulty toward the end of the year. The flood at Clifton, about the end of November, which seriously damaged the smelting works of that camp, also contributed to the restriction of the output. The smelting works at Douglas and at Globe produced considerable copper from ore received from Cananea and Nacozari, Mexico, the amount of which has been deducted from their reports. Of the total production of this Territory, the Copper Queen

[1]From Bulletin No. 284, U. S. Geological Survey, 1906, "Copper and Other Mineral Resources of Prince William Sound, Alaska," by U. S. Grant.

works, at Douglas, produced 96,481,904 lb. (of which 93,201,519 lb. was from domestic ore, chiefly from Bisbee, and 3,280,385 lb. was from Mexican ore), and the Detroit Copper Company produced 16,906,348 lb., making a total of 113,388,352 lb. for these two interests of Phelps, Dodge & Co. The works of the Arizona Smelting Company, at Humboldt, near Prescott, were put in operation on March 19. These works are particularly interesting, in view of the fact that reverberatory furnaces fired by oil fuel are employed, this being the first installation of that kind on a large scale.

(By James Douglas).—The production of Arizona during 1906 exceeded anticipation. The increase was attributable to legitimate expansion of operations, rather than to illegitimate pressure upon the mines to meet the extraordinary demand for copper. The surplus came mainly from the Warren district, and from miscellaneous sources. The Copper Queen mine production, treated at the large Douglas smelter, or shipped to Globe as sulphur ores to aid in matting the oxidized ores of the Globe district, exceeded the production in 1905 by approximately 18,000,000 lb. The production of the Calumet & Arizona smelter was also approximately 12,000,000 lb. in excess of that of 1905. This excess was contributed in part by mines, other than the Calumet & Arizonia mines, controlled by the same group of owners, and which became active producers during last year. Their output—with that of still other mines of the same group—will probably be doubled during 1907. While the Copper Queen mine may not increase its own production during 1907, not only will the Calumet & Arizona group yield more copper, but some other companies in the district will enter the active list, and continue to make the Warren district lead during 1907, as it did during 1906.

The Clifton district would during the past year have exceeded its actual production had it not been for the serious floods in early December which disabled all the works in the district for a time. As it was, however, the district produced about 5,000,000 lb. more copper than it did in 1905. The United Verde likewise increased its production, and the Globe smelter turned out 8,500,000 lb. of copper in excess of the previous year. Of the Globe output about 12,000,000 lb. was from Mexican ores or mattes, secured primarily for their sulphur contents. The quantity of sulphur, however, derived from the ores of the Globe district is increasing so rapidly that the prospects are that within a twelvemonth Globe will be independent of imports for its supply. One of the large veins under the Buffalo hill has recently been struck, in drifting from the Gray shaft of the United Globe mines, which contains besides copper and sulphur a notable amount of the precious metals. The Continental group of the Old Dominion company is producing more copper and sulphur than was anticipated, and the lower levels of the Old Dominion mine likewise are also producing sulphur

ores in such quantity as to warrant the prediction that, with ore coming from the Arizona Commercial and other local mines, the coming year will relieve the district of the necessity of looking to foreign sources for any ingredients of its furnace mixture.

In addition to these old standard sources of supply the Helvetia company has been producing; the old San Carlos Copper Company, now reorganized as the Saddle Mountain Mining Company, ran intermittently and produced a considerable amount of matte, and the smaller mines of Yavapai county, stimulated by the facilities offered by the Humboldt smelter, yielded more copper than the county ever before supplied from these secondary sources.

Adding together the increased production from the old and these contributions from new mines, the Territory has helped to meet the excessive demand of the world for copper by increasing her output from 223,000,000 lb., in 1905, to approximately 263,000,000 lb. in 1906, and a similar increase may be anticipated during 1907.

The furnace capacity of the Southwest was increased during 1906, and plans are suggested for further augmentation during 1907. The Copper Queen smelter at the present time consists of nine furnaces. A tenth is in process of erection and an increase is contemplated. The Calumet & Arizona smelter, it is understood, intends adding two new furnaces to its present plant of five and enlarging the five existing furnaces, thus doubling its capacity. The Shattuck-Arizona company, of Bisbee, whose ores have heretofore been smelted at the Copper Queen works at Douglas, has decided to erect its own smelter near those of the Copper Queen and the Calumet & Arizona companies. The Old Dominion will add one additional furnace, thus raising the number to five. The Detroit Copper Company is remodeling its converting plant. The new concentrator of the Arizona Copper Company was put into commission in July last, and the Detroit Copper Company's enlarged concentrator has been in full operation for several months. The Saddle Mountain Mining Company, as already stated, has become an independent producer, but the most important addition to the metallurgical equipment of the territory is the new Humboldt smelter on the Agua Fria, provided with both reverberatory and cupola furnaces.

Reverberatory smelting for cleaning the slags and handling the flue-dust and fine concentrates is supplementing the blast furnaces. Reverberatories will be erected at Douglas and Globe. The reverberatory furnace erected in 1906 has been running steadily at the Detroit Copper Company's works, at Morenci, receiving the slag continuously from the cupola settler and melting the flue-dust. The results have been, from both a technical and an economical point of view, very satisfactory, reducing the copper in the slags approximately 0.1 per cent., while the total cost of operations has been somewhat less than would have been the cost of briquetting and

smelting the 6000 tons of flue-dust which have been treated. The fuel employed has been crude oil, and the experiments have been directed to the best method of adding the flue dust to the stream of liquid slag as it enters the furnace from the settlers. If inserted by the barrow or carload, though it is rapidly melted down, it creates a superficial layer of slag, which, though liquid enough to flow, is more silicious and higher in copper than the lower stratum, with which it can be mixed only by mechanical agitation. The method involving least labor and the most perfect mixture is by allowing a stream of flue dust to strike the jet of incandescent gas; but the corrosive effect on the roof of the furnace of the dust thus created increases the cost of repairs. Experiments looking to the treatment of fine concentrates by throwing them upon this bed of liquid slag indicate that a very rapid process of liquation proceeds, the sulphides in the concentrates melting immediately, and descending through the liquid slag, leaving infusible silicious shells upon the surface which tend to form accretions on the walls of the furnace, as well as a stratum of highly silicious slag. Where, however, it is desirable to save the sulphur contents of the concentrates, no better method could probably be adopted of reducing to a minimum by volatilization the sulphur contents, while making a low-grade matte for subsequent treatment in the cupola. The percentage of costs chargeable to reverberatories during 1906 was as follows: Fuel oil, 73.11 per cent.; labor, 17.35 per cent.; repairs, 8.85 per cent.; shop expense, 0.42 per cent.; water, 0.21 per cent.; light, 0.16 per cent.; total, 100 per cent.

The furnace is 50 ft. long; the heat of the gases in the combustion chamber is 1400 deg. F., and at the flue end 1210 deg. F. A boiler to generate steam is being erected, and the economy which will result from the utilization of the escaping gases has therefore not yet been determined, but will notably decrease the cost of furnace treatment.

No radical changes have been made in mining. The Arizona Copper Company has introduced underground electrical traction, and the Copper Queen is planning to concentrate all its ores at the new Sacramento shaft, to use a skip, and at surfaces by a belt conveyor to distribute to a train of cars the ores from every section of the mine, thereby securing a more perfect smelting mixture than is now obtained. But at the Cananea mine the caving system so successfully used in the Lake iron mines has been adopted.

The mining and smelting operations during the latter months of 1906 had to struggle with the same conditions that beset nearly all the industries of the country—scarcity of fuel, scarcity of labor, and frightful congestion on all the railroads serving the territory. The cessation of the fuel-oil supply from both Texas and California necessitated a sudden return to coal, and strained the only accessible coalfields to the utmost

in meeting the exigency. The increased demand nearer home for Colorado and other Western cokes obliged the smelters of the Southwest to rely largely for their coke supply on Eastern sources, which, under existing railroad conditions, were only able to forward it with precarious uncertainty. But all the interests, appreciating the adverse conditions arising from excessive prosperity, bore with commendable patience and resignation the slight evils to which they were exposed in consideration of the benefits which the prevailing high price of their product conferred.

California.—The increase in the production of this State was due especially to the operations of the Mammoth Copper Mining Company, which made a large output in 1906. The new smelter which is being erected by the Balaklala company to treat the ore of the Balaklala and Trinity mines will increase the copper production of California in 1907, but it will hardly come into full operation before the autumn. A small increase in the production is shown by the mines outside of Shasta county, especially in Calaveras county. Copper mines in Nevada county attracted attention, and some deposits long known, but never worked, were opened. The Fresno mines made a small production. A new copper district, the Greenwater district, was discovered in Inyo county in 1906. It appears to contain important copper resources, but its development will be slow because of unfavorable natural conditions. A good deal of the ore and matte produced in California is shipped to Utah for smelting, this being the case with all of the Mammoth matte. The suspension of smelting by the Bully Hill company, pending the completion of a railway connection, and the failure of the Mountain Copper Company to resume production on its former scale, caused the increase in California production in 1906 to be less than was anticipated. The old smelter of the Mountain Copper Company at Keswick, Shasta county, which was shut down on account of smoke litigation, was idle throughout 1906, but the company's new smelter at Oakland, on San Francisco bay, was actively working. In connection with this plant there is a sulphuric acid factory and an electrolytic refinery. The Keswick smelter will be put in operation again in 1907, and the copper production of the Mountain Copper Company will show a considerable increase, but it is unlikely to reach the former maximum.

The Mammoth added two more blast furnaces to its plant, making a total of five; also mechanical roasters, and a reverberatory furnace to treat the fine ore and flue dust. This is another example of the increasing tendency to install reverberatory furnaces at copper-smelting plants. The Mammoth is looking up silicious ores in Oregon to be used as flux at its smelter at Kennett, Cal. The districts adjacent to Ashland are being examined with a view to obtaining a steady supply. The present transportation rates from Ashland, Oregon, to Kennett, California, amount to \$2.40 per ton in carload lots, though if fair tonnage becomes available, the

rate may be reduced to $1.50 per ton. The smelter makes no charge for treatment of these ores and pays 75 per cent. of the value contained. This makes it possible for the prospector, leaser and quartz mine owner to realize a fair profit on low-grade ore.

The copper region of Shasta county is about 30 miles long and a few miles wide; it may be compared to a crescent with Iron Mountain at the western end and the After Thought mine at the eastern extremity. While the whole area is mineralized, the productive districts are limited to two smaller zones, each having characteristics of its own. At the western extremity are the large masses of heavy sulphide. This district includes the Iron Mountain, Balaklala, Trinity, and Mammoth mines, and a number of prospects from which important developments may be expected in the future. These deposits occur almost entirely in acid volcanic rocks, notably a rhyolite made up of a succession of irregular flows and tuffs that have been so compressed and folded as to render obscure the original character of the mass. When fresh the rock is gray, generally with distinct phenocrysts of quartz, and in the neighborhood of zones of mineralization it is highly impregnated with pyrite or an iron stain. The principal deposits of the district lie on well-marked lines of shearing and mineralization. These are numerous, but no definite relation can be traced between them, though individual zones may extend for a mile or more.

Colorado.—The copper production of this State showed but little change from 1905. As in previous years the larger part of the production was derived from Leadville, which according to the State Commissioner of Mines furnished 4,028,497 lb. The San Juan region ranked next, with an output of 2,094,066 lb.

Idaho.—The copper production of this State showed a large increase in 1906. The largest part of the production came from the Snowstorm mine, in the Coeur d'Alene. The producer next in importance was the White Knob mine, at Mackay, Custer county. A small amount of copper ore was shipped from the Seven Devils district. The ore and matte of Idaho are shipped to other States and to British Columbia for smelting to pig copper, wherefore the production credited to this State is deducted from the reports of smelters in other States and British Columbia. The Snowstorm ore, which is highly silicious, is largely employed by the smelters for converter lining.

(By Robert N. Bell).—The most important development during 1906 was at the Snowstorm mine, three miles above Mullan, in the Cœur d'Alene. This property developed bonanza proportions, and with a light prospecting equipment produced shipping ore and concentrates to a total gross value of $1,200,000, and returned nearly half of that amount as net profit. There are a number of good copper prospects, both east and west of the Snowstorm, actively developed at present, and also on the opposite side of the

Cœur d'Alene river, in the Parke (or Stevens Peak) copper belt. The Parke copper belt has some remarkable surface showings of rich, brown, spongy gossan sprinkled with pebbles and kidneys of chalcopyrite. One of the main veins of this belt is traceable by such croppings for 15 miles. The Monitor mine on this belt has already made several shipments of high-grade sulphide ore, containing about 16 per cent. copper and $5 or $6 in gold.

The Lost Packer mine, at Loon creek, Custer county, with rich copper-gold ore, was equipped with a 100-ton matting furnace, which was started in the summer, but proved a failure. This property contains a shoot of 30 per cent. copper sulphide ore carrying 3 or 4 oz. of gold, per ton, together with 12 or 14 oz. silver. It is the intention of the company to readjust the smelter and get it into commission early in 1907.

From the Seven Devils district, in Washington county, 2792 tons of copper ore, mined by leasers, was shipped to the Sumpter Valley smelter; it contained 636,000 ℔. of copper, 7942 oz. silver, and 162 oz. of gold. The Copper Queen, in Lemhi county, made several shipments of high-grade bornite from a fissure in a schist and slate formation on Agency creek. This ore runs better than $10 per ton in gold, and over 30 per cent. copper. The deposit is being actively developed. The Weimer copper mine in Fremont county shipped several cars of high grade carbonate and oxide ore containing 64,000 ℔. of copper.

The White Knob mine, at Mackay, Custer county, was successfully handled by a Western company, holding a lease of the property, which mined and smelted 50,000 tons of ore, containing more than 4 per cent. copper. The ore is principally oxide and carbonate, and is mined from big surface quarries. The smelter is a matting plant, using iron sulphide shipped in from Bingham, Utah, to furnish the necessary sulphur, as the White Knob mine, although developed to a depth of 700 ft., never showed an important increase of sulphide ore. The ore occurs in lenses and irregular masses in garnet rock, a soft, gray porphyry, near a contact of pure blue limestone and eruptive granite.

The present leasers have replaced the old company's electric hauling system with a Shay geared locomotive, which hauls the ore from the mountain to the smelter over a 6 per cent. grade six miles long, at a greatly reduced cost. Operations would have been very profitable during 1906 but for a disastrous fire in the summer which destroyed the ore-house and machine shops at the smelter, and caused a loss of nearly $100,000, and interrupted smelting for more than a month.

The smelter has two stacks with a capacity of 600 tons per day, but the ore resources have justified the operation of only one thus far. The necessity of purchasing sulphide ore to make matte greatly increases the cost

of treatment. The total gross content of copper ore and matte shipped from Idaho during 1906 was 11,640,565 lb.[1]

Massachusetts.—In 1906 this State became a producer of copper, small pockets of pyrites containing 5 per cent. copper and upward having been found in the Davis mine, at Charlemont, which has for many years been a large producer of non-cupriferous pyrites (sulphur ore). The selected copper ore is shipped to New Jersey for smelting. The Charlemont Copper Company was organized to develop a copper prospect near the Davis mine.

Michigan.—The production of the Lake Superior district showed only a small increase. In most cases the output of the older mines showed decreases, which in some cases were large. The Atlantic mine suffered a cave-in, which caused the mine to be abandoned; the Quincy also sufered from a cave, while the Tamarack had a fire. A strike of the miners at the Adventure, Quincy and Michigan mines in mid-summer also was a drawback to production, but it did not prove to be of long duration or serious. The Calumet & Hecla made a considerable increase in output, as did also some of the new producers. These gains a little more than offset the losses of the other mines.

COPPER PRODUCTION IN MICHIGAN.
(Pounds of fine copper.)

Mines.	1900.	1901.	1902.	1903.	1904.	1905.	1906.
Adventure	....	....	606,211	2,182,608	1,380,480	1,606,208	1,552,628
Ahmeek	....	....	....	....	350,000	1,552,957	3,077,507
Allouez	....	....	....	....	....	1,167,957	3,486,900
Atlantic	4,930,149	4,666,889	4,949,368	5,505,598	5,321,859	4,049,731	1,439,082
Baltic	1,735,060	2,641,432	6,284,819	10,580,997	12,177,729	14,384,684	14,397,557
Cal. & Hecla	81,403,041	82,519,676	81,248,739	76,490,869	80,341,019	83,812,370	94,529,821
Centennial	....	806,400	....	....	641,294	1,446,584	2,253,015
Champion	....	....	4,165,784	10,564,147	12,212,954	15,707,427	16,954,986
Franklin	3,663,710	3,757,419	5,259,140	5,309,030	4,771,050	4,206,085	4,571,570
Isle Royale	*Nil.*	2,171,955	3,569,748	3,134,601	2,442,905	2,973,761	2,937,098
Mass	*Nil.*	837,277	2,345,805	2,576,447	2,182,931	2,007,950	2,106,739
Michigan	....	....	166,898	275,708	2,746,127	2,891,796	2,875,341
Mohawk	....	160,897	226,824	6,284,327	8,149,515	9,387,614	9,352,252
Osceola	11,200,000	13,723,571	13,416,398	16,059,636	20,472,439	18,938,965	18,588,451
Phoenix	....	93,643	....	202,823	1,162,201	273,219	*Nil.*
Quincy	14,116,551	20,540,740	18,988,491	18,498,288	18,343,160	18,827,557	16,194,940
Tamarack	18,400,000	18,000,852	15,961,528	15,286,093	14,961,885	15,824,008	9,832,644
Trimountain	....	....	5,730,807	9,237,051	10,211,230	10,476,462	9,507,933
Winona	....	....	101,188	1,036,944	646,025	*Nil.*	278,182
Wolverine	4,778,829	4,946,126	6,473,181	8,999,318	9,764,455	9,464,418	9,548,123
Others	4,000,000	640,591	700,067	75,000	50,000	....	....
Totals	144,227,346	155,507,465	170,194,996	192,299,485	208,392,485	218,999,753	224,071,103

Another furnace was built at the Michigan Smelting Company's plant. It is larger than any of the five previous furnaces, having a capacity of 100 tons daily, and being 16x40 ft. in dimensions. Of the old furnaces, two are 16x36 ft., two are 14x23 ft., and one is 15x18 ft.

[1]This figure, reported by Mr. Bell, does not allow for loss in smelting. In our editorial statistics, Idaho is credited with the actual production reported by the smelters as originating in that State.

Dividends declared in 1904, 1905 and 1906 were as follows:

Mine.	1904.	1905.	1906.	Mine.	1904.	1905.	1906.
Atlantic........	....	$ 50,000	....				
Baltic..........	....	1,250,000	$1,400,000	Quincy........	$500,000	$600,000	$1,250,000
Calumet & Hecla	$4,000,000	5,000,000	6,000,000	Tamarack.....	90,000	120,000	480,000
Central.........	....	160,000	....	Trimountain...	....	....	....
Champion......	200,000	1,000,000	1,200,000	Wolverine.....	450,000	660,000	660,000
Osceola........	192,300	384,600	1,541,400				
				Total........	$5,432,300	$9,224,600	$11,131.400

In the following table the dividend disbursements are given for every half decade from 1850 to 1900, and subsequent years up to 1906:

Year.	Dividends.	Year	Dividends.	Year.	Dividends.	Year.	Dividends.
1850..	$84,000	1870......	$ 700,000	1890.	$3,415,000	1902......	$3,440,000
1855..	168,000	1875......	1,920,000	1895.	3,280,000	1903......	4,980,000
1860..	120,000	1880......	3,080,000	1900.	9,811,200	1904......	5,432,300
1865..	510,000	1885......	1,970,000	1901.	7,496,900	1905......	9,224,600

(By W. Spencer Hutchinson).—Extraordinary activity in exploratory work characterized the Lake Superior copper industry during 1906. This was stimulated by the high price for the metal, and by the remarkable success attending the developments of the last seven years on the Baltic and Kearsarge amygdaloid veins.

The first copper produced from the Baltic vein was by the Baltic mine in 1899. About this time the same vein was discovered at the Champion mine, and in 1902 that property commenced producing. At the present time these mines, together with the Trimountain, which lies between them, have 12 working shafts, proving the lode for a length of nearly four miles. Explorations are now in progress on the Atlantic, Section 16, and the Superior tracts to the northeast, and on the Globe and Champion lands to the southeast, with the expectation of developing the extension of the Baltic vein.

The development of the Kearsarge vein has been slower, but although it was worked to some extent in the Kearsarge and Wolverine mines prior to 1898, it was in that year that the Wolverine mine paid its first dividend. Since then the development of the vein has been rapid until now eight mines are shipping from this vein. From the Mohawk mine on the northeast to the Calumet & Hecla, Section 19, on the southwest, a distance of more than five miles, there are 20 working shafts, all of which are producing copper rock. Work is now in progress on the Fulton tract on the northeast extension and on the C. & H., Section 23, the Tecumseh and Caldwell mines to the southwest, which will explore this remarkable vein for a further distance of five miles.

In Keweenaw county, the Keweenaw Central R. R. is in process of construction from Lac La Belle to some point near Calumet, where it will join the other railroads. It parallels the old wagon road between Calumet and Copper Harbor, passing near the old mines where great waste dumps and ruins of old buildings give evidence of former activity.

The Calumet & Hecla Mining Company, the Keweenaw Copper Company, and the Tamarack Mining Company now hold large areas on the copper range in this end of the district and all are carrying on extensive explorations. The Cliff and the Central were the best known of the old mines. In these, as in the others, the work was confined to transverse veins crossing the bedded rocks, and although the amygdaloid beds were known to carry copper and were explored to some extent, no producing mines were opened on them. The work now in progress is directed toward the exploration of the bedded deposits which have made the great producing mines in other parts of the district. The first work is by diamond drilling, by which a geological cross-section is obtained, locating the copper-bearing strata. Three shafts are now being sunk in the Montreal amygdaloid, and one on the Medora amygdaloid, all with promising prospects.

In Ontonagon county, in the south-western end of the district, work is progressing at many points. The Victoria mine has started milling and the results are awaited with much interest. The power plant at this mine is of remarkable novelty. Water power is developed by means of a dam and canal on the west branch of the Ontonagon river, giving a fall of about 70 ft. and yielding more than 4000 h.p., of which only a fraction is now utilized. Three compression pipes of a Taylor hydraulic air compressor are installed, of which one is now operating. The air, at a pressure of 116 ℔. per sq.in., is stored in a chamber cut in the solid rock, at a depth of 300 ft., and is delivered by pipes to the mill and mine. It operates the stamp head and is used in engines, hoist, pumps and drills, furnishing all the power required on the property. The air chamber is said to furnish storage sufficient to operate the present plant for a period of five hours, and being sealed by a water column it has the advantage over the receiver of the ordinary compressed air system, that the air is delivered at uniform pressure and the last cubic foot of air at the same pressure as when the chamber is full. Air compressed in excess of the requirements must be allowed to escape and a 12-in. blow-off pipe is provided for this purpose. This operates at intervals, several times an hour, throwing a tremendous stream of water into the air to a hight of several hundred feet, and making an artificial geyser of surpassing grandeur and beauty. All this disturbance causes not the slightest quiver of the pressure gage. The Taylor hydraulic compressor furnishes air which is dry and absolutely clean and it is pronounced an unqualified success.

Montana.—The decrease in the copper production of this State was due primarily to shortage of labor. Even if the supply had been adequate, it is doubtful if the production could have shown any increase, because the smelting capacity appears to have been utilized at the maximum in 1905. Plans are now being carried out to increase the smelting capacity, which will be consummated during 1907, large additions being made to the Great Falls works.

John Gillie, superintendent of mines of the Amalgamated company, estimates that each foot of shaft at Butte cost not less than $100, average; for while many cost less, many cost more, nearly all sinking below the 1000-ft. level averaging $150 per ft. Taking it for granted that the total length of these shafts combined is 108,900 ft., the cost of this part of mining in Butte foots up to $10,890,000. There are about 150 shafts in Butte, in 30 of which sinking is in progress. The deputy inspector of mines estimates that the underground openings from the shafts aggregate 800 miles.

According to the testimony of E. P. Mathewson, general manager of the Washoe smelter, before the master in chancery in the farmers' injunction suits against the Washoe smelter, at Butte, Mont., Sept. 11, 1906, the output in metals of the Washoe smelter for eight months of 1902 and for 1903, 1904 and 1905, and the first six months of 1906, was as follows:

Year.	Copper, Pounds.	Silver, Ounces	Gold, Ounces.	Year.	Copper, Pounds.	Silver, Ounces	Gold, Ounces.
1902..	82,992,361	3,368,133	15,027	1904...	138,078,499	6,481,318	46,344
1903..	109,726,420	5,190,879	31,183	1905.	165,505,144	7,046,485	49,685

The production in the first six months of 1906 was 94,244,911 ℔. of copper, 3,811,735 oz. of silver, and 22,567 oz. of gold. The smelter employs 2100 men. Since 1902 the gross value of the ores handled has been $104,-316,089. The biggest items of expense have been: $7,007,324 for labor, $4,293,455 for coal, $4,012,000 for coke, $740,047 for lime rock, $1,316,020 for machinery, $53,896 for lumber and $1,480,813 for freight.

Formerly the Anaconda Copper Company, under the inspiration of the late Hamilton Smith, used to publish an annual report which in the fulness and detail of the information communicated to its stockholders excelled the reports of most mining companies at that time. Since the Amalgamated Copper Company acquired control of it, in 1899, it has made no report until 1906, when a considerably detailed statement of its operations in 1905 was made public.

The production of copper during 1905 was 95,443,730 ℔.; of silver, 3,116,880 oz.; of gold, 19,165 oz. The output was derived from 1,626,306 tons of ore and other material, which yielded an average of about 60 ℔.

of copper, and 2 oz. silver per ton. In 1898, the year of the last previous report, the averages for ore were 85 lb. of copper and nearly 3.5 oz. of silver. The average cost of mining, including development work, was about $3.50 per ton in 1905, against $3.77 in 1898—no great reduction, but in view of the deeper mining somewhat better than appears at first sight, while, moreover, an allowance is included for depreciation of plant. In 1897, however, the cost of mining was only $3.46 per ton. The cost of reduction in 1905 was about $2.50 per ton, including depreciation, and presumably rental of the works, leased from the Washoe Copper Company. In 1897 concentrating came to about 75c. per ton, smelting to $1.85, and converting to 76.5c., total, $3.365, but in this figure no allowance is made either for depreciation, or for the investment in plant, wherefore a comparison which would show just what has been accomplished by the new reduction works is impossible.

However, President Ryan throws some light on this subject by the statement that the Washoe works treated 1,626,306 tons of material, including ore from the mines, together with slags, slimes and flue dust from the old works at a difference, after paying rental, of $1.5886 per ton in decreased operating cost and increased metal extraction as compared with the old works in the last year of their operation. This saving amounted to $2,583,549 in the aggregate. Although the cost of the new works was about $8,000,000, their construction was plainly justified, the return on the investment being upward of 30 per cent. Also, although the first cost of the plant was high ($3 per ton on the capacity, or rather ore treated, in 1905) the benefit of the large outlay is clearly apparent in the high efficiency, which manifests itself largely in the increased extraction of metals.

The mines in 1905 produced 1,543,316 tons of ore, wet weight, of which 61,149 tons of smelting ore remained on hand at the end of the year, and 1,470,694 tons were treated at the works. The works treated for all companies during the year 2,650,868 tons of material, of which 1,626,306 was for the account of the Anaconda Company. The cost of production in 1905 was upward of 10c. per pound of copper. The production of the Anaconda in 1906 was 94,963,835 lb. of copper, 2,979,908 oz. of silver, and 15,985 oz. of gold.

(By B. E. St. Charles).—The smelters which treated copper ore in Montana in 1906 were the Washoe and Great Falls plants of the Amalgamated company, the Heinze works, the Clark works, and the works of the American Smelting and Refining Company at Helena. The works of the Pittsburg & Montana Company were not operated after April. The Heinze works, which were acquired by the Red Metal Company, early in the year were operated until July 1, when they were closed down to be dismantled, and the ore produced by that company was shipped to the Washoe works.

The latter works treated also the output of North Butte company, besides that of smaller producers. The Washoe works treated an average of about 8000 tons of ore per day, while the Great Falls works treated 3500 tons per day. The average daily output of the entire Butte district was about 12,500 tons of ore. The average daily shipments from the Amalgamated company, exclusive of the Boston & Montana, were a little upward of 6000 tons.

Extensive development work was carried on in all of the large mines, nearly all of the main shafts having been sunk from 100 to 400 ft. each. Large bodies of ore were opened in the Cora, North Butte, Anaconda and Red Metal mines; and many mines that had not been operated for years on account of litigation were placed in commission and their development was begun. Among these properties were the Tramway, Snohomish, Nipper and Berkley. A shaft was started on the Badger State; another on the Greenleaf, and another on the Poser, by which this claim and the Elm Orlu, Clark mines, will be opened. The General Development Company started another on the Granite Mountain, adjoining North Butte.

About the middle of 1906 the Anaconda company bought the Belmont from the Coalition company for the purpose of using it as an outlet for the ore of the Anaconda mines. The company resumed operations in the Buffalo and Pacific mines, sunk the shaft on the High Ore from the 2200- to the 2600-ft. mark; added 300 ft. to the Neversweat shaft, crosscut the vein of the Anaconda at the 2400-ft. and connected the 2000-and 2200-ft. with a raise in the vein. Late in the fall it began equipping the Diamond with electrical apparatus for operating all of the compressors of the Amalgamated system.

The Boston & Montana retimbered the shafts of the Mountain View and West Colusa mines; and placed the Leonard in shape for increased ore production. It decided to install an electric-haulage system underground in its principal mines. It also began enlarging its smelter with a view of treating 1000 tons more ore per day.

North Butte started two crosscuts north from the 1200-and 1600-ft. levels of the Jessie to cut supposed veins in the Berlin group and sank its main shaft 200 ft. It opened the Edith May and Miners' Union vein, one of the largest and best in the district, and exposed 100,000 tons of ore in the old Speculator vein. It equipped its main shaft with an engine having capacity for 3500 or 4000 ft., and substituted skips for cages in the shaft and equipped the levels with skip pockets. Its possessions were greatly increased, several undeveloped claims having been bought by it. Toward the latter part of the year it decided to resume sinking in the old shaft on the Berlin, in order to have another opening in its property.

La France Copper Company operated the Lexington property continuously, but did not make much of a showing on account of the fact that

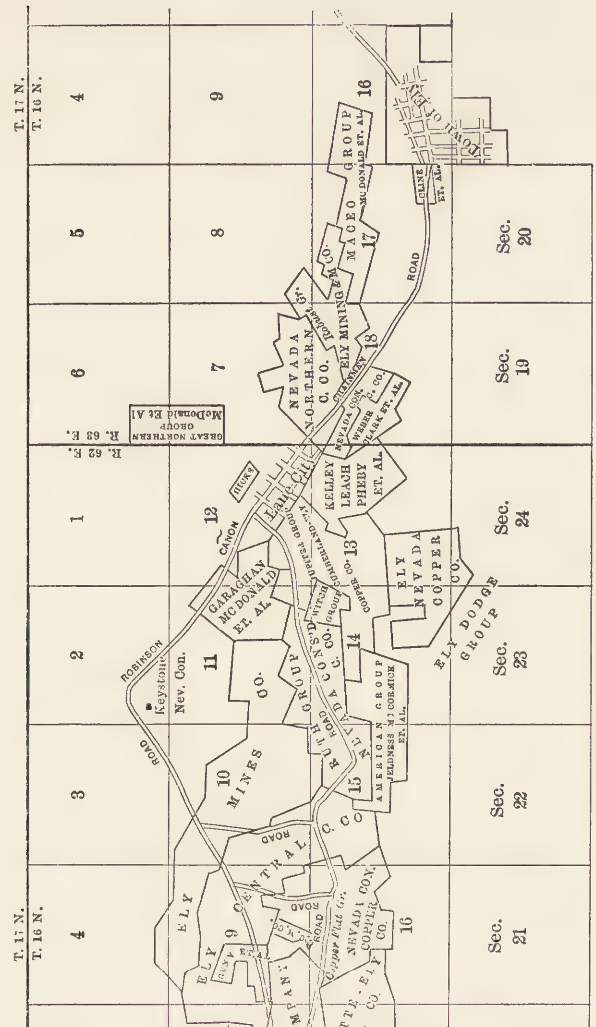
its work was confined to the upper levels. In November it finished unwatering the lower levels. Its daily average output was between 100 and 150 tons of ore carrying copper, silver, gold, zinc and lead. The destruction of the Montana Zinc Company's plant by fire in June prevented La France from realizing on its zinc ore, for it had no market for that class of ore after the fire.

Pittsburg & Montana made a better record than it did in 1905. It closed its smelter in April and began shipping ore to the Washoe works. Shipments during the latter part of the year averaged more than 150 tons of ore a day.

Improvements begun at the Clark plant in 1905 were finished about the middle of 1906. They included a 352-ft. concrete stack, with dust-chamber and connecting flues, and the substitution of converters for blister-copper furnaces. The mines were productive up to Oct. 1, but from about that date up to the close of the year the output averaged 100 tons a day less and the concentrator was idle part of the time. Custom ores were bought to make up the deficiency, and with their assistance the furnaces were kept continuously in operation.

Missouri.—Copper ore in workable quantity has been opened by the North American Lead Company, of Fredericktown, at the southern end of the disseminated lead district. A matte-smelting furnace was erected, which was put in operation early in 1907 and since then has turned out considerable copper. The ore averages 6 per cent. copper and 3 per cent. nickel-cobalt.

Nevada.—Ely was the center of attraction during 1906. The Nevada Northern Railway was completed during the autumn, and construction of the works of the Nevada Consolidated was begun. It is unlikely that the plant will be completed before the spring of 1908. The dressing works will be equipped with rolls and Huntington mills. During 1906, the Guggenheims purchased 51 per cent. of the stock of this company, the control being finally secured by the purchase of some large blocks at $12.50 per share. At the end of 1906 arrangements were made for the amalgamation of the Nevada Consolidated and Cumberland-Ely companies, which will be jointly interested in the works under construction. The Giroux company continued development work in its mines, and opened further large bodies of ore. This company will probably make arrangements for the erection of dressing works during 1907. The great interest in the mines of this district led to the bringing out of a number of new companies, the merits of which are more or less doubtful. However, there is no question that the bodies of low-grade, disseminated ore in this district are of enormous extent, and as soon as the smelting works have been put in operation Ely will be among the very important copper producers of the United States. The ore of Ely averages a little upward of 2 per cent.



MAP OF MINING CLAIMS, ELY, NEVADA.

copper. It is estimated that a year after the several works are put in operation, the Nevada Consolidated and Cumberland-Ely Companies will each be producing at the rate of 30,000,000 lb. per annum, while the Giroux will be turning out 10,000,000 lb.

New Mexico. (By Charles R. Keyes).—The output of copper in this Territory continues to increase slowly; but, with the full operation of new mines which have been opened, a rapid increase may be expected. The bulk of the copper ore comes from the mines of Grant and Socorro counties. In these districts extensive new operations were carried on during 1906. The Mogollon district will be especially worthy of attention as soon as better transportation facilities shall have been provided. Mines in Doña Ana county were more active than for some years previous. Provision has been made for handling the low-grade copper ores of San Miguel county. Some promising prospects have been opened up in the archæozoic schists of Santa Fe county. Considerable attention has been paid to the very low-grade copper deposits of the Red Beds, extending over large areas in the Southwest, and methods of leaching these ores have been proposed.

The Santa Rita mines are taking out native copper chiefly. This occurs in large plates in brecciated zones in porphyry. The new copper developments in the Magdalena mountains continue to be of exceptional interest on account of the relationships which the ores bear to those of zinc and lead. Important new developments have taken place in the Burro mountains, in Grant county, and some interesting deposits have also recently been opened in the Caballos mountains in Sierra county.

North Carolina.—The Union Copper Company made a small production of ore, which was shipped to Tennessee for smelting, where it is greatly desired because of its silicious character. Other producers in North Carolina shipped small parcels of ore to the Eustis Smelting Works, at Norfolk, Va., and to smelters at New York, but the aggregate of their production was insignificant.

Tennessee.—The production reported for the Eastern and Southern States is due chiefly to Tennessee. The increase in 1906 is attributable almost entirely to the enlargement of operations by the Tennessee Copper Company. Outside of that company, the Ducktown company, in the same district, was the largest producer. The Ducktown company smelts its ore to matte, which is shipped to the Tennessee Copper Company for conversion into blister copper.

Utah.—The decrease in the copper production of this State was statistical rather than real, being due largely to a more precise allocation to other States of the copper which Utah smelters received from them than was possible in previous years. Construction of the new dressing works for the Utah Copper Company and Boston Consolidated was begun.

These plants should add materially to the production in the latter half of 1907. When the plans, now being carried out, are fully consummated it is expected that these two companies will produce at the rate of 60,000,000 and 30,000,000 ℔. per annum, respectively. A drawback to the situation in Utah, however, is the recent injunction against the smelters of the Salt Lake valley, which probably will cause the abandonment of certain plants and the removal of some departments of others. These conditions are likely to upset to some extent the smooth running of the industry in 1907. The new Garfield smelter (American Smelters Securities Company) did not get into regular operation until late in 1906 and consequently was only a small producer. The smelters of Utah receive a great deal of ore and matte from other States, and although the major portion of such copper has been deducted from the smelters' reports and credited to the States of origin, it is probable that the statistics representing the Utah production still include some duplication.

(By L. H. Beason).—Bingham made greater strides than any other camp in the State. Its production was not materially increased, but a tremendous amount of development was done preparatory to ore extraction upon a much larger scale than heretofore. This was particularly the case with the Boston Consolidated, Utah Copper, Utah Apex and Ohio companies. The first has now under construction at Garfield, 13 miles west of Salt Lake City, a concentrating mill, which is planned to treat 3000 tons daily of the low-grade monzonite ore. This plant will probably be ready for operation about June, 1907. The Utah Copper Company is building a concentrating plant in the same vicinity. This plant is being put up in two units, each of 3000 tons capacity per day. This first unit will go into commission about March, 1907, if transportation facilities are provided by that time, and the second in the autumn of 1907. Both companies are using steam shovels to strip the overburden from the ore deposits, which average a little under 2 per cent. copper. The ores will be delivered to the mills by the Rio Grande Western Railway, which is building a second track into Bingham.

Control of the Bingham Consolidated and of the Ohio Copper Company was purchased by F. A. Heinze. Plans are now being made for the construction of a large concentrating mill for the Ohio mine, the initial capacity of which will be at least 2000 tons. The Utah Development Company, owner of the New Red Wing and Butler Liberal mines, developed a large, low-grade deposit, which makes a concentrated product worth \$10 to \$12 per ton.

The principal mines now producing in Bingham are: The Utah Consolidated, Boston Consolidated, Utah Copper, Utah Apex, Phœnix, New England, Fortuna, Ohio, Utah Development and Montezuma; the Dalton & Lark and Commercial mines of the Bingham Consolidated; the Old

Jordan, Galena and other properties of the United States Smelting, Refining and Mining Company; and the Bingham-New Haven. It is estimated that the present daily output of the camp is between 3000 and 4000 tons; but in 1908 it will probably be from 15,000 to 20,000 tons.

Next to Bingham, the Tintic district produced the largest tonnage of ore. Tintic has the deepest mines in the State. The leasing system, inaugurated in the Bullion Beck & Champion mine a little over a year ago, has been conducted successfully in other mines and has contributed largely to the increased activity of the camp.

In Beaver county, sensational developments were made in the Cactus mine of the Newhouse Mines and Smelters Corporation, which is now looked upon as being one of the great copper mines in Utah. The company is doubling the capacity of its mill, which has been treating from 600 to 800 tons of ore daily for some time. The Cactus ore is concentrated 8:1 or 10:1, and of late the concentrate has been worth about $60 per ton. The Newhouse company is using steam shovels for mining, and was the first to introduce them in this portion of the West. The Frisco Contact, at Frisco, developed a body of low-grade copper ore.

In the Sierra Madre mining district, in Box Elder county, recent developments indicate some good copper mines. The Eldorado, Napoleon and Santa Maria are the principal prospects.

The smelting situation in Salt Lake valley was unsettled in 1906, and still is, owing to the outcome of the "smelter smoke" litigation instituted by residents in the surrounding agricultural districts. The farmers won the case in a decision by Judge Marshall, followed by an injunction, which, if it becomes operative, will end sulphide smelting at Murray and Bingham Junction. The case has been appealed to the United States Circuit Court of Appeals at Minneapolis, and, pending its decision, the companies will continue smelting as usual, under bonds of $100,000 each. It is evident that the smelters do not expect any relief from the higher tribunal, for already the Utah Consolidated Mining Company is preparing to build a new $1,000,000 plant elsewhere and to go into custom smelting. It is practically certain that the Bingham Consolidated and United States companies will move also. In his decision, Judge Marshall prohibits the smelters from treating any ore, or combination of ores, carrying more than 10 per cent. sulphur, and from casting off any arsenic in fumes. The Yampa smelter in lower Bingham, and the new Garfield plant of the American Smelters Securities Company were not affected by the injunction.

Vermont.—Vermont re-entered the list of copper producing States, the mine at Pike Hill having been a small producer. The attempt to treat the ore of the Elizabeth mine, at South Strafford, by magnetic separation did not prove a success, although a similar process is used at Pike Hill. The concentrate from the latter place is shipped to New York for smelting.

Virginia.—The Virginia portion of the Virgilina copper belt was inactive, as regards copper, during 1906. In the Blue Ridge copper district of northern Virginia some development work was continued. About six miles south of Front Royal, in Harmony Hollow, Warren county, a 30-ft. shaft was sunk on the Beatty property. On the southwest end of the "Gossan Lead" at the Great Outburst, near Chestnut Yard, in Grayson county, the Pulaski Mining Company continued to mine the copper-lean pyrrhotite for trial tests in acid and ironmaking, with occasional shipments made of the black secondary copper ore concentrated beneath the gossan.

Wyoming.—The production of copper in this State showed a heavy decrease in 1906 owing to the burning, early in the year, of the smelting works of the Penn-Wyoming Copper Company, which was the only producer of blister copper in the State. The works have been rebuilt and will resume operations early in 1907. The small output of copper in Wyoming in 1906 was made by a few mines which shipped their ore to smelters in Colorado and Utah.

Copper in Foreign Countries.

The statistics of production in foreign countries will be found in the accompanying table, wherein the figures are as reported by Henry R. Merton & Co., of London, except in the cases of the United States, Canada, Mexico and Japan, for which we substitute the statistics collected by

TABLE OF WORLD'S COPPER PRODUCTION. (*a*)
(In metric tons.)

Countries.	1899.	1900.	1901.	1902.	1903.	1904.	1905.	1906
Argentina	66	76	793	244	137	157	157	107
Australasia	21,082	23,368	31,371	29,098	29,464	34,706	34,483	36,830
Austria-Hungary	1,529	1,377	1,356	1,626	1,407	1,473	1,346	1,458
Bolivia	2,540	2,134	2,032	2,032	2,032	2,032	2,032	2,540
Canada	6,838	8,595	18,575	17,765	19,360	19,490	21,588	19,106
Cape of Good Hope: Cape Co.	4,206	4,491	4,064	2,794	4,704	5,563	5,105	4,003
Cape of Good Hope: Namaqua	2,388	2,337	2,439	1,727	610	2,337	2,337	2,642
Chile	25,400	26,016	31,299	29,373	31,424	30,592	29,632	26,157
Germany—Total	23,836	20,635	22,069	21,951	31,214	30,262	22,492	20,665
(Mansfeld)	(21,118)	(18,684)	(19,082)	(19,050)	(19,810)	(19,578)	(19,878)	(18,085)
Italy	3,012	2,797	3,048	3,424	3,150	3,388	2,997	2,911
Japan	28,763	28,285	27,916	30,251	3,861	35,408	36,485	40,528
Mexico—Total	19,310	22,473	33,943	36,357	46,040	51,760	70,010	62,690
(Boleo)	(10,386)	(11,297)	(10,956)	(10,958)	(10,480)	(11,120)	(10,341)	(11 002)
Newfoundland	3.003	2,929	2,800	2,906	2,753	2,235	2,316	2,332
Norway	3,668	3,998	3,429	4,638	6,010	5,502	6,406	6,218
Peru	5,248	8,353	9,673	7,701	7,925	6,863	8,763	8,641
Russia	7,325	8,128	8,129	8,814	10,485	10,871	8,839	10,658
Spain-Portugal—Total	55,088	53,718	54,482	50,587	50,536	47,788	45,527	50,109
Rio Tinto	34,920	36,304	35,916	35,032	36,382	34,016	32,796	34,644
Tharsis	9,599	8,092	7,546	6,817	6,421	5,710	4,415	4,816
Mason & Barry	3,658	3,515	3,789	3,383	2,469	2,997	2,764	2,504
Sevilla	1,219	1,483	1,313	1,570	1,123	1,351	1,300	2,073
Sweden	528	457	457	462	462	542	559	508
Turkey	....	2,341	1,665	1,118	1,422	965	711	432
United Kingdom	647	777	610	488	544	501	726	508
United States	263,685	274,933	270,998	288,833	316,239	370,892	397.003	416,226
Totals	476,194	496,819	532,148	542.209	602,832	663,327	699,514	715.268

(*a*) The figures in this table are taken from the annual metal circular of Henry R. Merton & Co., except where returns have been received by *The Mineral Industry* direct from official sources.

ourselves. Some explanation of the changes in the statistics of the various countries will be found in the following reviews, which are presented alphabetically.

Argentina.—Operations were actively prosecuted at the Famatina mines, where ore of high grade in silver and copper was produced. The company is to erect a smeltery at Chilecito.

Australia.—There was an important increase in the copper production of this country, which was due largely to the appearance of new producers. Among these, the Mount Morgan mine opened its career as a producer of copper. This copper, as does also considerable other Australian copper, comes to the United States for refining, but the company is now planning to unite with other interests in the erection of an electrolytic refinery in Australia, which will probably be begun in 1907. The Mt. Morgan production at present is at the rate of about 4000 tons per annum.

Another new producer in Queensland was the Mount Molloy mine, which is expected to make an output of 2000 tons of blister copper during 1907, if the necessary additions are made to the reduction plant. The Chillagoe company, in Queensland, produced about 4000 tons of ore per month, yielding about 160 tons of copper and 20,000 oz. of silver. Its Morrison mine is opened to depth of 324 ft.

The Great Cobar copper mine, New South Wales, was floated in London and transferred to the new company on Aug. 21, the price being £800,000. It is understood that operations are to be prosecuted with greater energy, and the output of 4000 tons of copper per annum increased to 13,000 tons.

The newly discovered copper deposits at Mount Carmel, in the Heathcote district, Victoria, are receiving much local attention. They are believed to be extensive, permanent and rich, the surface indications being traceable for miles. The country has been pegged out for about seven miles, but so far only a moderate amount of development work has been done.

After nearly 40 years, mining has been resumed at Wallaroo, though only on a small scale. The first discovery of copper on the Yorke Peninsula came from the beach south of where the new jetty now stands. Attention was diverted from here by the discoveries at Moonta and Wallaroo mines, but some attempts were made subsequently to locate a lode. Unfortunately the surface prospects all lie below high-water mark, and efforts to sink shafts were unsuccessful. Last year the Wallaroo & Moonta company took an option over a mineral claim held by a local syndicate, and has since made a thorough search for payable ore. Finding no indications inland, toward the end of the year it sunk a shaft just above high-water mark, and is now driving from about 80 ft. out toward the sea.

The influence of high copper prices is best seen at the smelting works,

where, in addition to local ore, products from Queensland, Tasmania, and Western Australia are being handled. From the fortnightly reports published in the *Register* the accompanying figures are compiled:

WALLAROO & MOONTA STATISTICS.

Ore Received.	1906.	1905.	1904.	Ore Received.	1906.	1905.	1904.
From Wallaroo mines, tons	43,241	35,189	21,766	Copper placed in store, tons	7,556	6,501	5,835
From Moonta mines, tons	10,360	8,161	10,686	Silver to mint, oz	3,614	4,781	7,147
Precipitate, tons	957	930	949	Gold to mint, oz	1,643	1,646	1,260
Outside ores and matte, tons	3,520	5,151	5,381	Sulphuric acid delivered, tons	5,112	5,312	3,433
Ore smelted, tons	58,068	49,961	38,995	Bluestone made, tons.	328	340	181

Several improvements in smelting methods were instituted; the blast furnace was run almost continuously, and a series of sintering converters installed. To facilitate the handling of slag from the former a granulating plant was erected, and latterly the product has been forwarded in large quantities to the mines, where it is used for filling depleted stopes.

The well-established lodes of the Yorke are maintaining their productiveness, and vigorous search is being made for new orebodies. Although it may be too much to say that there is as much copper in the district as has already been taken out of it, yet there is no reason to believe that all the payable lodes have been discovered. Wages are high. The Wallaroo & Moonta company has paid 25 per cent. bonus on standard prices.[1]

(By F. S. Mance).—In Tasmania the Mount Lyell Company during the year ended Sept. 30, 1906, produced blister copper containing 9009 tons of copper, 703,945 oz. silver, and 23,088 oz. gold, the profit amounting to £495,000. The ore reserves are of a magnitude which assures a continuance of successful operations under present conditions for a long period.

In New South Wales the results achieved in copper mining were of a satisfactory nature. The Great Cobar mine was taken over by a new company, and arrangements are now in progress which, in the near future, will permit of the present output of 600 tons of ore per diem being increased to 1500 tons, a quantity which the orebodies seem well capable of furnishing. The other established mines in the Cobar district made substantial outputs, while several new and promising lodes were opened up during the year. The output of the Queen Bee mine was considerably augmented and the mines held by the Nymagee Copper Mining Syndicate were actively worked. The Girilambone and Mount Hope mines were re-opened during 1906 and copper to the value of £11,966 and £14,672 was obtained from them respectively. Discoveries made in the Budgery mine, situated about three miles from Hermidale, created much interest. Other

[1]London *Min. Journ.*, March 9, 1907.

copper mines which came prominently into notice during 1906 are the Cadia and the Cangai mines which are located respectively in the Orange and Grafton districts. Nothing has yet been done in the way of providing for the treatment of the large bodies of auriferous copper ore on the Cobar field, and the combining of interests to this end is much to be desired. The Lloyd Copper Company, Burraga, was fortunate in relocating the lode which had been faulted, and has thus received a fresh and vigorous lease of life.

The Wallaroo & Moonta Mining and Smelting Company, in South Australia, was able to resume the payment of regular dividends, despite the heavy outlay incurred during the last two years in repairing the damage occasioned by the fire in the underground workings of the Wallaroo mine, and also in the installation of new plant. The production in 1906, including that from purchased ores, was about 7560 tons of fine copper. It is expected that the beginning of the year 1907 will see the work of restoration completed, when the company will then be in a position to considerably augment its production. The Taylor shaft is now 2200 ft. deep, and workings at that level have proved the lode to be 9 ft. in width, with an estimated yield of 10 tons of 10 per cent. ore per lineal fathom. At the Moonta mines, the new workings at Truers, and the old North Yelta property, are being vigorously developed, and there is every prospect of large supplies of ore being made available.

Of all the states, Queensland made the greatest advance as a producer of copper. The Chillagoe company was benefited greatly by the rise in price, being thereby enabled to earn substantial profits. Some promising developments in the mines of this company are reported, the best being that of the Queenslander lode at Morrison's. The O. K., Mount Perry and Mount Molloy mines have come prominently into notice as large copper producers. Work on the Cloncurry field was active, operations being stimulated by the proposal to connect the field with the existing railway system. This field may be expected to supply an extensive output of copper in the near future. The Mount Morgan company was engaged extensively in the treatment of the cupriferous orebodies, which have previously been mentioned. The output in 1906 was 2281 tons against 223 in 1905. The output in 1907 should be well over 4000 tons.

Wallaroo & Moonta Mines.[1] (By L. Hancock).—Mining for copper on Yorke Peninsula dates back to just before 1860. The Wallaroo mines are about six miles east, and the Moonta mines about 12 miles south of Port Wallaroo. Both properties are connected by Government railway with the Port, where the company's smelting works are located.

At the Wallaroo mines the lodes traverse a metamorphic schistose rock,

[1]*Eng. and Min. Journ.*, Feb. 10, 1906.

with a direction approximately east and west. The various ore-bearing strata at the Moonta mines are composed chiefly of porphyritic rock, and the lodes (speaking generally) bear north and south.

The dressed ore from Wallaroo has, throughout recent times, averaged about 11 per cent., that from Moonta about 20 per cent. of copper, excepting that in later years it has been 2 or 3 per cent. lower. For a long time the vein stuff as raised to surface at both properties has contained on the average from 3 to 4 per cent. copper. Recently immense accumulations of tailings and slimes from mechanical dressing have been subjected to hydro-metallurgical treatment, affording good profits.

For about 30 years the Wallaroo mines and the smelting works were one concern, while the Moonta mines were worked independently, selling their ore to the Wallaroo company. The latter, being a private company, published no records; but from what information is available it would appear that during its separate existence £2,229,096 of copper was extracted, besides nearly £339,000 produced from purchased ores. Apparently about £430,254 was distributed in dividends.

These copper values do not include those from the Moonta company, whose published statements show that £5,396,146 worth of copper was raised, of which £1,168,000 was disbursed among shareholders. The Moonta mines have the distinction of being the first mining company in Australia to pay in dividends a total of £1,000,000, notwithstanding that the rich gold reefs of Victoria had been operated for years before the Moonta was discovered.

Since 1889 when the Wallaroo mines, with their smelting works, and Moonta mines became one concern, copper worth about £4,281,342 has been produced, of which £224,000 has been distributed in dividends. In nearly 45 years these mines have raised and extracted about £12,245,554 worth of copper, and paid £1,822,254 as dividends.

At the smelting works, in addition to the high-class copper of the well-known "Wallaroo" brand, sulphuric acid and bluestone are manufactured.

Early in 1904 a fire in the main shaft at Wallaroo mines completely destroyed the upper portion, and rendered useless the pumping appliances to a depth of 2000 ft. There was grave danger that the deep working would be lost through the flux of water, but by strenuous exertions, and despite great difficulties, temporary pumping appliances were installed, driven by compressed air. These are now being supervised by an up-to-date electric scheme; also an extensive central steam power and compressed-air plant is nearing completion. A new shaft has been sunk 840 ft., to join the old one where the underlay passed to the vertical.

When the new surface and underground plant is completed Wallaroo mines should be re-established on modern and profitable lines, and as the deepest levels (1800 ft. to 2000 ft.) show no diminution in value as com-

pared with those of less depth, the prospect of continued operations for years ahead is promising.

As further illustrating the operations at the mines, some additional statistics are appended, which, although not absolutely certified, are believed to be practically correct: Total dressed ore produced from the combined mines, 1860 to 1904, inclusive, 1,540,180 tons; average copper contents, 15½ per cent.; representing in fine copper, 235,630 tons; average yearly production of ore, 34,226 tons; maximum output in one year, 40,222 tons; total expenditure, £10,423,300; average annual expenditure, £231,629; average cost of each ton of dressed ore, £6, 15s. 4d.; number of employees, June, 1905, 2260; maximum number of employees, 2600; total amount of dividends, £1,822,254.

Austria.—The Mitterberg mines in the Tyrol were taken over by a British company. They are situated about 15 miles from Bischofshofen, on the railway from Salzburg to Innsbruck. The whole district is extensively mineralized. The copper ore is chalcopyrite, interspersed to a very small extent with pyrite, nickel and arsenical pyrite; there are only traces of gold and silver. The ore filling occurs chiefly in compact stringers in the lodes, varying from 1 to 30 in. wide; but the remainder of the lode contains sufficient ore to make it worth extracting. The picked and concentrated ore is sent to the smelter at Ausserfelden, which is about 1½ mile from Bischofshofen. The new company intends to build water-jacket furnaces and converters. The ore averages about 3 per cent. copper. During 1905 about 23,000 tons of ore were extracted. From this, 7000 tons of picked ore and concentrates were obtained, which yielded 615 tons of copper. The ore ready for stoping amounts to nearly 450,000 tons. An equal quantity of probable ore, partly developed, can be reckoned on with safety.[1]

A deposit of high-grade copper ore is said to have been discovered at Gemine in the Bukowina.

Bolivia.—The chief producing center is Coro Coro. Several companies, especially Berthin Frères, Careras Hermanos, and the Société Chilienne, mine the ore and concentrate it mechanically. Because of the railway line to be constructed from Arica to La Paz via Coro Coro, this mining center will further develop. The hills which bound the Alti-Planicie, south of Lake Titicaca, also contain more or less rich copper deposits, metallurgical treatment of which would be remunerative were the means of transport less rudimentary.

Canada.—The mines of the Boundary district, British Columbia, suffered from substantially the same difficulties as prevailed in the United States, namely, shortage of labor and fuel. In the autumn, a strike in the Crows' Nest coalfield temporarily cut off that supply of coke and com-

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[1]Edward Walker, *Eng. and Min. Journ.*, March 17, 1906.

pelled the smelters to obtain fuel, at greatly increased cost, from the United States. These troubles prevented the copper production of Canada from showing the expected increase. The furnaces of the Granby company, the largest producer, have now been remodeled, and if there be no further difficulties of the same kind as those which appeared in 1906, the output of the Boundary district will show a large increase in 1907.

The decrease in the Canadian production in 1906 as shown by our statistics is due partly to a change in the method of compilation. The figures for 1905 are as reported by the Canadian Geological Survey, which are based on the copper content of ore produced. Those for 1906 are based on our own reports received directly from the producers, and represent the copper content of blister copper produced. The statistics of copper production in Canada have become somewhat complicated, inasmuch as silicious ore is now shipped from Vancouver Island to Alaska, where it is smelted with basic ore of Alaskan origin, and the matte is shipped back to Vancouver Island, where it is converted into blister copper, which is exported to the United States for refining. Moreover, the smelters of the Boundary district receive considerable silicious ore from the United States, which they use for converter lining, and send all of their blister copper to the United States for refining.

In the year ending June 30, 1906, the Granby company smelted 796,188 dry tons of its own ore and 36,158 tons of foreign ore, a total of 832,346 tons. The yield was 19,939,004 ℔. fine copper, 316,947 oz. silver and 50,-020 oz. gold; the average prices realized being 17.78c. for copper and 64.68c. for silver. The average cost per ton, including all expenses, was \$3.2988; the net cost of copper, after deducting value of gold and silver, was 8.35c. per pound. "In view of the high prices for copper, it has been deemed wise to mine large quantities of ore carrying a smaller percentage of copper than the average run of the mines. Active development work has been going on continually, and diamond drills have opened up large orebodies in the Victoria and Aetna mines, where a new shaft is now being sunk and the necessary improvements installed for crushing and shipping this output. These, and other developments have largely increased the tonnage of ore in sight over that extracted in the year."

In June, 1906, the Granby smelter had three furnaces, 48x212 in. at the tuyeres, and five furnaces 44x167 in. It was then the intention of the company to remodel the five small furnaces and make them the same size as the larger ones, which will make the smelting capacity about 3500 tons per day; this work is now going on. A third converter stand has recently been added to the converter department, which makes three stands in all, 5x8 ft., capable of producing 30,000,000 ℔. of copper per year.[1]

[1] *Eng. and Min. Journ.*, Sept. 8, 1906.

In the year ending Apr. 30, 1906, the Tyee Copper Company, of Ladysmith, Vancouver Island, smelted 27,976 tons of ore, of which 23,979 were from the company's mine, and 6580 tons were custom ore. The Tyee ore averaged 4.85 per cent. copper, 2.80 oz. silver, 0.145 oz. gold, 12.16 per cent. iron, 7.34 per cent. zinc, 13.82 per cent. silica, and 37.57 per cent. barium sulphate. The raw ore fines and concentrate were bricked and roasted with satisfactory results, both as regards the thoroughness of the roasting and the condition of the product for smelting. The amount of ore bricked was 3107 tons, being 30.18 per cent. of the ore roasted. The smelting mixture was as follows: Tyee raw ore, 42.50 per cent.; Tyee roasted ore, 21.60 per cent.; Tyee roasted brick, 9.34 per cent.; custom ore, 12.80 per cent.; flux, 4.35 per cent.; flue dust, 2.98 per cent.; slag and barrings, 1.67 per cent.; and low-grade matte, 4.76 per cent. The ratio of coke to ore was 1: 9.93 and coke to charge, 1: 11.52. The concentration was 9.04 tons of ore to 1 ton of matte. The average contents of the slag were: Copper, 0.375 per cent.; silver, 0.13 oz.; iron oxide, 19.24 per cent.; silica, 32.34 per cent.; barium oxide, 27.89 per cent.; calcium oxide, 4.9 per cent.; zinc oxide, 5.89 per cent.; alumina, 9.78 per cent. The flue dust averaged: 3.77 per cent. copper, 2.74 oz. silver, 0.14 oz. gold, 11.77 per cent. iron, 18.56 per cent. silica, 28.30 per cent. barium sulphate, 1.5 per cent. lime, 6.79 per cent. zinc, 5.49 per cent. alumina, and 5.82 per cent. sulphur as sulphides. The smelter produced 3171 tons of copper matte during the year, which figured out to $17.67 per ton of ore smelted.

An interesting copper deposit was discovered near Dean Lake station on the Canadian Pacific, in the township of Thompson, on what is now known as the Leizert mine. The ore occurs at the contact between a trap dike, bearing inclusions of granite and quartzite, rounded and angular, and clay slate. Both the slate and the trap are impregnated, for a width at present unknown, with chalcoprite, which is accompanied by calcite, and carries a little silver and gold. The vein is shown up by cross-cuts and pits and can be followed for more than 1000 ft. The property is owned by the Northern Ontario Copper Company, Ltd., Sault Ste. Marie, Ont.[1]

According to our own statistics, the production of copper in Canada in 1906 was 42,121,000 lb., of which the smelters of the Boundary district, British Columbia, produced 24,800,436 lb. (deducting the copper content of ore which they received from the United States).

Chile.—A noteworthy feature of 1906 was the increased attention directed toward the copper resources of Chile. Several new British and American companies have become interested there. Doubtless this will soon result in a material increase of the Chilean production, although

[1] *Eng. and Min. Journ.*, June 30, 1906.

the statistics for 1906 do not show any gain over 1905. Among the new enterprises is the Braden Copper Company, in which interests affiliated with the American Smelting and Refining Company are engaged.

At Antofogasta, the Guanaco, once a gold mine, is now being worked for copper, which appeared at depth of 328 ft.; the bottom workings (820 ft.) are still in iron and copper pyrites. Several deep copper mines are being worked in Chile. In the department of Chañaral the Fronton mine has attained a depth of 1837 ft., the bottom workings still being in good ore. One vein of pyrites, about 6½ ft. in thickness, yields from 7 to 8 per cent. copper; while a second vein, about 2½ ft. thick, yields from 10 to 11 per cent. The Descubridora de Carrizalillo mine, in the same region, has a depth of 2132 ft. In Los Pazos district two large veins of oxidized ores, varying from 26 to 65 ft. in thickness, are being worked in shallow depths. In the department of Copiapó, La Dulcinea, worked by the Copiapó Mining Company, is the deepest mine and the largest producer in the whole of Chile. The vertical depth is nearly 2624 ft., while longitudinally it has been worked for 1640 ft. Copper pyrites was met with at a depth of 656 ft. In the department of Freirina, the Socabón mine has a depth of 1371 ft. The vein, from 5 to 8 ft. in thickness, occurs in a granite rock, the ore (chiefly copper pyrites) averaging about 18 per cent. of metal. In La Serena department the veins of La Higuera district occur in dioritic rock, the gangue consisting of calcite and asbestos, termed *piedrapalo* by the miners. The Brillador and San Antonio mines, to the south of the same department, are about 1800 ft. deep. The famous Tamaya district, in the department of Ovalle, which formerly was the largest copper producer of Chile, is at present worked only on a small scale. The Rosario mine has a depth of about 1900 ft.

Deposits of ore suited to leaching are widely distributed over Chile. They carry from 2 to 6 per cent. copper, and occur as oxide outcroppings. They stand up in relief and on this account, as well as on account of their green and blue color, are easily found. These deposits as a general thing stretch out horizontally more than they do vertically, and are very deceptive, often pinching out when least expected. The deposits are seldom of even width; they are sometimes from 2 to 8 m. broad and then pinch, their depth rarely exceeds 15 m., and if they do go farther down, their character usually changes to sulphide. A large average sample of ore from the neighborhood of Andacollo had the following composition: Cu, 3.58; $Si\,O_2$, 77.50; Fe_2O_3, 10.25; Al_2O_3, 2.85; CaO, 2.78; MgO, 0.32; SO_3, as gypsum, 0.74; CO_2, 1.19 per cent. From an ore of this character, 30 kg. of copper can be extracted per ton of ore by the use of 90 kg. of sulphuric acid of 65 to 66 deg. B.

Situated about 75 miles south of Valparaiso, and nearly 40 miles east of Rancagua, is the property of the Braden Copper Company, the head-

quarters of which are at 71 Broadway, New York. The mine is 8000 ft. above the sea level. The mill site is 1500 ft. lower, at a place called La Junta. The company has erected there a 350-ton experimental mill, which will be replaced by a modern steel structure, capable of treating from 2000 to 3000 tons per day. A railroad is to be built from Rancagua to La Junta, a distance of 40 miles, and upon its completion work will begin upon the new mill and the proposed smelter.

The copper exports in 1905 amounted to 29,631 metric tons of fine copper, a gain of 960 tons over 1904. The total is made up of 23,893 tons of bar copper; 1568 tons of fine copper contained in matte; and ore containing 4170 tons of fine copper. The bar copper exported was consigned as follows: 17,956 metric tons to the United Kingdom, 4710 tons to France, 191 tons to Germany, 314 tons to the United States, and the remainder to Belgium and Peru. The average cost of freight and insurance on these shipments was about $26 per ton to Europe or New York.

The high price of copper has had much influence in directing the attention of capitalists to the country's well known copper fields, with the result that many old mines have been taken up and are being reworked, and several important smelting works have been established.

Throughout the mining districts there is a scarcity of miners, and as a consequence wages have risen and promise to advance further. The scarcity is due to the lazy habits of the men, and to the severe epidemics of smallpox which have proved fatal to a very large number of the working classes in the past few years. The labor problem is the chief obstacle in the way of Chile's progress.

The Government has done much to facilitate the development of mining regions and has voted large sums for the building of new railroads. On Feb. 15, 1906, a new line was authorized between Valparaiso and Santiago via Los Andes. The line is estimated to cost $75,000,000, and will be able to accommodate all the traffic of the Transandine line without any break of bulk between Argentina and Chile. It will also take over the traffic of the existing branch line from Cabildo to La Caldera. This branch line is destined at an early date to be connected with the existing lines in the northern districts.

The Chañaral railroads, which are the property of the Government and serve Las Animas, Salado, Pueblo Hundido and Inca mining districts, are being extended to Los Pozos.

The most noteworthy developments in the chief copper-mining provinces, going from the south to north, are as follows[1]:

Santiago.—In addition to the large mines and works of the Volcan Copper Company, a French company is operating a group of mines and a smelter which produces about 2400 tons of copper per annum. Improve-

[1] *Eng. and Min. Journ.*, Nov. 24, 1906.

ments at the port of Valparaiso, in this province, at a cost of $20,000,000, are now progressing, and are expected to influence largely the development of mining as well as other industries.

Coquimbo.—This province has long been a copper producer. In 1905 about 3000 tons of copper were smelted at El Almendral works. The Central Chile Copper Company's smelter, at Panulcillo, is shipping from 4000 to 5000 tons of copper per annum.

Caldera.—A new copper smelter has been erected at Caldera, consisting of two water-jacket furnaces, each having a daily capacity of 150 tons, and two converter stands. The Tierra Amarilla smelter is working regularly, and a Chilean company proposes to erect another smelter at Puquios. Mining operations have been resumed in the old Carizal copper mines.

Chañaral.—At Guanaco the mines, which were formerly worked for gold, are now yielding both gold and copper. At Chañaral, the Inca mines are being operated, while French capitalists are developing a group of new copper mines and propose to erect a smelter.

Antofagasta.—The chief copper deposits in this province are at Chuquicamata. They were originally worked for a powdery ore called "atacamite," assaying beneath 2 and 3 per cent. copper; developments at depth have disclosed well formed copper lodes assaying 5 to 20 per cent. copper.

Tarapaca.—Recently some rich copper lodes were found at Collohuasi, and are being developed by English and Chilean capitalists. The mines are approached by the Onera and Antofagasta railroad, but they are situated at a high altitude, and mining is difficult. The climate of this region is cold. The thermometer in winter goes down to —24 deg. C. and in summer scarcely rises above 14 deg. C. In June, July, and August snow falls to a depth of from 3 to 6 ft.

Samuel Green visited the mines of the Collahuasi Syndicate in 1903, and reported as follows:

"Copper ore, in all forms known to mineralogists, is found in this bonanza associated with a little limonite and quartz. On the surface of all the veins carbonates and silicates are the prevailing forms of the ores; deeper down they change to red and black oxides and afterward to sulphides. In the Pergolesi lodes are occasionally found ores of as much as 72 per cent. copper. The vein actually worked is only about 68 ft. in length and 15½ ft. in width, producing 500 tons of 35 per cent. copper a month. There is also an appreciable amount of silver, gold, and cobalt."

According to the report published Feb. 28, 1906, for the six months ending Jan. 1 there were 51,311 tons of 27 per cent. ore in the reserves, 4110 tons of 22 per cent. at the grass, and 15,990 tons of 12 per cent. on the dumps, or a total of 71,411 tons of 23.35 per cent. ore.

Congo.—The explorations in this part of Africa which were referred to in The Mineral Industry, Vol. XIV, were carried further in 1906, with

highly important results. It is believed now that this region contains immense copper resources. They have been investigated by A. C. Beatty, of the Guggenheim Exploration Company, and that company has become largely interested in them. Early in 1907, Mr. Beatty, with a staff of engineers, returned to the Congo to inaugurate development work.

The Tanganyika Concessions, Ltd., was floated in London, to take over mineral concessions in Katanga, which are claimed to show ore containing 2,000,000 tons of copper. The company plans a smelter to turn out 60 tons of copper a day, with expectation of increase to 200 tons a day, or 145,000,000 lb. a year. It is declared that this will be produced cheaper than anywhere else in the world. It would appear that these statements should be taken with some grains of salt.

A new copper company has been formed in Brussels to mine deposits in Upper Katanga, a grant of which has been obtained from the Congo State. The capital is 10,000,000 francs, one-half of which has been subscribed by the English company, Tanganyika Concessions, Ltd., and the other by the Société Générale de Belgique.

According to the official newspaper of the Congo Free State, the *Bulletin Officiel de L'Etat Independent du Congo*, a royal decree, dated Nov. 9, 1906, transfers to the Ryan syndicate the right to prospect for metals over the whole of the unassigned public domain for six years. Whatever mines may be established by this exploration may be worked for 99 years, one out of every three becoming immediately the property of the State.

A company known as the International Congo Lumber and Mining Company has been organized to finance this concession with a capital of 3500 shares, half of which are taken by the American syndicate and the other half by Belgian capitalists. The American syndicate paid $150,000 down and is pledged to bring the total payment for its half of the concession up to $1,500,000. The grant covers an area of about 2,500,000 acres. Besides its mining rights, the company has obtained a grant of about 741,000 acres of forest land and the right to harvest a rubber-producing plant over an area of 2,470,000 acres. The syndicate has obtained additional rights to 12,000 acres on each bank of navigable streams and a 10-year option on 1,000,000 acres more.

The American directors of the company are Wm. H. Page, attorney for the Continental Rubber Company; A. Chester Beatty, of the Guggenheim staff of mining engineers, and J. G. Whitley, consul-general of the Congo Free State to the United States.

Various financial concerns are now exploring a cupriferous district in the French Congo, about 200 km. from the coast. J. Bel, an engineer, recently set out at the head of an exploring company organized by the Société Française pour l'Industrie et les Mines, of 45 Boulevard Haussmann, Paris.

Germany.—In this country, where there is only one important producer, viz., the Mansfeld company, the production continues about the same from year to year, and 1906 was no exception to the rule.

Italy.—According to B. Lotti,[1] northeastern Sicily is rich in iron, lead, zinc, and copper ores. The copper deposits of Pizzo della Croce yield ore containing 6.73 per cent. of copper.

Japan. (By M. Otagawa.)—The steady advance of the copper market has encouraged the Japanese copper mining industry, with the result of an increased production. The following figures show the production of copper in Japan during recent years: 1899, 23,890 long tons; 1900, 28,930; 1901, 27,000; 1902, 28,570; 1903, 32,600; 1904, 34,600; 1905, 36,600; and in 1906, 39,890. These figures are taken from official returns of the Department of Agriculture and Commerce and other reliable sources.

Copper mining in Japan is an ancient industry. Its origin is almost beyond the time of authentic history. As early as 708, copper coins were minted in Japan. The famous huge image of Roshana-Butsu was erected in the capital of Nara between 743 and 752. The larger mines, whose record is well known and preserved, have been steadily operated for three or four hundred years. There are a number of smaller mines, some of which date back for more than double that time. In the list of the world's copper producers, Japan stands fourth, its total being exceeded only by the United States, Mexico, and Spain-Portugal. The output supplies all home needs, having been sufficient even to meet the great demand for the metal for use in cartridges and other war material during the recent conflict with Russia, in addition to considerable exports, chiefly to China and other eastern countries. The earlier operations followed rather primitive lines, but the Japanese metallurgists succeeded in turning out copper of fine quality. The early Japanese metallurgists had a process similar to the modern bessemer process in principle, which enabled them to produce this superior grade of metal, which was known as "mabuki." The years following 1868 were a period of transition, during which the old methods of mining, smelting and refining were replaced by more modern systems, and by improved machinery. The work was at first under the superintendence of American, German and English engineers; but at the present time the mines and smelting works are entirely under the direction of Japanese engineers. This period also saw an important change in the economical position of the industry. The larger mines passed from the old individual or government ownership into the hands of large companies; and these corporations also absorbed a number of the smaller mines. The industry is now in strong hands. The mines of importance are well supplied with the best modern machinery—hoists, pumps, crushing mills, concentrators, etc. Electric transmission of power is largely used, includ-

[1] *Zeit. f. angew. Chem.*, Jan. 18, 1907.

ing electric locomotives for haulage in the mines. The smelting furnaces are of approved types, and there are several electrolytic refineries. Japanese practice is now entirely up-to-date. In some respects, hand labor is used more than in American mines; but this is to be expected in a country where the supply of labor is large, and wages are low.

The best known copper mines in Japan are the Ashio and the Besshi; the former producing about 7000 tons of refined copper, and the latter about 6000 tons yearly. There are nearly 300 places where copper ore is or has been obtained, but only about 50 mines are actually producing commercial quantities at the present time.

The Ashio mine, in the province of Shimotsuke, was first opened about 1608 and 100 years later was a large producer. For many years it was operated by the Government, either directly, or under lease. In 1877 it passed into the possession of the late Mr. Ichibei Furukawa, who established the Furukawa mining firm, which existed until 1905, when it was reorganized as the Furukawa Mining Company. As early as three centuries ago, Ashio copper was exported to Holland. As a further matter of interest, the tiles on the roofs of the famous Nikko temples and the Tokio (Yedo) Castle were made of this copper.

In addition to the Ashio, this company now owns seven other copper mines, viz., the Ani, the Kusakura, the Nagamatsu-Otori, the Mizusawa and the Furokura, and the Kune and Innai silver mines. It operates four collieries, at Katsuno, Shiogashira, Shakanoo and Shimoyamada, besides copper works (in Nikko and Tokio), the Shinonome smelting works and a coke manufactory (in Tokio), which furnish all the coal and coke used by the company, with a large surplus for sale.

The ore of the Ashio is almost entirely sulphide. The deepest workings are about 1250 ft. The average smelter return of the ore mined is about 4 per cent. copper. A small quantity of high grade ore, hand-picked, is sent directly to the smelter. Most of it is concentrated, however, to a product averaging about 15 per cent. copper. There are at the mine three concentrating mills, in Honzan, Tsudo and Kodaki. The smelting works have a capacity of 700 tons of ore daily. The bessemer copper is refined in the electrolytic plant.

About a year ago it erected the Nikko electrolytic refinery, and the Daiyagawa power plant of 5000 h.p., which is used both for the refinery and for the mines. The Furukawa company markets its copper in several different forms, viz., bessemer bars, direct from the Ashio converters; ingot copper, made at the Honjyo works from crude copper obtained from some of its mines, which has a high ductility; "tiles," made at the Shinonome works, mostly from the Ani copper; and electrolytic copper, in ingots, wirebars and cathodes, from the Honjyo works. The company also turns out wire, sheets and other finished products.

The Besshi mine was opened in 1690, and has always been a large producer. It has always been in the possession of the Sumitomo family. It is situated in the Ehime district of Iyo province in the island of Shikoku. Like the Ashio, it has been provided with modern machinery, and work is conducted on approved lines. This firm has recently built a new and modern smelter in Shisakajima, a small island in midsea, where the smoke problem will cause no trouble. The ore is chalcopyrite and pyrite; the average yield is about 6 per cent. copper. The new smelter consists of stalls, blast furnaces and both large and small reverberatory furnaces of such capacity that the production of the ancient and famous mine can be increased largely.

The chief feature in the recent mining history of Japan is the remarkable development of the Kosaka mine, in the northern part of the main island, i. e., Honshiu. This property, which belongs to Fujita & Co., was originally worked as a silver mine, but in passing below the line of oxidation, the ore changed into an enormous mass of mixed sulphides.

The Kosaka ore deposit appears to be a blanket vein, dipping at an angle of about 45 deg. The hanging wall is an andesite tuff; the foot wall is diorite. The deposit is about 400 ft. in thickness, measuring at right angles to the dip of the vein. Immediately under the hanging wall is a large zone of mixed sulphides, so closely mingled that mechanical separation is impossible. This ore averages 40 to 45 per cent. barium sulphate, 10 per cent. zinc, 10 per cent. lead, 2 per cent. copper, 0.02 per cent. silver, and a trace of gold. Immediately under it is a zone of iron pyrites containing 1 to 2 per cent. copper, and below the latter occurs silicious ore containing from 1 to 1.5 per cent. copper. Many metallurgists had tried to smelt this exceptionally refractory ore without success, and it was not until recently that a satisfactory process was developed. The metallurgists of the company deserve high praise for their achievement in this respect, which has led to the present great value of this property.

The three kinds of ore are mined, and mixed in such proportion as to produce a self-fluxing product. This is smelted semi-pyritically, about 2 per cent. of anthracite being employed as fuel. Smelting is done in six furnaces, each 3.5x25 ft. at the tuyeres. These furnaces reduce the charge at the rate of about 3 tons per square foot of hearth area per 24 hours. The slag is essentially a silicate of iron and baryta. A new furnace, 3.5x60 ft., is now being erected. The product of the blast furnaces is a lead-copper matte containing about 30 per cent. copper. This is crushed, roasted, and smelted for blister copper in reverberatory furnaces. The blister copper is refined electrolytically, and the capacity of the refinery is now being increased so as to take care of the entire product of the works.

The Kosaka mine has even displaced the famous Ashio mine from the premier position among Japanese copper producers. Its output in 1905 was approximately 7200 tons of copper, against 7000 tons from the Ashio mine, and 6000 tons from the Besshi mine. The success of the Kosaka mine has directed attention to other deposits of similar character. Among those which have recently been opened is the Kano mine, near Aidzsu, in the northern part of Japan.

Thus, the success of metallurgy in the Kosaka mine added much to the copper production of Japan. It is expected that the Furukawa Mining Company will shortly build a smelter for its Kune mine which will at least produce as much as the Ashio—making a great increase of output of copper of Japan.

Mexico.—The copper production of Mexico showed a material decrease in 1906, but it is manifest that this Republic is bound to increase more and more in importance as a producer of copper. However, even in Mexico, shortage of labor is the general complaint. For this reason, and also because of the riots at Cananea late in the summer, and other troubles of the Greene company, the production of Mexico in 1906 was not so large as it ought to have been. According to our statistics, the production of Mexico in 1906 was 135,800,000 lb., against 144,350,962 lb. in 1905.

The leading feature of 1906 was the development of a great new mine (the American) by the Cananea Central company, and at the very end of the year the amalgamation of that company with the Greene Consolidated, under the name of the Greene-Cananea Copper Company, in which Messrs. Cole and Ryan, who are affiliated with the Amalgamated Copper Company, are the controlling interest. Colonel Greene is still identified with the company, but the management of the mines which he developed has passed out of his hands. The ore of the Cananea Central properties has been smelted at the works of the Greene Consolidated, while a good deal of the ore of the Greene company has been shipped to the United States for smelting. The production of the Moctezuma Copper Company also was smelted in the United States.

The American mine had been attracting attention for some time, but more particularly since early in May, when it began to show large bodies of 5 per cent. copper ore. The main shaft of the American is only 200 ft. from the Cobre Grande shaft of the Greene. The property embraces 126 pertinencias (300 acres). In May the orebodies had been proved up in three different shafts, several thousand feet apart, and the ore was being blocked out with the intention of putting in a large smelting plant at Cananea. Arrangements were made in the meantime for treating at the Greene plant such ores as were taken out in the development work. Since that time the ore has been opened up in three additional shafts, making six in all, and runs in places high in copper.

Elsewhere in Sonora the production did not show any great increase. The Bufa Mining and Smelting Company made regular shipments of matte. The Cieneguita Copper Company at Chipiona (about 40 miles southeast of Sahuaripa), has a large plant, and the company owns a number of properties situated in the surrounding district. Among these is the Ostimuri group, which has been worked off and on for nearly 200 years. The plant consists of four wood-burning reverberatory smelting furnaces, two reverberatory roasters and some 50 roasting stalls. Half a million dollars (gold) is said to have been spent on the property. Over 7000 ft. of drifts are said to have been run and a large body of ore opened up. This mine is the best developed in the district and should soon be an active producer, but hitherto operations have been unsatisfactory because of managerial troubles.

The Sonora-Bonanza Mining Company, contemplates the erection at Imuris of a smelter and concentrating mill of 500 tons capacity. The Anaconda-Sonora Copper Company, operating in the Sahuaripa district, has opened ore assaying 6 to 10 per cent. copper and has commenced the erection of a 100-ton smelting plant on the Yaqui river, nine miles from the mines. The West Coast Mining and Smelting Company has been organized to operate along the west coast of Sonora, and in the country opened up by the Southern Pacific now building between Guaymas and Guadalajara, both in Sonora and Sinaloa.

(By C. A. Bohn.)—The great increase in the production of copper, which has put Mexico among the chief producers of this metal, is due especially to the mines in Sonora, and particularly in the district of Cananea. The greatest at this camp is the Greene Consolidated Copper Company. In eight years this company has become one of the most important producers of the world, notwithstanding the fact that three-fourths of its ore is of second grade and has to be concentrated. Development has been carried on beyond the ability of the surface equipment to handle the tonnage, but this is being rapidly brought up to meet the requirements; the caving system is being introduced in the mines; an elaborate system of underground and surface roads, with electric traction, between the mines and mills has been installed; the concentrators have been increased by 1000 tons, and brought up to a daily capacity of 3500 tons; an extensive system of Robins belt conveyors, for bedding the ores and feeding the furnaces from the beds, is about ready to be put into operation; the fines and flue dust are being sintered by a patent process to relieve the work of the blast furnaces, and the eight blast furnaces now in commission are being replaced by larger furnaces, 56x180 in., at the tuyeres, and with automatic feed; so that 1907 should see the present production of 5,000,000 ℔. per month greatly increased. It is said that the company will be able profitably to handle ore running as low as 1 per cent. copper by the improved and enlarged plant.

What promised to be another immense company was organized in July by T. F. Cole and associates, among them W. C. Greene; it was known as the Cananea Central Mining Company, with a capital of $10,000,000 gold, and owned, among others, the American mine (for which $2,500,000 gold was paid,) and the Cananea-Duluth, which was obtained for $500,000 gold and a certain interest in the new company. By the middle of September the company was shipping 100 tons of ore per day. Recently a merger was effected between the Cananea Central and the Greene Consolidated, and for the combination a new holding company, the Greene-Cananea, was formed, with a capital of $15,000,000. Other producers at Cananea are the Sierra de Cobre mines, of the Copper Queen Consolidated Copper Company, now producing 200 tons a day of 5 per cent. ore and continually increasing; the Southwestern Copper Company, the Cananea Eastern, the Ronquillo Copper Company, and the South Cananea, besides which there are several prospects developing.

Among other more important copper properties of Mexico are La Dicha mines of the Mitchell Mining Company, in Guerrero, producing 15 tons of copper per day; the Jimulco Mining Company, with its railway and mines at Jimulco and Panuco, whose work on the building of its own smelter at the latter point was checked by an exceptionally favorable contract with the American Smelting and Refining Company; the Mazapil Copper Company, with its mines, smelters and railroads in the States of Zacatecas and Coahuila, whose new smelting plant in Saltillo has just been completed; the Rio Tinto, with its 250-ton smelter at Terrazas, Chihuahua, which has just been blown in; the Teziutlan Copper Company, with its large smelting plant in the State of Puebla, which company added largely to its holdings during 1906 and greatly increased its ore reserves; and a number of smaller properties in the southern part of the State of Jalisco, and in the States of Michoacan and Guerrero.

Newfoundland. (By W. Spencer Hutchinson).—The Tilt Cove mines are operated by the Cape Copper Company, Ltd., and produce more than half of the copper ore of Newfoundland. Their output, which is shipped to England and New York, now approaches 100,000 tons annually, and they have a production record exceeding 1,000,000 tons. This ore is pyritic and is used by the sulphuric acid makers. The average copper content is about 3.5 per cent.

Tilt Cove is situated in Notre Dame bay on the north shore of Newfoundland. Its harbor is no more than a notch in a shore line of precipitous cliffs. Vessels can dock for loading cargo only in fair weather, but good harbors are found in the deep arms or bays a few miles down the coast.

The mines have been worked more or less continuously for 35 years. The principal orebodies occur in serpentine schist and are adjacent to a strong dioritic dike. The largest mass of ore was 350 ft. by 280 ft. at the

surface and 220 ft. deep. At the bottom it pinched out, the mass having had the general form of an inverted cone. This was worked formerly in a system of chambers, leaving pillars and floors. A few years ago the overburden of gravel and peat was stripped off, and they are now taking out the pillars and floors, working from the top downward in a great open pit. Another large ore body recently opened is now being mined by a system of chambers.

Economic conditions are unique; labor is cheap and fuel expensive; in consequence the mine is operated practically without machinery. The ore is hard but all drilling is by hand. The cost of mining from the big open pit is $1.21 per ton. The equipment at Tilt Cove is in marked contrast with that at the Pilley's Island mine, also in Notre Dame bay, where an American company has installed modern hoisting machinery and a fine air compressor.

Peru.—The copper production of this Republic showed an important increase in 1905, and again in 1906, but not so much in the latter year as was expected, the new smelting works at Cerro de Pasco having failed to operate as smoothly as was hoped. It is believed that Peru has highly important copper resources and will within a few years attain a noteworthy position in the statistics of the industry. Besides the Cerro de Pasco and Morococha basin, there are various other mining centers, including Huayllay, Huachocolpa, Quiruvilca, and Apaicancha.

The Ica district has veins of oxidized ore; in some veins oxichloride of copper (atacamite) appears to have been precipitated; in depth appear the transition zones of sulphides, in part enriched by secondary concentration. The rock in which the veins occur is labradorite diorite. In no important mine has the sulphide zone of secondary concentration been worked, much less that of the primary sulphide zone; in no mine has the water level been reached, except in the Adelaide mines of Cansa, where rich black sulphides appeared when water was struck, the workings being abandoned on account of the latter. The dip of the veins is almost vertical. The orebodies take the form of columns, and the bunches, *bolsonadas*, of rich ore in these columns are separated by sterile or very poor zones. The ore so far produced in the district has averaged 25 per cent. copper.

The Cerro de Pasco smelter blew in one furnace on Jan. 7, 1906, but had to stop shortly to repair machinery, resuming on Jan. 22, since which date it was run at intervals. This furnace is 56x180 in., and is rated at 500 tons daily capacity, though its actual capacity is still to be determined. The plant has four furnaces, all of them being of the above dimensions. Each is blown by a Roots blower, No. 11. The plant has four stands of upright converters. The ore now goes about 6 per cent. copper, this being the second-grade ore of the mines; the first-grade goes up to 12 per cent.

copper. All the ore now treated comes from the Diamante mine, six miles from the smelter, with which it is connected by standard gage railway. The smelter is using coke from England and coal from Australia, but a railway will soon be completed to the company's coal mines about 15 miles from Cerro de Pasco. A coal washery is to be installed there. A coking plant of 72 beehive ovens has been built at the smelter.

The ores are smelted raw, fluxed with 5 to 10 per cent. limestone, using 12 per cent. coke on the charge. The blast furnaces are charged mechanically, and so far as possible all work throughout the plant is done by machinery. The native labor is poor and is relied upon as little as can be helped. Owing to the generally unfavorable conditions, the natural infantile difficulties of a new plant, and troubles in the management, the smelter did not get into regular operation in 1906, and consequently the large production of copper that was anticipated from this source did not materialize. It is hoped that the mistakes will be rectified in 1907, R. H. Channing, formerly manager of the Utah Consolidated Mining Company (Salt Lake City), having been sent to Cerro de Pasco to take charge.

The Peruvian Mining and Smelting Company, affiliated with the United States Smelting, Refining and Mining Company, will erect a custom smeltery at Rio Blanco, on the coast, in 1907.

Rhodesia.—Important developments have been going on in northern Rhodesia. The Cape to Cairo Railway has been pushed forward with great energy. The river Kafue, one of the great feeders of the Zambesi, has been spanned by a bridge 1400 ft. long, and the railroad is now at Broken Hill, 130 miles farther north. The three important mining companies in the district are the Rhodesia Copper Company, the Northern Copper Company, and the Broken Hill Development Company.

Railway surveys are being made to Bwana M'Kuba, about 100 miles north of Broken Hill. Mines there are said to be rich in copper. The old workings by the natives extend 2500 ft. in length, and go down 100 ft. They show two reefs, one 12 ft. and the other 8 ft. wide, with a space of 30 ft. between them. The two reefs come together and form a body of ore 12 to 15 ft. in thickness. The reefs are reported to have been proved to a depth of 280 ft., and the copper is claimed to average 15 per cent.

Russia.—The chief copper producing districts of this Empire heretofore have been the Caucasus and the Ural. In 1906 Siberia began to come into prominence through British and French investments in its copper mines. Siberian enterprises were, indeed, quite fashionable in London in 1906.

In the Caucasus, the principal copper mines are situated in the Ganskinski mountains (Governments of Tiflis and Elizabethpol); in the Sangesour and Nakitchevan districts, in the basin of the Bergonschet, a tributary of the Arados; north of the Ganshin mountains, at the headwaters of the Alasan, Telaw district; in the region north of the Kutais Govern-

ment, at the headwaters of the Riou; and in the Batum and Arterin districts, south of Batum. The principal smelting works are the Siemens, at Kedabeg, the Allaverdi, belonging to a French company, and the Sangesour. The annual production of copper in the Caucasus is 3500–4000 tons.

The following are the quantities of copper produced by the Ural copper smelting works during 1902–1905 (in poods: one pood equals 36.114 lb.)[1]:

Works.	1902.	1903.	1904.	1905.
Vijski works of the estate of P. P. Demidoff, in the Nijni Tagilski district.....	144,644	130,398	131,776	87,125
Bogoslovski works..........	27,064	76,416	74,447	93,035
Pishminsko-Klutchevskoi, of Count Stenbok-Fermor, in the Upper-Issetzk district	19,655	31,080	25,078	25,582
Verchotorski works of Mr. Pashkoff................	19,511	27,492	26,013	16,931
Yugovski works, of Mr. Zacharov................	48,750	....	....	....
Total....................	259,624	265,116	267,324	222,674

The decline in the output in 1905 occurred entirely at the Vijski mine of the executors of P. P. Demidoff, which was caused by the flooding of the mine. At the Bogoslovski works, the output of copper in 1905 increased, due to the discovery in that district of new and rich deposits of ore.

In the Orenburg province the copper smelting industry is beginning to decay. In that province, copper ore is produced principally at the Kargalinski mines of V. A. Pashkoff. In 1905 twelve mines were in exploitation. The ore has been smelted at the Verchotorski works of Mr. Pashkoff, which were closed in 1906.

The property of the Spassky Copper Mine, Ltd., in Siberia, were described in *Eng. and Min. Journ.*, March 31, 1906. The first report of the directors covers the period from the formation of the company in July, 1904, to Sept. 30, 1905. When the properties were taken over, it was found to be the best policy to continue to work the old furnaces until new ones could be erected, especially for the purpose of training the employees, but also with the object of maintaining the influx of money into the district and keeping the people employed. At the outset the production of copper was only 50 tons a month. This has steadily risen to 100 tons a month, with the same plant. During the same time the working costs of mining, smelting and refining have been reduced from £55 to £35 per ton of copper produced. The actual amount of copper produced during these 15 months is not given, but the receipts from sales were £85,000. The price of copper in Russia is higher than in other parts of the world owing to the high import duty. The price obtained varied from £90 to

[1]Reported by *Vestnik Finansov, Promishlennosti i Torgovli*, through *Min. Journ.* (London).

£105. The quality of the copper produced is good, the analysis giving 99.61 per cent. New plant is now on the way to the mine and by the autumn of 1906 it was expected the output should be increased to 300 tons a month. Subsequent arrangements for an increase in production will depend largely on political and labor conditions. The Spassky mine contains large quantities of rich ore, and the prospects are excellent.

Early in 1906 the Russian Government put a duty of 3.75 to 5 rubles per pood on all copper imported into Russia. The higher figure—5 rubles per pood—is equivalent to about 7c. per pound. The duty is intended for revenue; and also, incidentally, to assist the domestic mining industry.

Spain and Portugal.—The copper production of the Huelva district showed a small increase in 1906. Several new enterprises were brought out in 1906. Among these was the Esperanza. The ore bodies in the Esperanza mine are at the contact of slates and porphyry, along the same line as the Pena and San Miguel, and it is estimated that the known and probable ore amounts to over a million tons, divisible about equally into the three classes known in Spanish pyrites mining. The first class is rich enough to be sold in England as copper ore, the second is leached locally and the exhausted pyrites subsequently sold for its sulphur contents, and the third is sold at once for its sulphur contents. At most of the mines in the Huelva district there is comparatively little smelting ore and the profits are obtained by leaching the copper and selling the sulphides to the chemical merchants for sulphur contents. The Esperanza, on the contrary, has a large amount of ore running 6 per cent. copper and 48 per cent. of sulphur. The capital of the newly formed Esperanza Mining Company is £350,000 in ordinary shares and debenture bonds amounting to £100,000. The purchase price was £381,500, partly in cash, shares and debentures. The issue of shares was underwritten in London, where the chief office is.

At the Rio Tinto, the opencast mining operations were steadily pushed on with, and the removal of overburden from one of the lodes has for some time past been carried out largely by the use of steam shovels, which not only cheapens the cost, but expedites removal. The copper output in 1905 was seriously interfered with by the drought which prevailed in the province of Andalusia, the scarcity of water making it impossible to wash the mineral heaps with sufficient frequency. The lack of rain does not reduce the quantity of copper being prepared for washing, but the output of copper suffers in a dry season. This shortage is made up again when water is sufficient, so that over a period there is no reduction in the quantity of the copper marketed.

The Rio Tinto company has put some new furnaces into operation at the mine to treat a portion of the poorer sulphide ore by the bessemer process, instead of by leaching as heretofore. The leaching method would not have released the copper for two to three years, and inasmuch as it

is anticipated that 1000 tons of ore will be smelted daily, the quantity produced is likely to be from 4000 to 5000 tons of copper per annum. Unless, however, the output of ore is raised, this is merely an anticipation of the production. The move is considered to be a wise one, inasmuch as the water problem becomes more and more difficult every year; on the other hand, the sulphur content of the ore will be lost by the bessemer process.

The quantity of copper from the old furnace bottoms at Cwmavon, Wales, recovered during 1905 was 912 tons, against 3011 tons in the previous year. Before the end of 1906 it was expected to finish this operation and to abandon the old works and surrender their lease. The company has commenced to refine copper in a small way at the new refining plant erected on the dock side of Port Talbot, into which its own vessel directly delivers materials from Spain, and it is hoped soon to transfer the whole of the refining operations to these new works.

The old mines at Alpartir (Zaragoza), which were worked by the Romans, but have been idle since 1850, when they were abandoned because of the heavy influx of water, were reopened in 1906 by the Sociedad de Minas y Sondeos. They are said to have important reserves of ore.

Ores of copper form the largest part of the mining yield of Portugal. The largest mine is the San Domingos, at Mertola, near the Spanish boundary. The ore is carried to Pomarao, on the Guadiana river, and thence to England. The cupriferous pyrites mines, Aljustrel, are owned by a Belgian company of Antwerp. Most of the ore is sent by railroad to Barreiro, on the bank of the Tagus, opposite Lisbon, but part of it is leached at the mine, and cement copper recovered. The Tharsis mine, of the Huelva district, is no longer an important producer of copper, the pyrites having become impoverished with depth and now being valuable chiefly as a sulphur ore.

Sweden.—The chief part of the copper production of this country is derived from the Nautauen district, near Gellivare, and from the Falun mine. A new company was organized in 1906 to develop a property in the Gellivare district.

Tunis.—Copper is mined at Chouichia in Tunis, where the ore is smelted to a matte, assaying 35 to 50 per cent. copper, which is exported.

The Copper Markets in 1906.

New York.—The history of 1906 may be summarized in the single statement that consumption outran production. The stocks of refined copper in this country, and abroad, having been practically exhausted, there became a famine with more or less serious consequences to the manufacturing industry during the last few months of 1906, when all at once the demand became larger than the available supplies. Under such cir-

cumstances a very high level of prices was unavoidable, although the larger selling agencies did everything they could to prevent a material rise in prices from the already comparatively high level which existed earlier in the year, but they soon found themselves unable to stem the tide.

Important factors, which developed more and more as the year progressed, were scarcity of labor and insufficient supply of fuel and transportation facilities. Although wages were repeatedly increased during 1906, there was continually a shortage of miners and smelter men. Both in the Southwest and Northwest the smelters were very short of oil, coal and coke, and repeated shut-downs were the consequence. The railroads were utterly unable to provide the necessary cars for the transportation of fuel and partly also for the transportation of ore, and all these causes contributed largely toward reducing the production from the estimated figures.

The year opened with Lake copper at 18½ to 19c.; electrolytic at 18½ to 18⅞c., and casting copper at about 18¼c. During January and February the market was rather sluggish. The Orient was re-selling copper in Europe and here. American consumers were covered and held off, and the European speculative market declined somewhat. By the middle of February, Lake copper had declined to about 18c., and electrolytic to about 17¾c.

Meanwhile, the winter in this country had been exceptionally mild, permitting much work to be done which usually must be deferred until spring. This made business very active, so that our home manufacturers who found their supplies of copper exhausted entered the market again.

At the end of February the Europeans who had been skeptical as to the maintenance of the level of prices had also to replenish their stocks. As a result, prices gradually advanced, and during March, in spite of the fact that China was still selling copper, Lake again went to 18¾c., and electrolytic to 18½c. The London market for standard then began to show signs of the shortage of supplies. In fact, toward the end of March, it was practically cornered, having advanced £7 during the month.

In April it became evident that consumption was much in excess of production and consumers bought ahead with confidence, covering their wants until about the first of September, at about 18¾c. for Lake, and 18½c. for electrolytic. The producers recognized the situation, but sold freely at these prices, which were quite satisfactory to them; and besides, they were desirous of preventing a runaway market, with its inevitable reaction and bad effect upon the trade generally.

In May copper for early delivery became very scarce, and premiums were paid for it. Toward the end of the month consumers abroad as well as here again bought heavily for future delivery at about 18¾c. for Lake and 18½c. for electrolytic, with casting selling at 18½c.

June and July were dull months because buyers had already covered their requirements. The political situation in Europe at this time looked grave, and this induced the London speculators to press upon the standard market, selling it down to about £78, but by August the immense increase in the consumption again made itself felt. Manufacturers were receiving orders far ahead and they came into the market, buying heavily for delivery up to the very end of the year.

September opened with producers practically sold out for both September and October, and with large orders on their books for November and December, and prices advanced to 19c. for Lake and 18¾ for electrolytic. The orders which manufacturers were receiving extended far into 1907, and led them to buy for the first six months of 1907. Meanwhile, copper for early delivery commanded fancy prices, and the month closed with Lake at 20c. and electrolytic at 19¾c.

During October the shortage in supplies led to a squeeze in the London market, and standard copper went as high as £102 15s. Some of the largest producers who were sold out for 1906, and had refrained from booking any orders for 1907, now offered to sell for that delivery at 22½c., and were quickly cleaned out. In fact, at this time several of the important producers were already entirely sold out for the first quarter of 1907.

In November the market was somewhat more quiet, but in December the European buyers who had held off during November made heavy purchases for 1907, and prices advanced rapidly, closing at about 24½c. for Lake, about 23⅝c. for electrolytic, and 23¼c. for casting copper.

AVERAGE PRICE OF LAKE COPPER PER POUND IN NEW YORK,

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900	16.33	16.08	16.55	16.94	16.55	16.00	16.16	16.58	16.69	16.64	16.80	16.88	16.52
1901	16.77	16.90	16.94	16.94	16.94	16.90	16.51	16.50	16.52	16.60	16.60	14.39	16.55
1902	11.322	12.378	12.188	11.986	12.226	12.360	11.923	11.649	11.760	11.722	11.533	11.599	11.887
1903	12.361	12.901	14.752	14.642	14.618	14.212	13.341	13.159	13.345	12.954	12.813	12.084	13.417
1904	12.533	12.245	12.551	13.120	13.000	12.399	01.505	12.468	12.620	13.118	14.456	14.849	12.990
1905	15.128	15.136	15.250	15.045	14.820	14.813	15.005	15.725	15.978	16.332	16.758	18.398	15.699
1906	18.419	18.116	18.641	18.688	18.724	18.719	18.585	18.706	19.328	21.722	22.398	23.350	19.616

AVERAGE PRICES OF ELECTROLYTIC COPPER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900	15.58	15.78	16.29	16.76	16.34	15.75	15.97	16.35	16.44	16.37	16.40	16.31	16.19
1901	16.25	16.38	16.42	16.43	16.41	16.38	16.31	16.25	16.25	16.25	16.22	13.82	16.11
1902	11.053	12.173	11.882	11.618	11.851	12.110	11.771	11.404	11.480	11.449	11.288	11.430	11.626
1903	12.159	12.778	14.416	14.454	14.435	13.942	13.094	12.962	13.205	12.801	12.617	11.952	13.235
1904	12.410	12.063	12.299	12.923	12.758	12.269	12.380	12.343	12.495	12.993	14.284	14.661	12.823
1905	15.008	15.008	15.125	14.920	14.627	14.673	14.888	15.664	15.965	16.279	16.599	18.328	15.590
1906	18.310	17.869	18.361	18.375	18.457	18.442	18.190	18.380	19.033	21.203	21.833	22.885	19.278

London.—The year opened very auspiciously, trade having lately expanded in all directions. Metals generally had advanced in value, in consequence of genuine trade demand, and stocks were everywhere light, with no prospect of early repletion. The opening prices for standard warrants were £79 15s. for spot, and £79 7s. 6d. for three months. The market was forthwith affected, partly by speculative holders, who desired to realize, partly also by "bear" selling, under which influences, three months' standard gradually relapsed until £75 12s. 6d. was recorded, but spot warrants were firmly held, and as bear sales matured, the premium on these warrants was at one time as much as £2 5s. per ton. A recovery of nearly £2 was recorded in the price of three months' standard before the month closed, when spot warrants stood at £78 10s., and three months' at £76 17s. 6d.

February opened with a firm tendency induced by further shrinkage in the visible supply; but an early relapse followed, due to large realizations combined with the pressure of bear sales, which brought down the price of three months' standard to £74 15s., followed by a sharp recovery to £76. Thereafter bears showed more caution, and the market was steady in consequence. Spot copper continued to command a premium. The month closed with spot warrants at £78 15s., and three months' at £77 2s. 6d.

March opened with every indication of firmness, prompted by successive advances in American prices. Manufacturers who had held back in hope of lower prices, now hastened to cover their requirements, and the demand for consumption was brisk. The covering of bear sales, which had been less active in the early part of the month, developed new importance toward the end. The month showed a steady advance in the standard cash price from £79 at the outset to £83 15s. at the close, three months' warrants closing at £80 17s. 6d.

April showed increased orders for standard copper for forward dates, both from speculators and consumers, and a gradual advance in price was recorded throughout the month. The market closed strong, with cash warrants at £85 5s., and three months' at £83 5s.

May opened with a downward tendency, due to disturbing influences in American finance, and during the first eight days spot warrants relapsed gradually. Producers, however, were well sold and would make no concessions, though second-hand holders were restive. Refined copper continued in good demand for consumption, notably in Germany and chiefly for electrical work. The situation was therefore ripe for improvement, which commenced early in the month, by which time the financial outlook had greatly improved. With the approach of the Whitsun holidays the turnover dwindled, and operators hesitated to initiate any movement

either way. Closing prices were, for spot standard warrants, £85, and for three months, £84 10s.

June opened quietly, but with favorable indications. The long impending labor troubles in Germany were happily averted. Prices gradually advanced until, on June 12, spot warrants commanded £86 5s., and three months' £85. By this time demand was temporarily satisfied, and the disquieting aspect of Russian affairs tended to disturb the money market. Bears were alert to take advantage of the opportunity, and no support was forthcoming. Meantime there was some active business in spot warrants which were in request chiefly for the covering of bear sales, but prices relapsed in the absence of any pressing demand, and under unfavorable reports from Wall street, where industrial shares were sharply attacked. The close of the half-year was quiet, but there was a manifest disposition to buy in the neighborhood of £80, and producers would make no concessions. Closing prices were £81 5s. for cash standard, and £80 10s. for three months'.

July opened uneventfully, with little speculative enterprise. A further increase in the visible supply, and a general depression in the stock markets, combined to initiate a gradual decline, which, followed by heavy liquidations induced by losses on tin, brought down the prices to £78 5s. for cash, and £77 15s. for three months' warrants, at which figures the market closed, July 13. Toward the close of the month the market became free of extraneous influence. Manufacturers bought largely, and raised their basis price for the rolled metal by £2 per ton. Bears became alarmed, and hastened to cover their commitments. The month closed with cash warrants at £82 10s., and three months' £81 10s.

August was marked by increased evidence of large requirements extending over long periods, with bookings in many cases extending into the next year, and at persistently advancing prices. One natural result was the early disappearance of the backwardation, which had ruled so long. Aug. 13 found the price uniform for cash, three months', and intermediate dates, and thus it continued for the rest of the month, with sometimes a fractional premium for the latter dates. The speculative element gradually dwindled so that the public turnover was often very meager, and official prices little more than nominal, but the consuming industries were none the less active in absorbing all that was offered. German consumption, in particular, was on a larger scale than had hitherto been known. India supplied numerous orders, contrary to her custom when prices are high, and China did likewise. The home consumers alone showed any timidity, for which they had to pay the penalty after their stocks were worked down and replenishment could no longer be delayed. There was a slow but steady advance in prices all through the month, closing at £85 5s. for cash warrants, and £85 7s. 6d. for three months'.

September showed a steady advance. Leading producers had already sold out for some months ahead. Japanese supplies were taken for the first half of 1907, and consumers secured important contracts extending well into the spring. Important orders came in from India and China. Toward the end of the month it was manifest that manufacturers were committed to a larger output than the producers could furnish. Bears hastened to cover—in most cases at heavy loss—and bulls were no less prompt in paying the already advanced price for forward warrants. Speculation, however, was comparatively unimportant beside the enormous business passing between producers and consumers. Home consumers were for the most part late in apprehending the real significance of the situation, and in some cases they paid full prices for delivery up to June, and even beyond. The closing prices of standard copper were £91 5s. for prompt delivery, and £91 for three months'.

October opened with an advance to £93 for all deliveries, followed by a slight check due to bear enterprise, which latter was, however, quickly closed at a loss. There was furious buying in the English market, particularly of refined sorts, and outside speculative operations intensified the excitement. On Oct. 16, £100 was paid, and on Oct. 18, £103 10s. was paid, which proved to be high-water mark. Thereafter enterprise was sobered somewhat by the official bank rate of interest being raised from 5 to 6 per cent., as also by the failure of a prominent house having large commitments in copper on the bear side. Prices were forced down to £97 5s. on Oct. 25. The month closed in comparative quietness, and with practically the entire production of the metal engaged for four months to come. Closing prices of standard warrants were £97 for cash, and £97 15s. for three months'.

November opened with an improvement in the financial situation, and a revival of speculative interest in the metal. European consumers showed some reserve, but the volume of business increased with each advance. Top prices were recorded Nov. 19, when cash warrants commanded £102, and three months' £103 5s. A slight reaction followed, and on Nov. 21 cash warrants closed at £100 15s., and three months' £102, but further bear attacks were repulsed. The month closed with cash warrants quoted £102 10s., and three months' £103 7s. 6d.

In December the advance was persistent. A remarkable episode was the sale of over 2000 tons by the Rio Tinto company at the record price of £110, for January and February delivery, besides a large volume of miscellaneous business with consumers. Toward the middle of the month it was found that electrolytic copper was practically unobtainable for earlier delivery than April, demand being particularly keen for this class of copper. Speculative business also became lively, and a large bull account was manifest toward the close, whereby a contango of 30s. was

established for three months' warrants of standard. Even the Indian markets—which usually offer a passive resistance to a high range of prices—now found outlets for appreciable quantities of manufactured material, while demand from the European continent—especially from German manufacturers—was brisk. At the close cash standard was held for £106 15s., and three months' for £108 5s.

AVERAGE PRICE OF STANDARD COPPER (G. M. B.'s) IN LONDON.
(In pounds sterling per ton of 2240 lb.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1901	71.78	71.17	69.54	69.61	69.60	68.83	67.60	66.34	65.97	64.11	64.51	52.34	66.79
1902	48.43	55.16	53.39	52.79	54.03	53.93	52.89	51.96	52.68	52.18	51.03	50.95	52.46
1903	53.52	57.34	63.85	61.72	61.73	57.30	56.64	58.44	56.82	55.60	56.30	56.36	57.97
1904	57.500	56.500	57.321	58.247	57.321	56.398	57.256	59.952	57.645	60.012	65.085	66.375	58.884
1905	68.262	67.963	68.174	67.017	64.875	65.881	66.887	69.830	69.667	71.406	74.727	78.993	69.465
1906	78.869	78.147	81.111	84.793	84.867	83.994	81.167	83.864	87.831	97.269	100.270	105.226	87.282

PROGRESS IN THE METALLURGY OF COPPER.

BY L. S. AUSTIN.

Chemical and Physical Data

Fusion Points of Copper and its Oxides.[1]—Admitting that the temperature of fusion of gold was 1065 deg. C., that being the average of closely agreeing results of independent observers, it has been possible to make concurrent observations upon it and also upon pure copper, and upon its alloys with cuprous oxide in various proportions, as shown in Fig. 1. The temperature of solidification of pure copper has been determined at 1085

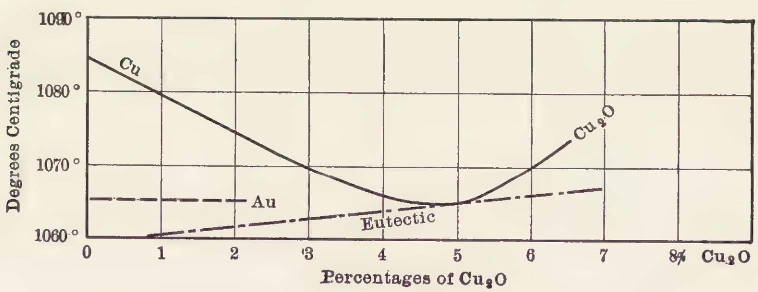

FIG. 1.—FREEZING CURVE OF COPPER-CUPROUS OXIDE ALLOYS.

deg., or 20 deg. above that of gold, while the eutectic (Cu 95 per cent., and Cu_2O 5 per cent.) solidifies at 1065 deg., the freezing point dropping with increasing cuprous oxide to the point of 5 per cent Cu_2O and then rising as the oxide increased in quantity. The slight indentation of the line of the eutectic is probably due to the presence of small amounts of impurities which the Cu_2O entirely takes up.

[1] *Revue de Metallurgie*, III. p, 233.

Refining Copper With the Use of Silicon or Manganese Silicide.[1]—The crude metal contained 97.25 per cent. of copper and 1.77 per cent. of cuprous oxide. Silicon had no effect in reducing the protoxide, but better success was attained by the use of manganese silicide of 19.77 per cent. silicon adding 0.44 per cent. of the silicide to react on the copper. The refined copper was very dense, free from oxide, manganese and silicon, and the loss of copper was but 1 per cent.

Constitution of Copper Matte.[2]—E. Keller, discussing a paper by Allen Gibb and R. C. Philp, in the *Bimonthly Bulletin* A. I. M. E., November, 1905, thinks these writers have taken insufficient account of the presence in copper mattes of magnetic iron oxide, in many cases dissolved in the matte, especially in low-grade reverberatory mattes, while high-grade mattes of the same origin contain little or none.

Distillation of Copper.—According to Moissan, copper may be readily distilled in an electric furnace, and when its vapor is condensed on a cool surface, a filiform felting of copper is obtained having all the properties of ordinary copper. At its boiling point copper dissolves graphite, and upon cooling, the graphite crystallizes out.

Electrolytic Deposition of Copper.[3]—The action of organic colloids upon the deposits has been studied as follows: With gelatin the copper deposit in an acid solution, with a current density of 3 amperes per sq. ft. and at ordinary temperatures showed numerous vertical bright bands or stripes, and, as the current density increased, the bands become wider, until with 3.25 amperes, brilliant, quite homogeneous deposits of high reflecting power were obtained. For their formation clear, well-filtered solutions are needed, and their occurrence has been explained as due to the fact that the metal is kept in amorphous condition owing to the mutually protective influence of colloids on one another.

Alloys.

Study of Special Brasses.[4]—If in a standard brass of Cu 60 per cent. and Zn 40 per cent. we substitute another metal, as tin or aluminum, the metal so substituted will take the place of and make an equivalent alloy. The amount so substituted will have a certain ratio to that of the original zinc, called the coefficient of equivalence.

Thus, take the alloy Cu 70 and Zn 30 per cent., in which we will replace 2 per cent. of zinc by 2 per cent. of aluminum. We shall then have the alloy Cu 70, Zn 28, and Al 2 per cent. Suppose now that aluminum has such an equivalence that 1 per cent. of it is equal to 6 per cent. of zinc. It will give us a fictitious alloy of $Cu=70$, and $Zn=28+(2x6)=40$, or 110

[1] *Metallurgie*, 1906, III, 253.
[2] *Bi-Monthly Bulletin*, A. I. M. E., March, 1906.
[3] *Journ. Soc. of Chem. Ind.* May 3, 1906.
[4] *Revue de Metallurgie*, 1906, III, 243.

parts in all. Reducing this to give a total of 100 per cent., we get Cu 63.63 and Zn 36.37 per cent. and having the properties of a straight alloy containing the copper and zinc in those proportions. This may be expressed algebraically by the formula

$$A^1 = \frac{100\ A}{100q\ (t—1)}$$

in which A^1 is the fictitious percentage in copper of an alloy containing in fact A per cent. of the metal and /q per cent. of the substituted metal whose coefficient of equivalence is t.

TABLE OF COPPER-ALUMINUM ALLOYS.

No.	Composition Cu. %	Composition Al. %	Fictitious value found by calculation %	Micrographic Examination.	Tension Tests. Ultimate Strength	Tension Tests. Elastic Limit.	Tension Tests. Elongation.	Tension Tests. Reduction of Area.	Resistance to Shock.	Hardness.	Remarks.
1	68.73	7.14	58.1	Solution *a* and *b*	36.8	28.9	1.5	....	6	140.0	(*c*)
2	69.02	0.40	68.0	Solution *a*	20.3	4.7	59.0	....	13	33.6	
3	70.01	0.90	67.1	Solution *a*	22.7	6.6	67.0	....	13	33.8	
4	70.53	3.10	61.5	Solution *a* and traces of Solution *b*	33.9	13.5	50.0	....	7	65.0	
5	69.38	4.95	55.7	Solution *b* and a little of Solution *a*	40.1	15.0	11.0	11.5	8	104.0	
6	70.10	5.20	56.3	Solution *b* and a little of Solution *a*	50.8	7.4	11.0	15.3	5	107.0	
7	68.47	4.50	55.0	Solution *b* and traces of Solution *a*	44.8	23.8	24.5	28.4	11	88.0	
8	68.49	5.60	53.5	Solution *b*	54.4	47.6	1.0	1.4	5	148.0	(*c*)
9	69.16	6.59	52.0	Solution *b*	51.0	51.0	1.5	1.4	3	153.0	
10	70.11	9.45	47.6	Solution *b* and *a*	....	....	....	....	.....	....	(*d*)
11	64.73	1.64	58.8	Solution *b* and *a*	36.0	12.4	38.0	40.7	10	69.0	
12	59.61	0.36	59.0	Solution *b* and *a*	32.3	9.9	51.5	44.5	19	51.0	
13	59.88	1.17	56.5	Solution *b* and a little of Solution *a*	33.5		30.5	35.4	10	67.0	
14	59.92	0.83	57.5	Solution *b* and *a*	30.9	9.5	45.0	40.7	14	52.0	
15	59.61	2.62	51.2	Solution *b*.......	46.0		14.0	15.3	9	101.0	
16	59.06	1.88	53.8	Soluiton *b*.......	37.4	37.4	11.0	15.3	5	101.0	
17	59.03	3.16	51.64	Solution *b*.......	46.9	40.8	12.5	15.3	5	107.0	

(*c*) Cannot be worked. Very fragile. (*d*) Poorly broken.

Ordinary Brass.—Micrographic examination. The characteristics of the alloy are mainly as given by Shepard[1] and are as follows:

Between 100 and 71 per cent. of copper we have (*a*) crystals of a full yellow changing toward a bright copper-red, large and with rounded lobes. From 71 to 63 per cent. and (*b*) crystals, the latter being red, or of a coppery color, and in narrow masses; the whole surface red, changing to a full yellow. From 63 to 55 per cent., (*b*) and (*a*) crystals, the surface being of a reddish yellow with a yellowish cast. Throughout this range it is possible to know to within 0.5 per cent., and by the relative proportions of (*a*) and (*b*) crystals, the percentage of contained copper in

[1] *Journ. of Physical Chemistry*, 1904, VIII, 421–424

the alloy, so that one can determine quickly by a micrographic examination whether, for example, a brass can be worked hot or cold. If containing 55 to 63 per cent. Cu, it can be hot-rolled, while those brasses of over 60 per cent. may be cold-rolled.

The various types enumerated above merge into one another, a given type of crystals gradually disappearing as the copper percentage diminishes.

To sum up: Experiment shows that the characteristic element in the hot-working of brasses is solution (*a*) in part. Those brasses containing solution (*b*) only (55 to 56 per cent. Cu) have the drawback that they are hard to roll.

Aluminum-Brass.—For purposes of illustration, we give a table of copper-aluminum alloys with the fictitious copper value (A^1) calculated according to the formula given above, the micrographic equivalent, and the properties of the alloys in tension, for shock, and for hardness.

Copper Steel Alloys.[1]—A series of copper-steel alloys containing 0.10 to 0.17 per cent. as well as those having 0.28 to 0.41 per cent. carbon was alloyed with copper in eight members of, 0, 0.5, 1, 2, 4, 8, 16 and 32 per cent. copper. The ingots up to 4 per cent. showed no color on the fracture, but from 8 per cent. upward the color was marked and increasing. All the alloys up to 4 per cent. copper could be hammered or rolled, the others not. The hardness increased with the copper. All were magnetic when cold.

Details of Works—Products.

Briquetting Mill at the Washoe Plant.—The briquetting material consists of one-third ore (undersize of ⅜-in. trommel), one-third pond-slimes, one-third table-concentrates, and coke to the extent of 5 per cent., this coke being the result of washing the reverberatory grate-droppings. This latter material pugged into the brick gives them a more open texture for the easy escape of moisture. The coke thus incorporated amounts to about 40 tons daily. From the four chutes of the inclined-bottom storage bins which hold them, these materials fall upon a conveyor belt, the discharge from each bin being regulated by a feeder, while the discharge from the coke-bin is attended to by the pug-mill man. The men become expert at regulating the flow and modify it according to signals from the pug-mill man. The above-mentioned conveyor discharges to another which takes the materials to the supply hopper of the pug-mill. Here a double-spindle horizontal pug-mill, making 22 r.p.m., incorporates the various ingredients with just enough water for proper molding in a brick press situated below the discharge opening of the pug-mill.

This press, made by Chambers Brothers Company, Philadelphia, Penn.,

[1] *Comptes Rendus*, 1906, CXLII, 1421–1424.

is a No. 7 end-cut worm-feed brick machine. It delivers a continuous slab of molded mud, which, as it issues from the press, is wire-cut into brick at the rate of 840 tons per day.[1] These brick weigh 8 to 10 ℔. each; and the machines, when in full operation, can turn out 160 bricks per min. These are transmitted by a conveying belt to another at right angles and parallel to the long side of the building, and over a row of hoppers of one ton capacity each. Hand-operated deflecting-boards sweep these briquettes into the hoppers, charging them successively, the man in attendance filling the even, then the odd-numbered hoppers. The hoppers are steel-lined and have drop-bottoms. The bottoms of the odd- or of the even-numbered hoppers are dropped simultaneously, filling the 18 charge-cars below. The precaution is taken of first filling the cars with dry coarse concentrate, so that, when the soft bricks drop, they will not stick to the car-bottom.

Ore Bedding at Cananea.[1]—The novel system for receiving and handling the ores of this plant may be thus described: The ore comes in hopper-bottom cars by an elevated track commanding the track-bins. These storage bins have hopper-bottoms. A traveling automatic feeder can be set beneath any one of these, so as to deliver a regular stream of ore to the troughed conveying-belt, which carries it to the crushing and sampling mill. Here the lump ore is crushed, and, again joining the fine, the whole passes through Vezin samplers, where two cuts, each of 1/10, are removed; so that the sample becomes 1 per cent. of the total ore. The principal part of the ore then goes to trommels, the undersize to the reverberatories, and the coarse ore to the blast-furnaces. This coarse material is conveyed to the ore beds (each of 8000 tons capacity or 24,000 tons in all) where it is stored in one of three long piles. These are of triangular cross-section, and are built up by a tripper which moves on an overhead belt over the center of each ore-pile, and, in its shuttle-like passage back and forth, drops ore always along the crest of the pile. It thus places the successive additions in concentric layers, and gives a uniform composition at any cross-section from end to end of the pile.

From the reclaiming machine at the ore bed a second system of belts conveys the ore to the blast-furnace building. A trench passes along at one side of the ore bed, in which runs a conveyor belt. The reclaiming machine, a girder-like carriage, spans the ore floor and the trench, and, as it gradually moves toward the ore bed, cuts a slice clear across at the bottom of the pile by means of a scraper-conveyor, and delivers the material in a continuous stream to a hopper, which shoots it to the just mentioned belt conveyor. A rake-like "tickler" keeps the end-face of the pile dressed at a slope toward the machinery, so as to present a thin edge for the plow blades of the scraper-conveyor and to prevent under-cutting and caving.

[1] *Eng. and Min. Journ.*, LXXXII, 624.

The carriage is actuated by electric motors, and the speed of cutting and of moving up can be regulated by the operator, who is guided by signals automatically set at a semaphore bin near the furnaces. This bin is supported on springs, and has a few inches of vertical movement, depending upon the weight of the ore in the bin. Settlement of the bin alters the position of a semaphore arm at the bin and informs the operator at the reclaiming machine that the bin is emptied and he speeds up to meet the increased demand.

The capacity of the receiving, sampling, and bedding machinery is 2000 tons in 10 hours, while the reclaiming and feeding machinery handles this amount in 24 hours. Each of the three ore floors will hold 8000 tons. Assuming one bed completed, one half-made, and one half-used, we have an actual capacity of 16,000 tons, or 8 to 10 days' supply. This should provide for all ordinary irregularities of supply.

The new furnaces at Cananea are to be fed mechanically. The device consists of a long narrow box on each side of the furnace, large enough for a charge. It is set upright to take a charge, then tipped to deliver into the furnace. The old-style furnaces are fed by dropping the charges upon the charge-plate from overhead measuring boxes filled by the conveyor, and feeding them into the furnace by hand. It will be noticed that charges are measured rather than weighed, but this serves well enough.

The concentrate fed to the furnaces, which constitutes the bulk of the charges, runs : 9.5 per cent. copper; 22.8 per cent. silica and alumina; 28.5 per cent. iron; 31.8 per cent. sulphur; and a small amount of silver. Raw ones, running above 5 per cent. copper, are smelted directly. Custom ores are also treated. For the treatment of the fine and flue-dust four McDougall roasting-furnaces have been put in of 18 ft. diameter with a capacity of 50 tons each daily; also a reverberatory matting furnace 100x 19 ft., the waste heat from it being taken to produce steam in three Sterling water-tube boilers, of 300 h.p. each, so arranged that two of them can be continuously operated.

The Largest Smelter Stack.[1]—The Boston & Montana Mining Company has let a contract to the Custodis Chimney-Construction Company for the erection of a stack at their smelter at Great Falls, Mont., to be completed by Sept. 1, 1907, at a cost of $200,000. The stack is circular, 50 ft. internal diameter at the top, and 75 ft. external diameter at the bottom. It is to be 500 ft. high, and will weigh 16,600 tons. It is to be made of perforated brick molded to the circle of the stack, and manufactured on the ground. There will be needed 5,700,000 brick. The top of the stack will be 742 ft. above the charging-floor of the blast-furnace. A flue 1800 ft. long will connect the furnace and the stack, 1450 ft. of which will be 48 ft. wide, and 20 ft. high, while 350 ft. will be 175 ft.

[1]*Eng. and Min. Journ.*, LXXX, 199.

wide and 20 ft. high, this portion being the dust-chamber. This large size is to provide for future expansion of the plant.

Products of Washoe Plant, Anaconda, Montana.—The McDougall roasting furnaces give:

	SiO_2	Fe_2O_3	S.	Cu.	Al_2O_3
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent
Roasted concentrates.......	30	42	9	10	10
Lumps..................	8	60	6	12	5
Flue-dust..................	36	27	19	10	10

The lumps build upon the rabbles and roof at the third floor, and are removed each shift from the furnace and sent to the blast-furnace. The escaping gases contain 5 per cent. of SO_2 gas. The flue-dust is briquetted and goes to the blast-furnaces.

The reverberatories produce:

	SiO_2	Fe	S	Cu	Al_2O_3	As & Sb
	Per Cent.	Per Cent	Per Cent.	Per Cent	Per Cent.	Per Cent.
Matte............	1	22	24	47	3	2
Slag..............	40	46	12	0.2	....	....

The blast-furnaces produce:

	SiO_2	Fe	CaO	Cu	Al_2O_3	S	As&Sb	Fe
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Matte...........	1.9	......	......	44	4	26	1	19
Slag............	41.0	28	22	0.2	8	......	......	......
Flue-dust.......	36	22	......	......	......	10	......	......

The converters produce: blister-copper of 97 per cent. Cu, traces of As, Sb, Fe, S, and Al_2O_3 and 70 oz. Ag per ton; and converter slag assaying 35 per cent. SiO_2, 56 FeO, 4 Al_2O_3, 4 CaO and 1.8 Cu. The blister copper is refined electrolytically, the electrolyte containing 3.2 per cent. Cu and 14 per cent. H_2SO_4, and the cathode copper 99.5 per cent. Cu and 0.0015 per cent, As and Sb.

Roasting.

Chemical Reactions in the Roasting of Pyrites.[1]—A. W. Warwick, from his experience in roasting, and from deductions made from numerous pyrometric observations and analyses, gives the following chemical reactions as taking place in roasting pyrite.

[1]*Mining Magazine*, XII, 196.

The complete transformation of pyrite into ferric oxide and sulphur dioxide takes place in the following stages:

(1) $2\ FeS_2+3O_2=2\ FeO+2SO_2$

(2) $2SO_2+2O+$catalytic agent$=2SO_3$

(3) $SO_3+FeO=Fe\ SO_4$

(4) $FeSO_4+FeOFe_2O_3+SO_2$ (This is rather $2FeSO_4+$heat$=Fe_2O_3+SO_2+SO_3$, since both of these gases are evolved. L. S A.)

The final result is:

(5) $4FeS_2+11O_2=Fe_2O_3+8SO_2$

But magnetic oxide is finally formed, in presence of a short supply of air, thus:

(6) $FeS_2+O_2=FeS+SO_2$

(7) $FeS+10Fe_2O_3=7Fe_3O_4+SO_2$

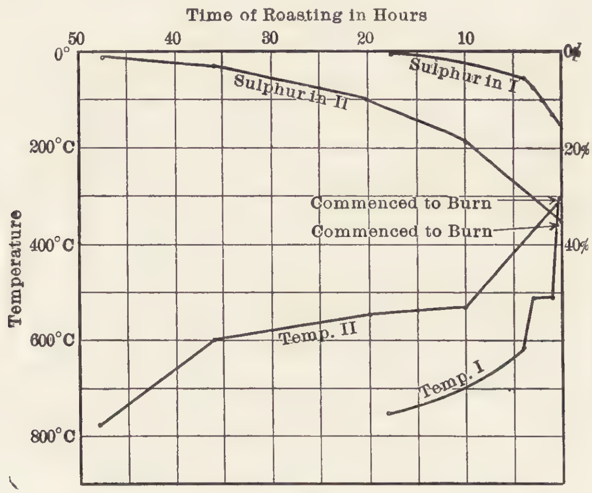

FIG. 2.—ELIMINATION OF SULPHUR IN ROASTING.

Roasting Pyrites Ores.[1]—A. W. Warwick gives the following regarding the roasting of pyrite:

When operating a 6x12 ft. Brückner cylinder roaster side by side with an 8x22 ft. one, he found the smaller furnace roasted a charge in 14 hours, while the larger one took from 32 to 34 hours, due, as he considered, to the greater mass of ore compared with the surface exposed in the larger cylinder. Experiments in roasting a pyrite ore in a Brückner cylinder showed (as is already well known) that the sulphur passes off rapidly and uniformly at first, then more and more gradually to the finish of the roast, even at the highest température of the furnace. Two ores were used, of the analyses given: (I) S, 15.2; As, 9.7; Fe, 22.2; SiO_2, 49.8; other con-

[1]*Mining Magazine*, XII, 196.

stituents, 3.1. (II), S, 35.1; As, 1.2; Fe, 29.4; SiO_2, 31.4; other constituents, 3 per cent., Fig. 2 gives the temperature and rate of elimination of sulphur.

In large furnaces, roasting is most rapid when the effluent gases contain no more than 2 per cent, by volume of SO_2 or 4.4 per cent. by weight (being 25 lb. of air per pound of SO_2 or 50 lb. per pound of sulphur eliminated). When the percentage of SO_2 reaches 4 per cent., roasting becomes slow; at 8 per cent. it is very slow; and at 9 per cent., the reactions cease. When so much air enters as to bring the SO_2 below 2 per cent., the charge is too much cooled. It would appear, therefore, that the Brückner furnace should be inefficient, since when the charge is burning briskly the air supply is too short, and at the end of the roast there is little SO_2 evolved, and hence too much air. The best furnaces would, therefore, seem to be those where the layer of ore is thin and extended, and where

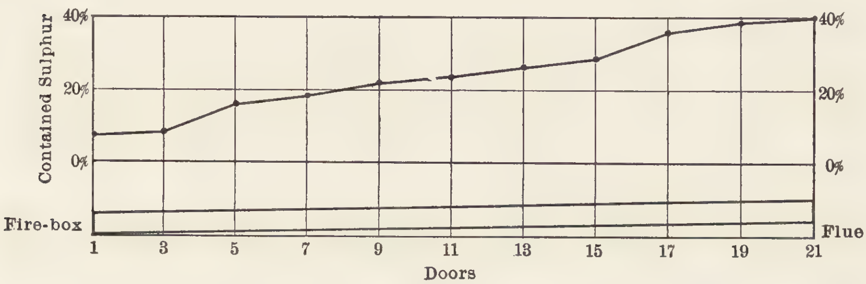

FIG. 3.—ROASTING IN EDWARDS FURNACE.

the SO_2 as fast as formed, is rapidly removed by the draft, as is the case with the Ropp, the Brown, and the Edwards furnaces.[1]

Mr. Warwick cites various points in favor of the Edwards roasting furnace as follows: It is compact and yet has large capacity. The heat is well conserved, and the stresses on the structure are not severe. The rabbles are always within the furnace, and so no heat is lost by cooling them, nor is the draft disturbed by the opening or closing of rabble-doors. The end-rabbles may be water-cooled, but are very durable, and, even if not so cooled will last for months. Labor, maintenance and supply costs are low, two furnaces at the Yampa plant costing $15.20 per month for oil, waste, etc.

The Edwards furnace gives a satisfactory roast, at a cost at Kalgoorli of about $1 per ton. In California the cost on copper pyrites is 20c. to 30c. per ton. Following, we give data of two Edwards furnaces at the Yampa works, Bingham, Utah.

[1] *The Mineral Industry*, XII, 257.

	Fixed Furnace	Adjustable Furnace.
Hearth dimensions	11x90 ft.	6x57 ft
Inclination	5.3 deg.	1.6 deg
Rev. of rabbles per min	1.5	2.0
Horse power	4.0	1.0
Daily capacity in tons	72.0	25.0
Sulphur in raw ore	....	26 to 39 per cent.
Sulphur in roasted ore	....	2.2 to 8.6 per cent.
SO_2 in escaping gases	....	4.3 per cent. by. vol.

Roasting Furnaces.

McDougall Roasters at the Washoe Plant.—In addition to the data regarding these furnaces given in The Mineral Industry, Vol. XII, p. 98, there is the following regarding their operation : At the beginning of the eight-hour shift the workman stops the furnace and removes the lumps of ore or crusts which have accumulated on the rabbles and on the roofs of the third and fourth floors. These lumps are sent to the blast-furnace as being well suited to smelting. Then, starting the furnace, the remaining lumps are taken out as they are swept round by the rabbles. This takes an hour. The side drop-holes, where the ore goes down to the next hearth, are situated in front of side-doors and can be conveniently poked down by a curved poker. The ore, collecting on the central shaft and at the central draft- or drop-hole, is easily removed because of the water-cooled surface to which it adheres. The dust escaping from the furnace is mostly caught in an overhead flue from which it can be withdrawn to closed-top cars set below. This is sent back to be again fed through. These overhead flues are of brick to withstand the corrosive effect of the escaping gases. The ore, in dropping from hearth to hearth, is showered through the ascending air which actively roasts it; but, at the same time, the air current carries off the fine particles of ore, making flue-dust to the amount of 3 to 5 per cent.

When a roaster is to be repaired, the feed is stopped, and the rabbles gradually sweep out and clear the hearth of ore. Upon starting again about two firings of wood are put on the bottom and on the fifth hearth. This is followed by coal. When the wood is gone the rabbles are started, as well as the feeding of ore, until it comes down to the fourth or fifth hearth. At the same time coal is added to the hearths as they heat up until they have become hot enough to start the burning of the ore. The heat creeps up, and the furnace gradually gets into operation.

The sulphur in the roasted ore is from 6.5 to 8.5 per cent., based upon an output of 40 tons daily per furnace composed of 36.5 tons fine concentrates, 1.75 tons of finely crushed limestone, and 1.75 tons of flue-dust returned from the flues. This tonnage may rise to 47 tons daily. The fine concentrates may be replaced in part by fines from first-class ore and from Wilfley-table concentrates.

In roasting, each hearth eliminates about 5 per cent. of the sulphur. The escaping gases from the upper hearths contain 2.25 per cent. SO_2 by volume. The temperatures of the hearths are as follows: first hearth 315 deg.; second, 600 deg.; third 900 deg.; fourth, 935 deg.; fifth, 960 deg.; and sixth, 860 deg. C. The calcines are removed to the reverberatory furnaces for melting down, still of a temperature of 420 deg. C. The rabble-arms and shaft are water-cooled, but this in no way hinders the roast. The raw ore will average 34 per cent. sulphur. The composition of the escaping gases, with the CO_2 from the limestone included, is as follows, the figures in brackets being by volume, the others by weight: SO_2, 4.95 (2.25) per cent.; SO_3, 1.45 (0.53) per cent.; O, 19.60 (18.45) per cent.; N, 73.85 (78.68) per cent.; CO_2, 0.15 (0.10) per cent; total 100 (100) per cent. This figures 32 lb. air per lb. of sulphur removed. The roasted product of the furnace by screen analysis gives the following: On 10-mesh, 9.8 per cent.; between 10- and 30-mesh, 25.3 per cent.; between 30- and 80- mesh, 30.7 per cent.; over 80-mesh, 33.4 per cent.

Blast Furnace Smelting.

The Blast Furnaces of the Washoe Plant.[1] Figs. 4, 5 and 6 show respectively a half-sectional and half longitudinal elevation, a transverse sectional elevation and cross-section of the blast-furnace building, and end elevation of one of the large blast-furnaces, 56x612 in. (51 ft.) interior dimensions at the tuyere level of which there are three.[2] The side-bosh is 8 in., making the furnace 6 ft. wide at the throat, and the jackets are two tiers high. There are two hearths occupying the two ends of the furnace, leaving a central bridge carried by water-cooled plates. The drainage is to sumps or crucibles at the front of the hearth, and the matte and slag discharge thence into two settlers or fore-hearths, each 16 ft. diameter and 5 ft. high. Above the upper tier of jackets comes the heavy mantel-plate, 2 ft. 3 in. high, having a sloping front, and being surmounted by apron or receiving plates 21. in. high, and sloping at an angle of 45 deg., these together making a hopper on which the ore slides into the furnace. There is a closed top and there are three down-takes to carry off the fumes to the dust-chamber. There are 88 tuyeres, 6 in. diameter, set at 14-in. centers, having jacket openings 4-in. in diameter.

The bustle pipe goes entirely round the furnace, and connects through two blast-gates to the blast-main. The slag is granulated by a flattened stream of water, and swept away along a semicircular cast-iron launder to the dump. The smoke from the blast-furnaces goes to a dust-chamber 40x280 ft.x20 ft. high, and provided with sheet-steel pyramidal hoppers,

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[1]*Bi-Monthly Bulletin*, A. I. M. E., July, 1906, p. 529.

[2]The above length applies to two of the blast-furnaces; the third, 56x1032 in. (86 ft), has three settlers, and three hearths with two intervening bridges, and has been completed since the above paper was written. The 51-ft. furnaces were made of two furnaces 56x180 in. thrown into one with the space between them, this space being bridged over as shown in the illustrations. See *The Mineral Industry*, XII. 101.

forming the bottom of the chamber about 6 ft. above the ground. Thus it is possible to pass under and reach any of the hoppers to remove its contents. It is connected to the large main flue by means of a flue 20x15 ft. The main flue, branches, and main stack are fully described in THE MINERAL INDUSTRY, XII, 115.

The furnace has seven feed-doors on each of the long sides, giving access to its entire length. The coke is dumped in from two-wheeled buggies, each 30 cu.ft. or 900 lb. capacity, and makes 10 per cent. of the

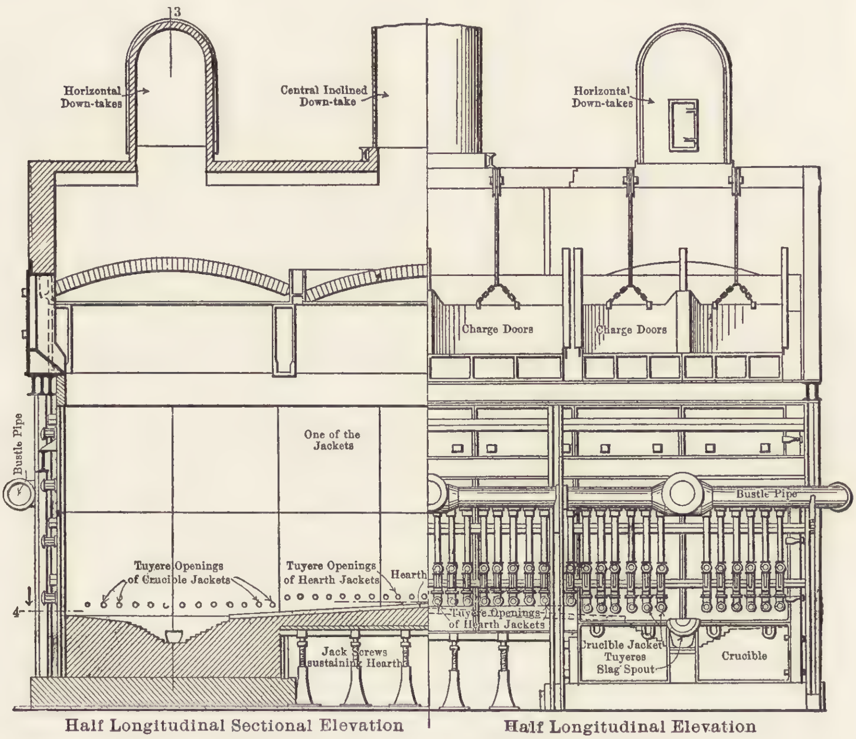
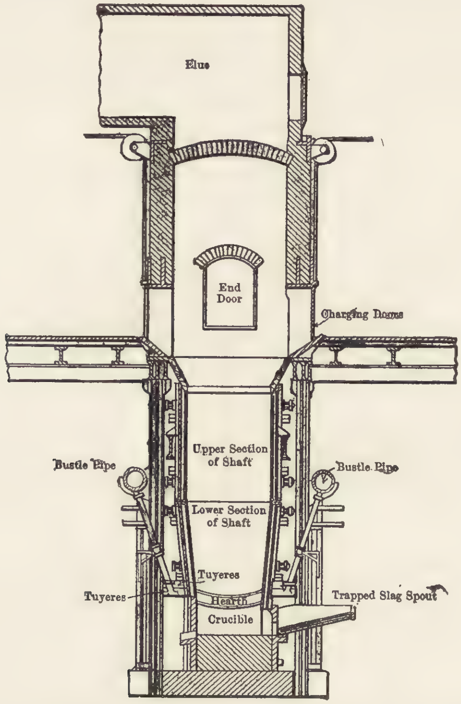


FIG. 4.—SECTION AND ELEVATION OF 56x612-IN. BLAST FURNACE.

charge. The charge is brought in by a train of 18 cars, each car holding 2.5 tons, and is dumped into the furnace, the hind end of the car being raised so that its contents are shot into the furnace.

The cars are loaded from the storage bins as follows: They are set at the coarse-concentrates bins, where the desired quantity is drawn off and weighed on track-scales beneath. The train is then run to the briquette-hoppers, and the soft freshly-made briquettes are dropped on top of the concentrates. These are overhead hoppers, each holding a ton of briquettes and with drop-bottoms which can be opened simultaneously. The train

then goes to get the first-class ore, the slag, and the limestone at their respective storage bins, and is now ready to be run to the furnaces. When set in place opposite the long side of the furnaces, the doors are lifted by



*F*IG. 5.—TRANSVERSE SECTIONAL ELEVATION OF 56x612-IN. BLAST FURNACE.

TYPICAL CHARGE, WASHOE WORKS.

Name of Ore.	Moisture.	Weight.		Cu.		SiO_2.		Fe and Mn.		CaO and MgO.		S.	
		Wet.	Dry.	Per Ct.	Wt.	Per Ct.	Wt.	Per Ct.	Wt.	Per Ct.	Wt.	Per Ct.	Wt.
Coarse concentrates....	3.0	1,550	1,500	8.7	130	22.0	330	27.0	405	......	......	35.7	535
Briquettes............	8.0	1,620	1,500	6.9	104	45.0	675	11.0	165	......	......	17.0	255
First class ore.........	3.0	2,060	2,000	8.0	160	50.0	1,000	12.1	242	0.5	10	15.4	308
Converter-slag.........	......	1,500	1,500	2.0	30	30.0	450	42.9	643	1.0	15	0.7	10
Limestone.............	......	3,500	3,500	......	......	5.0	175	1.0	35	51.0	1,785	......	......
Coke................	......	......	1,020	......	......	6.9	70	1.2	12	0.8	8	0.7	7
		10.230			424		2,700		1,502		1,818		1,115

Weight of slag, 2700÷0.40 = 6750 ℔., or 67 per cent. of charge.
Cu in. slag, 6750 by 0.33 per cent. = 22 ℔., or Cu_2O = 25 ℔.
Cu in. matte, 424—22 = 402 ℔., and the matte weighs 402÷0.40 = 1005.
Fe in matte, 1005 by 30 per cent. = 301 ℔., leaving for the slag 1502—301 = 1201 ℔., or of FeO 1544 ℔.
S in matte, 1005 by 23 per cent. = 231 ℔., and in slag 26 ℔., or in both 257 ℔.
S volatilized 1115—257 = 859 ℔., or 77 per cent. of the total sulphur.
From these data we have in the slag SiO_2 2700 ℔. (40 per cent.); FeO, 1526 ℔. (22.5 per cent.); CaO, 1818 ℔. (26.9 per cent.); Cu_2O, 25 ℔. (0.4 per cent.); S, 26 ℔. (0.4 per cent.); other bases, by difference, 9 8 per cent.

compressed-air power, an air-hoist is attached to the rear of the car-body, which is raised and the charge of 2.5 tons slides into the furnace.

The escaping gases vary in temperature from 100 deg. C. in a freshly charged furnace to 560 deg. C. just before a fresh charge is put in.

Herewith is given a typical charge, which expresses briefly the weights and composition of the charge and out-products of the furnaces. There is a pressure of 40 oz. per sq.in. because of the slagging-up of the tuyeres,

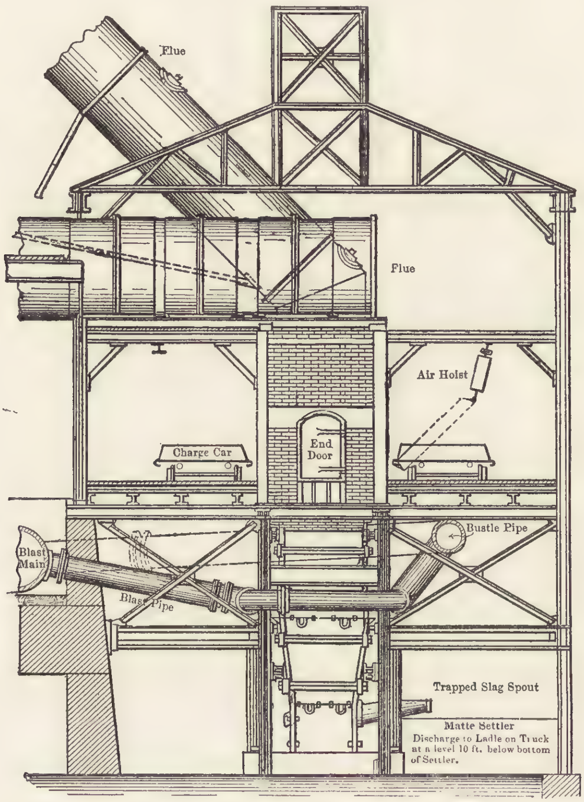


FIG. 6.—CROSS-SECTION OF FURNACE BUILDING AND END ELEVATION OF 56x612-IN. BLAST FURNACE.

which have to be frequently punched to keep them open. The sulphur loss by volatilization, as will be noticed, is 77 per cent., due to the large amount of air supplied; and this shows itself by much overfire, and by the violent agitation of the surface of the charge.

Adjoining the fore-hearth, and 10 ft. below the level of its bottom, is a track on which the matte-ladles are run in by a locomotive, and the fore-hearth, when full, tapped into them. The flow of matte is arrested by a

stopper rod or dolly having a 6-in. button-head armed with a conical plug of clay, the tapper pressing it with all his force, while his helper taps the end of the dolly to compact the clay into the tap-hole.

The advantages of the large 51-ft. furnace over the smaller ores of the same aggregate length are: Saving of fuel, since the large furnace needs but 10 per cent. as compared with 11 per cent. in the smaller ore. Saving of jacket-water, since, while the capacity has increased 3.4 times, the radiating perimeter has increased but 2.8 times. Quick and large discharge of matte, resulting in more rapid change in the contents of the internal crucible, and in the fore-hearth, whereby both places are kept hotter and more open. The slag in the larger furnace has a temperature of 1140 deg. C., so that even a 40-per cent. silica slag is quite fluid. Decrease of incrustation due to the lessened wall surface, less end-wall and corners, and at the same time side-walls so long as to be easily barred off. The high blast used (40 oz.) causes at least 3 per cent. of flue-dust, even though the ore has before feeding to the furnace been screened over a 0.375 in. trommel. Elasticity of operation, so that while the furnace is in use a jacket can be removed, the crust on the inside preventing the falling out of the materials of the charge. Also, in case any part of the furnace has become dead, this dead charge can be removed while the rest of the furnace is still running. Thus the operation of the furnace might be considered as being endless. Variations in the composition of the slag. The part most exposed to corrosion by the slag is the central bridge. To lessen this action, the charge at this portion of the furnace can be made more silicious so as to lessen its corrosive action, while at the ends it may be harmless when more basic. Again, to smelt out a crust, the slag at one of the furnaces may be quite basic and corrosive in its action upon the crust, while the rest may be working well on a normal slag. The cost of labor is naturally less on the large furnace so that for the prime cost we may take for the small furnace $1.08 per ton against $0.94 per ton on the large one—a saving of 13 per cent. The initial cost of a large furnace, with 3.8 times the capacity, would be but 2.5 to 3 times that of the smaller one, besides which the building containing it would be proportionally smaller.

Smelting at the Copper Queen Works.[1]—These works are situated at Douglas, Arizona, 31 miles from the mines, and treat ores from Bisbee, silicious ores from Globe, and custom ores. They occupy a space of a mile long by a little less than a half-mile wide. The Copper Queen ores from Bisbee vary as follows: SiO_2, 5 to 50 per cent.; S, 0 to 45 per cent.; Fe, 0 to 45 per cent., and from low-grade to high-grade copper, but without lime. Many are clayey in character, varying from 2 to 35 per cent. in Al_2O_3, and, when wet, will not run through the ordinary chute. They are accord-

[1] *Eng. and Min. Journ.*, LXXXII, 242.

ingly placed in long parallel pits or yards being bedded together. At the mine the ore is loaded into six 50-ton side-dump cars at a time, and on a seventh, every tenth mine car is unloaded, thus serving as a sample for the six cars. At the smelter this car goes to the sampling works, the others going to the bedding-pits. There are four of these pits or beds, each being 800 ft. long, 40 ft. wide, and 11 ft. deep below the track level, and having a capacity of 70,000 tons. The pit is divided into four train-length sections of 200 ft. each, and the ore of each train goes to its own section, of which an account, kept in a diagram-book, gives a record of the ore of each section. The ore piles up chiefly near the track, and is leveled by means of a spreader or plow mounted on a car, having outriggers and an inclined deflector or share, which plows or scrapes the ore, pushing it over into place and leveling the top surface. This share has a vertical movement, so that it can be varied as the pit fills. The spreader, devised by George B. Lee, the superintendent, has taken the place of 75 Mexican laborers. When these beds or pits were first used there was much trouble because of the varying character of the ore; but, by attention to regularity of delivery and to spreading the beds, they have been made uniform, as also has the performance of the furnaces. It has been the custom to add the lime-rock upon the top of these beds in the right proportion for fluxing, but this is being superseded by the weighing of it as the charge-cars are on the way to the furnace, and, in fact, as the coke is now weighed. The charge is loaded into side-dump charge-cars standing upon the charge-track by a steam shovel which runs on permanent track upon the floor of the pits, 11 ft. below the rails upon which run the charge cars. The steam shovel dippers will hold ¾ yd. It has been found that in thus measuring the charge it will vary no more than 2 per cent. in weight. The charge-cars weigh 5000 lb. with 5000 lb. capacity, but this has been found too heavy a charge, and it has accordingly been diminished to 2500 lb. Trains of 20 cars are used, handled by 13-ton electric locomotives.

There are 10 furnaces, five of 18 ft. long and five of 20 ft. They are charged alternately upon the long side with three charges (7500 lb.) for the smaller, and four charges (10,000 lb.) for the larger ones. The surface of the charge is carried 6 ft. below the feed door, making a smelting column of 10 to 11 ft. They are built with two tiers of jackets with side-boshes of ¾ in. to the foot. For each of the large furnaces there are 40 tuyeres each 4 ft. in diameter, one tuyere pipe serving two tuyeres, and having a rotating valve to close it. The ten furnaces are arranged on one center-line 15 ft. apart and with the long side parallel to that line, with one oval settler 10x23 ft. between each pair of furnaces. The lower half of the settler is laid with chrome brick (neutral brick); the upper, with fire-brick. The slag from the settlers is caught in 18-ton slag cars handled by an electric locomotive. There are two cars to each settler, one being in position while

the other is dumping. These cars enter the building at right angles to its length, the tracks forking to the two sides of the settler.

The down-takes are carried over the furnace-building roof to a spacious dust-chamber on the farther side where the dust can quietly settle. It consists of a steel frame with brick walls roofed with reinforced concrete supported on steel trusses. Its floor, a series of V's, is made of fire-proof tiling. The building is 648x160 ft. with a crane aisle 60 ft. wide, equipped with three 60-ton five-motor cranes which travel the entire length of the building. There are 8 converter-stands and 36 shells, each 8x10.5 ft. These are lined with silicious ores from Globe, Arizona, and from Nacozari, Sonora, to which has been added sufficient clay for bond, mixed in three wet-pans. The lining is rammed by power-rammers. The converters are operated by hydraulic pressure given by a plant of three triplex pumps, motor-driven, which deliver into compression tanks at 275 to 300 lb. pressure. The eight stands are operated by means of five blowing engines, each of 650 cu. ft. capacity per min., and one engine of 1500 cu. ft. capacity.

Each blast-furnace has its own blower and blast-main so that it has independent regulation. The 18-ft. furnace takes a No. 9 blower; the 20-ft., a No. 10. The engines which drive them are set staggered, a novel arrangement which saves room, the blower of one set being placed back of the fly-wheel of the next. By means of a header at each furnace it is possible to connect the various blast-mains into one, if so desired. A two-stage compresser of a capacity of 2000 cu. ft. free air per min. is used to supply air for the tamping machines, to operate furnace doors, and to pump water.

The sampling mill is a steel-and-brick building 65x165 ft. It contains three crushers and six sets of rolls, set along one side of the building, all on the same level. There are two Vezin automatic sampling machines. This simple arrangement is necessary because of the clayey character of much of the ore, which precludes the free use of elevating and conveying machinery. Indeed it is at times necessary to dry the ore by spreading it in the sun before it can be crushed and rolled at all. Besides the regular ore-beds or pits, there is a large ore-storage of 70,000 tons, held as a reserve in case of necessity.

Britannia Smelting Company.—The works of this company at Crofton, Vancouver Island, B. C.[1] were erected at the water-side to treat ore brought 8.5 miles by narrow-gage railway from the Lenora mine, as well as custom ores, coal, and coke, arriving by water. The ore comes in hopper-bottom cars by two overhead tracks to 14 ore-bins of 300 tons each, placed below them; the coke is unloaded from a track between the storage bins and the furnace buildings, and the copper is removed at a low-level track

[1] *Brit. Col. Min. Record,* XII, 371.

adjoining the converter building. The sampling mill is 26x32 ft.x84 ft. high framed with 12x12 in. posts, double boarded outside, and roofed with "Malthoid" roofing. It is provided with two 10x20 in. Blake crushers, a set of Davis 12x18 in. rolls, two Snyder automatic samplers, two belt elevators having 16x6 in. buckets, and, in the finishing room, three sampling grinders, bucking-plates and steam coils for drying. All this machinery is run by a 45-kw. direct-current motor.

At the end of the other elevated track is the briquetting plant, 72x35 ft., supplied with a Chambers No. 7 brick machine of a capacity of 60,000 to 70,000 brick per day. Adjoining is a drying shed, 104x78 ft., supplied with 5000 sq. ft. of steam coils for indirect heating, the air being drawn over the coils by a No. 6 Sturtevant fan operated by a 7-kw. motor, and

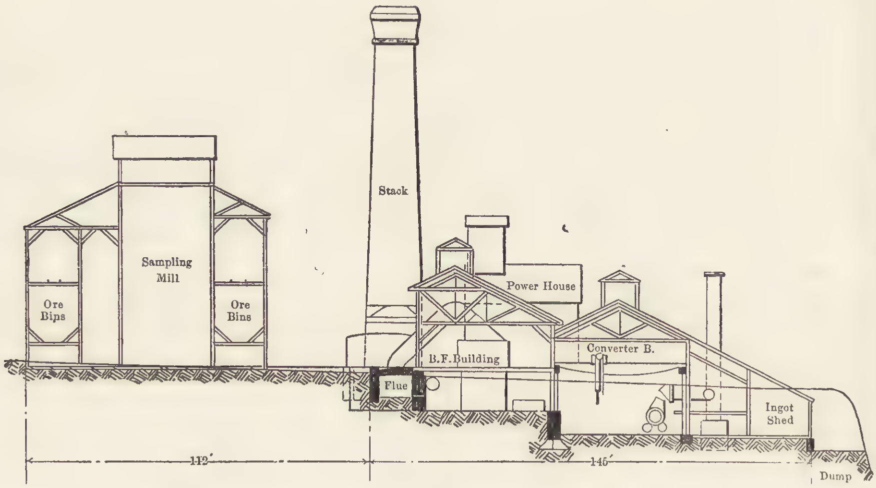


FIG. 7.—SECTIONAL ELEVATION OF BRITANNIA SMELTING WORKS.

delivering through a 12-in. galvanized-iron main to 6-in. branch pipes running the length of the drying tracks, the branches being perforated every 18 in. with $\frac{3}{8}$-in. holes. A boiler room, 18x35 ft., has a 100-h.p. boiler to supply steam for the drying room. The brick are piled in kilns and roasted, or else sent direct to the furnaces. Lump ore is roasted in nearby roast-yards. The dust-flue, 10 ft. wide, 12 ft. high, and 200 ft. long, ends at a dust-chamber 24x40 ft.x20 ft. high, and next to this comes the main stack 120 ft. high and 12 ft. interior diameter, which stands on a concrete base. The blast-furnace building, 73x45ft., is 50 ft. distant from the storage bins, and contains one 65-ton cupola for remelting matte, one 42x160 in. blast-furnace, and one Garretson combined smelting and converting furnace. The slag is generally granulated by water, but, in case of accidents, several large slag cars are provided to take the slag

flow during repairs. The slag floor of the furnace building is 14 ft. lower than the charging floor, and the converter floor still 8 ft. lower, so that it was necessary to excavate 17,000 cu. yds. to obtain the advantage of transferring matte from the settlers to the converters by gravity.

The converter building, 73x65 ft. contains one converting stand, with four shells of the trough type, 84x126 in., and operated by hydraulic power through an accumulator. The building is arranged for a traveling-crane, but, at present, the shells are transferred by a hydraulic truck. The converter is directly charged from the settler by launder. The fines from the converter go by a hood and flue into a water-sprayed chamber, where the dust is settled, the smoke then passing to a sheet-steel stack. Back of the converter is an extension of the converter building, 20x25 ft., to accomodate the hydraulically operated rack and pinion, which moves the blister-copper mold. The lining material is tamped into the converter shells by a pneumatic rock-drill tamper operated by a small air compressor.

The engine room, 50x60 ft., contains an Allis-Chalmers engine, 18x36x42 in., driving two Connersville blowers (one a No. 8, running 125 r.p.m.; the other a No. 5, at 165 r.p.m.) connected to a main blast-pipe 54 in. diameter. The engine also drives a 110-kw. Westinghouse direct-current generator, which furnishes current for the motors driving all outside machines. There is also a blowing engine, 16x36x42 in., with a capacity of 9000 cu.ft. of air per min. at 15-lb. pressure for the converter. In the boiler house, 40x 48 ft., are three 100 h.p. horizontal return-tube boilers, with foundations for a fourth when needed.

Old Dominion Smelting Works.[1]—At the mouth of the shaft are located three storage bins, of 1460 cu.ft. each, for the storage respectively of first-class ore, concentrating ore, and lean-ore for the converters. From the bins the ore is drawn upon a 30-in. conveying belt, and passed through a trommel with 2-in. screen openings, the over-size going to a crusher and thence to a picking belt. Here, from the concentrating ore, one man sorts out first-class ore, talc, fragments of wood, and barren rock. The sorted ore is again crushed finer in an ore-breaker, then goes by elevator to rolls, and is finally carried by troughed conveyor to a bin house where a tripper delivers the various ores to the proper bins, eight in number, and each of 4150 cu.ft. capacity. From two of the bins on one side, concentrating ore is drawn by automatic feed to a conveying belt, and thence it goes to a concentrating mill. On the other side it is drawn from six bins in cars to go to the converter and smelter ore bins. The cost of these operations is 7c. per ton, the ore being handled at the rate of 75 tons per hour. The concentrates pass to the smelter storage bins.

Three of the furnaces are 42x180 in., while the new No. 4 is 50x196 in. This is provided with side-jackets 33 in. wide by 14 ft. high instead of 96

[1] *Eng. and Min. Journ.*, LXXXII, 8.

in. by 14 ft. high as in the older furnaces. The narrow jackets are expected to warp less under the action of the heat. The ore column is carried at 7 ft.

The ores are coarsely crushed first-class ore, concentrates from Nacosari and Cananea, Mex., custom ores, often containing chrysacolla and carrying 70 per cent. silica, and slag from old dumps carrying 4 per cent. Cu. Nacosari concentrates carry 15 per cent. Cu., those of Cananea 8 per cent. The matte produced assays 50 per cent. Cu., while the slag contains SiO_2 38 per cent., Al_2O_3 8 per cent. and CaO 11 per cent. The large settlers are lined with magnesia and fire-brick packed by a sand-clay tamping. The flue-dust is briquetted with a binder of lime, but trial is being made of concentrator slimes carrying 1.5 per cent. Cu.

Giroux Hot-blast Furnace-top.[1]—In this, as in some other systems, the air intended for the blast-furnace is heated by the escaping waste gases of the furnace. The Giroux method consists in lining the furnace-top with a series of vertical sheet-steel pipes. Three pipes, each 30 in. in diameter by 20 ft. in hight, occupy each end of the furnace, while at each side there are seven, 27 in. in diameter by 15 ft. in hight, which begin immediately above the feed-doors. The latter expand, as they go upward, to 27x45 in. Thus the pipes pretty well fill the interior of the furnace-top. These pipes are connected in series alternately at the top and at the bottom. The air from the blower, entering at the bottom of an end-pipe, leaves at one at the other end, which connects to the bustle pipe and to the tuyeres. The increase of temperature will, it is estimated, be 400 to 500 deg. F.

Kiddie Hot-blast System.[2]—This consists in utilizing the waste heat of the escaping gases (400 to 600 deg. F.) for the purpose of obtaining a hot blast for the smelting furnace to which it is attached. The course of the air to be heated is from the blast-main into a flattened absorption pipe 5 ft. 5 in. high by 9 in. wide. The air traverses this pipe, taking up the heat from the escaping flue-gases. Large surfaces are presented to be warmed by the gases, so that a temperature nearly approaching them can be attained.

The general average of ore at the Tyee smelting works, Victoria, B. C., where the system is in use, is as follows: Cu, 4.08; Fe, 10.5; Zn, 7.4; SiO_2, 13.5; Al_2O_3, 7; $BaSO_4$, 37.6; CaO, 2; and S, 6.3 per cent., with Ag, 2.7 and Au, 0.131 oz. per ton. With cold blast the following charge was used: 63 per cent. of the above ore roasted to 6.3 per cent. sulphur; raw ore containing 16 per cent. sulphur together with 14 per cent. of coke. There was produced a matte of 42 per cent. copper. Using hot blast, the charge was 37 per cent. roasted ore, and 63 per cent. of raw ore, the coke being 9.4 per cent. of the charge. The matte contained 42 per cent. of copper.

[1] *Eng. and Min. Journ.*, LXXXII, 698.
[2] *Eng. and Min. Journ.*, LXXXII, 598; Canadian Patent, 96,665.

These results indicate that, with the same tonnage put through in both cases, there was a saving of 33 per cent. in the fuel, while with the hot blast, carrying a much greater proportion of raw sulphide, it was possible to get the same grade of matte as with cold blast.

In general it is claimed that a furnace working hot blast can be kept in better condition than one using cold blast; that it oxidizes sulphur, iron, and zinc better; and that there is a greater saving of coke as compared with cold blast. To lessen the velocity of the escaping gases so that they may better settle out flue-dust, and give up their heat to the heating-pipes, the draft is regulated by a damper in the stack.

Use of Wood in Copper Smelting.[1]—At the Mitchell Mining Company's mines, Mexico, Mr. Mitchell has successfully substituted for coke blocks of wood which have first been dipped in a paste or wash of silica and clay, the mixture being of such a consistency that the blocks could be readily dipped in it, and on being withdrawn would retain a thin coating of the paste. It was found that when so treated they did not catch fire upon the top of the charge as the un-treated ones did, but passed down to the fusion zone. It has thus been possible to get a concentration to 38 per cent. in the matte, as compared with 20 to 25 per cent. with un-treated wood. It cannot however be compared with coke for smelting purposes; and its use is only justified in a country where wood is abundant, and where the price of coke is prohibitive.

H. F. Collins in 1901 used at Santa Fé, Chiapas, Mexico, a mixture of coke and wood in a blast-furnace, and found that in place of 15 per cent. coke, he could use 11.5 per cent. together with 3.5 per cent. of wood in sticks, but could not exceed this proportion on account of the difficulty of hot-top.[2] Contrary to the experience of Mr. Mitchell, he found that the grade of the matte was increased by 4.5 per cent. in copper, and to the high tenor of 50 per cent.

Pyrite Smelting.

R. C. Alabaster and F. H. Wintle are of the opinion[2] that the practice of the Tennessee Copper Company is the most satisfactory, with the use, however, of furnaces of 40 to 44 in. wide. The final concentration would depend on the further treatment of the matte, i.e., if to be converted, 40 per cent. matte will serve; but, if it is to be shipped, the matte should be concentrated to 60 or 65 per cent. The slag from the concentration furnace should be returned to the ore furnace, and a clean slag must be obtained in the latter.

Within certain limits, a furnace makes its own slag as regards silica and iron, so that, if the silica is increased, it calls for more iron, which iron is

[1] *Eng. and Min. Journ.*, LXXXII, 700.
[2] *Trans.* I. M. M., XV, 276.

robbed from the matte, whose tenor in copper is thus increased. They consider a slag of Ca SiO_3+Fe SiO_3, containing, for example, 40 per cent. SiO_2 to 44 per cent. of FeO+CaO as ideal, and in which CaO may be replaced by FeO in ratio of their atomic weight.

They think alumina greatly increases the infusibility of the charge, and cite conditions at the Copper Queen, Bisbee, Arizona, under which the slag carries 14 per cent. of Al_2O_3, and where as much as 19 per cent. of fuel is used in smelting. Where, as in concentrating matte, alumina is absent, the tendency to form crusts is decidely lessened. Zinc does not interfere either in the slag, or by the formation of crusts, being largely volatilized (78 per cent. in charge I). The 1.7 per cent. left in the matte is expelled in converting.

Hot blast has been tried, but later abandoned at the works visited by the authors. Thus, at Mount Lyell, Tasmania, it was given up after four years trial; at Ducktown, Tenn., Mr. Freeland tried a U-pipe stove but reverted to cold blast; at the Copper Queen, where furnaces were designed for hot blast, they decided to revert to the cold-blast furnace. Advocates of hot blast claim that cold blast tends to give less bottom heat, and to produce a crust near the tuyeres; but, as a matter of fact, crusts are chiefly found several feet above. Again hot blast calls for a more cumbersome plant, great wear and tear, and additional labor costs. As against these claims, it should be remembered that pyritic smelting depends upon the principle that the sulphur and the iron ore are being oxidized, and that, if coke can be given up (using in place of it preheated air), we have an atmosphere free from the deleterious gases evolved in the burning of the coke.

At the Tennessee Copper Company's works pyrometric tests of a sulphide-ore furnace using 2.5 per cent. of coke, as compared with one carrying a charge of roasted ore and 12 per cent. of coke, showed a higher average temperature by 100 deg. C. in favor of the former.

In comparing pyritic smelting with that of roasted ore, the following points are to be considered:

Fuel Economy.—At the works of the Tennessee Copper Company, when smelting roasted ore, the coke consumption was 12 per cent.; and against this we find but 3.5 per cent. needed for both stages of pyritic smelting. At Ducktown the coke consumption is but 4.4 per cent. as in former practice.

Labor.—In pyritic smelting the labor in the two-stage process is no more than it was for smelting roasted ore, while the cost of roasting (40c. per ton) has been saved. A furnace crew consists of three feeders and three tappers, whether on roast or raw sulphides, and, with a smoothly working furnace two tappers should suffice.

Output.—It is urged against pyritic smelting that the output is dimin-

ished, and this is indeed so. Thus the Tennessee Company's furnace, smelting 500 tons of roasted ore daily, now only puts through 400 tons of raw sulphides. (It has later increased this, often to 480 tons.) Again, at Ducktown, the tonnage was decreased 30 per cent., a 20 per cent. matte being produced. This is due to the furnace performing roasting as well as smelting.

Furnace Manipulation.—Pyritic smelting needs careful charging to prevent the formation of crusts, which rapidly grow upon the sides and corners of the furnace; and to get rid of them, when they do form, sometimes necessitates the stopping of the furnace, resulting in a decreased output. Mr. Olden, in commenting on this, had observed that fine ore conduced to crusting, as also insufficient punching of tuyeres—a duty that the furnacemen naturally neglected.

To work off a persistent crust, which tended to form on one side of a furnace, it was customary to close two tuyeres on that side, so that the crust above would burn out. Then these were opened and two adjoining ones closed, and so on, until the crust had disappeared. This, together with most careful feeding, removed the crust in the course of six or eight hours.

Coke is useful in pyritic smelting because: (1) it keeps the charge open at the sides; (2) it assists in maintaining a high blast at the sides and over the tuyeres as it comes down to them; and (3) it eats up the crusts which naturally form upon the cold jackets.

As regards the hot blast he feels that, with less than 3 per cent. of coke, there is less free air to oxidize sulphides, and hence less FeO to take care of silica; so that difficulty fusible slags are likely to be formed, and, at the same time, the grade of the matte be increased.

To get good concentration and fusible slags, therefore, it is necessary with the same volume of air, to drive more slowly. This is illustrated in the furnaces of two plants. Thus, at the Tennessee Copper Company's plant the ore put through was 5.7 tons per sq. ft. of hearth-area per 24 hours, with a concentration of five into one, while at the Ducktown Sulphur, Copper and Iron Company's plant, with a tonnage of but 3.7 tons per sq. ft., there was a concentration of 10 into one.

In discussing the subject of smelting raw sulphides A. G. G. Wilson[1] gives an instance where, at the Highland Boy Smelter, the ore, because of its fine physical condition, was not suited to blast-furnace treatment. This ore, containing Fe, 26; SiO_2, 37; S, 30; CaO, 5; and Cu, 3.5 per cent. together with gold and silver, was accordingly treated in reverberatory furnaces after preliminary roasting, giving at one smelting a matte of 33 per cent. Cu, which went direct to the converters. He considers that for pyritic smelting the quartz ores should be not coarser than 1.5 in., and

[1] *Trans.* I. M. M., XV, 290.

the sulphides not less then fist-size. For charges where silicious slags of over 40 per cent. SiO_2 are run, he considers that hot blast is essential. He had smelted thousands of tons where he had carried a slag of 46 per cent. silica, and had noted there a decided tendency to chilling at the tuyere zone. Punching only aggravated the trouble; and the only way was to keep four or five of the tuyeres, out of the eighteen, plugged and in process of burning out, in order that in a few hours they should open bright. Here hot blast should surely have greatly assisted in keeping the tuyeres in good condition. This should be done by utilizing the heat of the waste gases in heating the air and this supplemented by the hot-blast stove. That hot-blast had been abandoned at Mount Lyell and at Ducktown, where the slags carry no more than 36 to 33 per cent. SiO_2, is no argument against its use with slags high in silica.

For low-grade copper ores it would seem that two-stage smelting was necessary; but for ores carrying 5 per cent. Cu it should be possible to make a 25 per cent. matte which could be bessemerized according to the Baggeley process as now practised at Butte. This would require a larger converter plant, but it would do away with the blast-furnace matte concentration.

The questions of slag composition will depend upon the quantity of silica, iron and earthly bases available; and the latter need be no more than 10 per cent. So long as the slag is fluid, other characteristics cut but a little figure. The iron set free as ferrous oxide to enter the slag will depend upon having but little or no carbonaceous fuel and an excess of air.

G. H. Blackmore found at the Great Cobar copper mine that his furnace produced a better grade of matte and ran faster on cold than on hot blast. The ore ran about 12 per cent. S, and the hot blast was 260 to 315 deg. C. At the Nymagee mine, where the ore carries 18 per cent. S, he stopped the use of hot blast and increased the smelting column from its former hight, 5 ft. 6 in., to a new hight of 11 ft. 6 in. The grade of the matte remained the same (10 to 15 per cent. Cu), but the furnace produced weekly 220 tons more than with hot blast. He considers the whole secret of successful copper ore-smelting to lie in a large furnace, a deep ore-column and a large volume of air from a positive source, such as a blowing engine, the rotary blower being, for high pressure, ineffective. He gives for a proper-sized furnace, 240x48 in. at the tuyeres, 16 to 18 ft. ore column, and 30,000 cu.ft. of free air per min. delivered by a blowing engine. Such a furnace should put through daily 450 tons of ore carrying no more than 20 per cent. S. For ores of 30 to 40 per cent. S, the tonnage should increase to 610 tons daily. He considers that it does not matter, so far as tonnage is concerned, what is the pressure of the entering air,

and holds that because of the slip or backward leakage of the air, the so-called positive blast blower is inefficient and misleading.

Robert C. Sticht, whose extensive experience at Mount Lyell and elsewhere has made him an authority on pyritic smelting, gives the following views of the theory and practice of smelting.[1] He holds that the dissociation of the pyrite is gradual, and at first to magnetic pyrite, and, that by the time matte has been formed, the elimination has gone so far that we have not the monosulphide FeS, but rather FeS holding reduced Fe in solution; thus, $4\ FeS + Fe = Fe_5\ S_4$, and in cooling the Fe separates out as such.

The ascending gases progressively absorb sulphur vapor. At the middle zone when FeS (formed from the pyrite) is melting (925 deg. to 950 deg. C.) the burning to FeO takes place, the pyrite having been gradually dissociated above. A sample taken at 7 ft. down from the surface of the charge (or at the middle zone) still showed that the sulphur was being eliminated. All this sulphur is eventually carried to the sufrace of the charge and there burns upon coming in contact with the ore which enters at the charge doors. Were the surface of the charge cool enough, sulphur would condense there. However, in good practice, this need not occur, even when the surface appears dark. This is a better way of running the furnace than to permit it to become red-hot on top. At Mount Lyell the ores are free from zinc or lead, and therefore crusts or accretions do not form on the jackets. When the furnace is working well the top of the ore column may be about at the melting point of sulphur (250 deg. C.), while above, at the charge door, the burning of it as it meets the air would give a higher temperature. Below the preparatory zone silica and lime are being transformed into slag, the silica being the last so to change. This unmelted material keeps the charge porous and open, and hence must not be too fine when charged, since it would hinder the escape of the rising gases and the downward movement of the forming FeS. Thus, instead of the ore column resting upon a bed of burning coke in the crucible, in coke smelting, it rests upon a network of quartz pieces which have thus far escaped slagging off, this network reaching from the crucible upward to the zone where the iron sulphide begins to fuse. At the sides this network is moving more or less slowly, and sustaining the portion which is descending regularly. The CO_2 from the combustion at this point of the coke is evolved actively, and at the places where the melting of the FeS occurs. This point is 5.5 to 6. ft above the tuyeres, or, with the 12-ft. smelting column at Mount Lyell, at 6 to 6.5 ft. below the surface of the charge. Mr. Sticht thinks that this furnace hight might well be increased to give a column of 16 or 17 ft.

He thinks that the amount of bosh cuts but little figure, since the charge

[1] *Metallurgie*, Feb. 22, March 8, April 8 and May 8, 1906.

will adjust its bosh to its needs as the result of the hanging or rather the hindered movement at the sides of the silicious network of partly sintered quartz mixed with slag. This network contains no lime or raw sulphides, which have become incorporated already into the slag and matte, and these latter have completely flowed away, leaving the silicious skeleton permeated by cavities and fissures. When the furnace gets in bad shape it is generally easier to clean it out and start again, which can be done in two days, rather than to try to burn it out with coke, and, accordingly, furnace runs average only a month, so that one can count on continuously running four out of five furnaces.

The composition of the furnace gases, whether two, three, six, or seven feet below the top, is approximately the same; and the author, therefore, concludes that the region or focus of slag-formation and of oxidation of the FeS is identical, that it is higher up in the furnace than in coke smelting and that it is limited in extent. This is also the place of formation of the CaO and Al_2O_3 silicates. Since slag rapidly forms in the region, we have vigorous reactions which seem necessary for successful pyritic smelting, and this is attained by an increased supply of air. These reactions are intensified by the heat of formation of slags, i.e., the silicates of FeO, CaO, MgO, and Al_2O_3.

The concentration of the copper in the matte depends upon the sulphur available to form it, or conversely upon the amount of FeO which is found. This latter again depends upon the presence of silica to combine with it, and hence, there should be free silica ready to unite with the just-formed FeO in the focus already described. Silica united to Al_2O_3 or to CaO or MgO is satisfied and is not, like the silica in a quartz ore, available to unite with FeO. Hence, where high concentration is desired such bases should exist in limited quantity in the charge. If the matte-fall is to be decreased, it should be done by adding straight silicious ore, and this is more effective than lessening the quantity of pyrite, which indeed would be the same as lessening the intensity of the reactions arising from the formation of FeO and slag.

Practice at Keswick, Cal.—According to Lewis T. Wright[1], in the earlier practice at Keswick the charge consisted of coarse heap-roasted ore, fine briquetted reverberatory-roasted ore, raw matte, slag and fluxes and with 9 to 10 per cent. of coke. With this the furnaces ran regularly, producing a matte of 40 to 45 per cent. Cu. Later the coarse or lumpy ore was charged unroasted, and a semi-pyrite smelting was performed. When pyrite smelting was first attempted, 9 or 10 per cent. of coke was used, and it was possible to produce a matte of not more than 12 to 15 per cent. of copper, which was hot and corrosive in its action on the furnace. On this account the practice was abandoned for a time. It was found,

[1] *Min. and Sci. Press* XCII, 124, 237

however, that the use of hot blast was an aid in the semi-pyrite smelting above described, and that the furnaces ran one-fifth faster, used one-fourth less coke, and produced a 25-per cent. higher grade matte than was possible with cold blast.

Satisfactory smelting was now done in two stages. In the first stage a charge was composed of roasted fine, 26 per cent.; coarse unroasted ore, 35 per cent.; flue-dust, 10 per cent.; slag, 15 per cent.; fluxes, 14 per cent.; and this was run with 4 per cent. of coke, giving a first-matte of 31 per cent. Cu. At the second stage 27 per cent. of the first matte was smelted with 31 per cent. of roasted fine, 9 per cent. of coarse sulphides, 13 per cent. of slag, and 20 per cent. of fluxes, using 5 per cent. of coke, and producing a matte of 53.5 per cent. Cu.

Finally regular pyrite smelting was again adopted in two stages. In the first stage the charge was sulphide ore, 74 per cent.; flue-dust, 9 per cent.; slag, 3 per cent.; fluxes, 14 per cent. With it was used 0.3 per cent. of coke only, while the matte produced contained 27 per cent. of copper.

In the second stage it was roasted ore, 9 per cent.; first matte, 19 per cent.; sulphide ore, 31 per cent.; flue dust, 6 per cent.; slag, 18 per cent. This was smelted with a little over 4 per cent. of coke, producing a matte of 42 per cent., which was well suited to converting. With 32 ounces of blast the furnaces were putting through charge at the rate of nearly 9 tons per sq. ft. of hearth area, which is very fast running, and apparently the condition needed for success. It has not, however, thus far been possible to dispense with hot blast. The amount of fuel for this purpose is equivalent to 1 to 1.25 per cent. coke.

Finally, pyritic smelting was managed at a single stage, dispensing with the roasting of part of the ore, and still obtaining a matte of 34 per cent. copper. To do this the following charge was used: Sulphide ore (coarse and fine), 66 per cent.; flue-dust, 8 per cent.; slag and furnace cleanings, 12 per cent.; fluxes, 14 per cent. With it was taken 1.8 per cent. of coke because of the larger proportion of inert material in the first stage of the pyrite smelting. The matte obtained in this single operation was of sufficiently high grade for direct converting, and the omission of another stage had simplified operations.

Mr. Wright thinks the Fe_3S_6 (FeS_2) breaks up as follows: It loses two out of the six equivalents at once, leaving Fe_3S_4 and this S_2 is burned, heating the escaping gases. Half-way down the furnace another equivalent is driven off, leaving Fe_3S_3 (FeS) which heats the charge at that point. Then, when the Fe_3S_3 is burned to FeO and SO_2, the final and most effective heating takes place. This gradual decomposition and burning is the cause of scaffolding, due to the incipient fusion at this high temperature of particles of the charge, and their sticking to the sides of

the furnace and to each other. He figures pyrite as replacing coke in the charge in the ratio of their formation-temperatures, so that a charge containing 50 per cent. of pyrite would evolve heat equivalent to 7.1 per cent. coke. Practically, when using hot blast, he found pyrite replacing coke as given in the accompanying table:

SMELTING AT KESWICK, CAL.

Using Cold Blast.			Using Hot Blast.		
Coke %	Copper in matte %.	Sulphur in charge %.	Coke %.	Copper in matte %.	Sulphur in charge %
6.8	20.1	24.5	0.3	26.8	74.3
7.1	19.3	22.8	2.3	39.6	58.5
8.5	24.6	19.5	2.7	42.0	51.5
10.2	41.4	13.6	3.4	46.3	45.7
16.3	60.9	7.7	5.1	52.4	32.1
15.7	69.4	6.8	5.2	53.5	24.9

In the accompanying table, when using hot blast, pyrite smelting was done, when using cold blast, roasted ore was added to the furnace burden. The grade of matte in copper increases as the pyrite is burned to FeO and SO_2, since the iron in oxidized form goes into the slag, and little combined sulphur is left for matte. Oxidation increased with the increase of air blown in. In matte-smelting of oxidized ores the ferrous oxide had an oxidizing action as follows: $5\ Fe_2O_3+FeS=7\ FeO+SO_2$, with a development of 710 calories per ℔. of pyrite present in the charge. The author calls the oxygen thus available, "solid oxygen."

A six months comparison of hot and cold blast with the coke consumption in each case corrected to the same grade of matte is as follows:

Percentage of Coke to Burden.	Percentage of Copper in matte.	Tons Smelted per sq-ft. Hearth Area in 24 hours.
Hot Blast 6.8	49.2	6.81
Cold Blast 9.9	49.2	5.87

The actual saving in fuel by the use of hot-blast is therefore 31 per cent., (nearly one-third), with an increased tonnage of 18 per cent. This is comparable with its advantage in iron smelting where, with a blast temperature of 485 deg. C., the furnace ran two and one-half times as fast, and reduced the coke consumption 30 per cent., as compared with the performance while using cold air.

Practically, when the furnaces are running on pyrite ore and without coke, 77.7 to 90 per cent. of the sulphur is eliminated as SO_2, while at the

same time, the burden per sq. ft. of hearth area should be 8 to 11 tons in 24 hours, and the sulphur present, in form of pyrite, should be as much as 30 to 33 per cent., so that at least 23 to 27 per cent. of the sulphur and 19 to 25 per cent. of the iron shall be oxidized in the smelting operations. In one instance the respective amounts were S, 28 per cent. and Fe 23 per cent., which, at 9 tons per sq, ft. per hour, would equal 3.50 ℔. S and 2.75 ℔. Fe per min. needing 4.28 ℔. O to oxidize them. Allowing an efficiency of 70 per cent. for the air needed to supply this oxygen, and 55 cu. ft. of air per ℔. of contained oxygen, we have 4.28 ℔. oxygen by 55 cu. ft. divided by 0.70 equal 326.3 cu.ft. of air per sq.ft. of hearth area, or, for a furnace of 44 sq.ft. hearth area, 14,357 cu.ft per min.

Ducktown Sulphur, Copper and Iron Company.[1]—This company has adopted the two-stage process of smelting. The ore charge, with 3.4 per cent. coke, is charged into a blast-furnace 204x48 in. by means of an automatic travelling belt of iron plates, which distributes it uniformly upon the surface of the charge. The outflowing matte and slag are separated in rectangular fore-hearths, 7x5 ft., and the 20 per cent. matte is tapped to ladles which pour it into flue-dust molds. The slag has the composition, SiO_2, 32; FeO, 38; CaO, 8; and Cu, 0.37 per cent.[2] The furnace requires 12,000 to 13,000 cu. ft, of air per min. at a pressure of 22 oz. per sq. in., and produces 280 to 300 tons daily.

The elliptical concentration furnace (60x30 in.) takes 4500 cu.ft. of air per min. at a pressure of 17 oz. per sq.in., and with a capacity of 60 tons daily. The matte, containing 20 per cent. of copper, is brought up to 50 per cent., using 8 per cent. of coke figured on the matte smelted, or a total of 4.4 per cent. on the ore smelted. The furnace is run with a smelting column of 7.5 ft. and with a hot top almost at the point of fusion. The slag contains: Cu, 0.6; FeO, 56.6; S, 12; SiO_2, 33.7; CaO, 2.0; MgO, 0.6; ZnO, 2.6; Al_2O_3, 2.2; MnO, 0.6 per cent.

Tennessee Copper Company.[3]—Illustrations of the furnace, 180x56 in. at the tuyeres, have been given in THE MINERAL INDUSTRY, XIV. It is to be noted that the crucible is water-jacketed, resting on a cast-iron sole-plate 1.5 in. thick. The spaces between the jackets are caulked with asbestos-felt. The discharge is continuous through a water-jacketed spout having an independent lip jacket, the lip being 10.5 in. above the top of the aperture of the tap-jacket, and high enough for the 40-oz. pressure employed. The settlers, of ½ in. boiler plate, are 16 ft. diameter by 5 ft. high, lined with chromite brick.

The process is two-stage, the ore being smelted to give a 10 per cent. matte (recent practice gives 12.5 to 15 per cent.), and this resmelted to a

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[1]*Trans.* I. M. M, Vol. V., 274.
[2]*Eng. and Min. Journ.*, May 2, 1903.
[3]R. C. Alabaster and F. H. Wintle, *Trans.* I. M M Vol. XV, 269.

matte of 45 per cent. (through one of 35 to 40 was common). The second matte is converted to blister-copper of 99.0 or 99.5 per cent.

The coke (100 lb.) is charged near the adjoining side of the furnace, and the charge of 4000 lb. dumped upon it, either from the adjoining door when full, or from the opposite door if the furnace is low, but, in either case, so that the charge shall fall upon the coke. In fact careful charging is essential to avoid the formation of crusts or accretions upon the sides of the furnace. It is thought that these crusts are due to the partial fusion of silicious material which, adhering to the sides on the furnace wall, builds up increasingly into a ridge. Fine ore, lodging upon the ridge, assists in the crusting. With careful charging crusts are not to be feared, as they can be burned off by raising the charge, the same to be again dropped when the furnace's sides have become clear.

The furnaces were run with (for copper smelting) a cool top, and with a smelting column of 12 ft. Later practice has been to keep the furnace evenly red-hot on top.

CHARGE SHEET I, ORE SMELTING FURNACE.

Name of Ore		Weight		Cu		SiO_2		Fe & Mn		CaO & MgO		S		Zn	
	H_2O	Wet	Dry.	Per Cent	Wt.	Per Cent	Wt.	Per Cent	Wt.	Per Cent	Wt.	Per Cent	Wt.		
Polk Co. Ore.......			1000	2.4	24	20.7	207	34.2	342	9.2	92	20.4	204	2.8	28
Burra Burra Ore...			3000	2.1	63	9.4	282	38.0	1140	8.3	249	30.3	909	4.9	57
Quartz Rock.......			700		......	97.0	679	....	......						
Coke..............		100			87		1168		1482		341		1113		85
			4700		Mn	+ Fe	for sl	ag	1075					877	Volatilized
					Mn +	Fe f	or Mat	te	407				918	41	S in Slag.
								Left	over f	or m	atte		195		

Slag SiO_2 37.2% 100.0%
MnO+FeO 44.1 34.3 Fe 92.0
MgO+CaO 10.9 29.0
S=1.3 3.5 of SiO_2 present
Zu= .3 0.8
Cu=0.2 = 0.5

Matte. Cu 10.2
Fe 53.9 64.1 per cent.
S 25.4
Zu—1.7

$\frac{Cu+Fe}{S} = 2.52$

Weight of slag=3140 lb.
Weight of matte=760 lb.
S Volatilized=79 per cent.
Matte fall=19.8 per cent.

Cu in slag...... 6
Cu in flue-dust. 3
Cu in matte..78
87

Cu in matte............... 78
Fe in matte...............407

Cu and Fe in matte..............485
Cu and Fe in matte (by calculation) 481

Flue-dust loss 200 lb.=4.3 per cent. of charge. Cu in flue-dust 3 lb.
Zinc in slag 6 lb.
Zinc in matte 13 lb.
Zinc Volatilized 66 lb. or 78 per cent.
85

Charge sheets Nos. I and II give the data regarding burden and out-products of the furnaces, the latter worked up as near as may be from the data given. Sheet I is for the ore-charge, and sheet II for the concentration charge of the matte resulting from the ore smelting.

CHARGE SHEET II, MATTE CONCENTRATION FURNACE.

Name of Ore.		Weight.		Cu		SiO_2		Fe & Mn		CaO& MgO		S			
	H_2O	Wet	Dry.	Per Cent	Wt.	Per Cent	Wt.	Per Cent	Wt.	Per Cent	Wt.	Per Cent	Wt.		
Matte			3000	10.2	306	....	......	53.9	1617	....		25.4	762		
Quartz Rock			1400			97.0	1358	....	......	....		....	......		
Marble			200			4.0	8	....	......	53.0	106	....	......		
Converter slag			400	2.0	8	30.0	120	55.0	220	1.0	4	1.0	4		
			5000		314		1476		1837		110		766		
					F	e for	slag		1712					564	
					F	e for	matte		125				605	41	S Volatilized
															S in slag.
								Left	over	for	matte		161		

Slag. SiO_2 35.0 =100.0 per cent.
FeO 52.0=40.5 Fe=116.0
CaO=3.6 = 10.0 of SiO_2 present.
S= 1.0 = 2.8
Cu= 0.7 = 2.0

Matte. Cu 43.5 per cent. 62.0
Fe= 18.5
S= 24.0

$$\frac{Cu+Fe}{S}=2.6$$

Weight of slag =4100
Weight of matte= 670

Cu in slag= 21
Cu in matte=293
314

S volatilized=74 per cent.
Matte fall=13.4 per cent.

Cu in matte....................293
Fe in matte....................125

Cu and Fe in matte....................418 lb.
S in matte (by calculation)...........161 lb.

The ore furnace puts through 400 tons daily (often as much as 480 to 500 tons) with a coke consumption of 2.5 per cent. of the ore smelted; the concentration furnace, of the same size, smelts 280 to 300 tons of matte, and with a coke consumption of 3.5 per cent. Each furnace receives 2000 cu.ft. of air at a pressure of 40 oz. per sq. in. A new furnace of this company has been constructed, 270x56 in. at the tuyeres, blown at the rate of 30,000 cu.ft. of air per min. at 40 oz. pressure. The temperature of the escaping jacket-water is 55 to 61 deg. C.

In regard to the treatment of converter slag the practice has been modified by letting it stand in pots to settle after pouring from the converter, and, pouring off the still liquid portion, while the shell, containing the drops of matte, is returned to the concentrating furnace.

Reverberatory Furnace Smelting.

Reverberatory Furnaces of the Washoe Plant.—There are seven of these reverberatory copper-matting furnaces, varying in length of hearth from 102.5 to 115.8 ft. and 19 ft. wide. For example, No. 1 furnace (see Fig. 8), 115.8x19 ft. hearth dimensions, has a fire-box 16x7 ft. and an area at the neck or outlet flue of 60x38 in. or 16 sq. ft. The waste gases pass in series through two 300 h.p. Sterling water-tube boilers to the main flue. When the boilers are to be repaired, the damper at the neck is closed and the other outlet flue opened which leads direct to an underground flue, connecting to the main flue. This is shown in both plan and elevation in Fig. 8.

The front or skimming door, 12x15 in. wide, sets 4 in. above the hearth, which slopes 8 in. toward the tap-holes at the fourth door from the fire-bridge. The rise of the roof is 19 in. To allow for the longitudinal expansion of the roof, there are 10 transverse expansion joints, 3 in. open when the furnace is built. For the same purpose three similar joints are left across the fire-bridge, which close up when the furnace is fully heated. The bridge, 4 ft. broad, is 27 in. above the hearth and 24 in. above the grate. Near it the hight of the furnace from sole to crown of the arch is 6.7 ft. There are 20 checker-holes across the furnace above the bridge for the admission of air. This is generally a sufficient air supply. Just above the level of the coal in the wall of the fire-box are set two 12-in. stoke-holes with wheel doors, and along its middle line are the four charge-

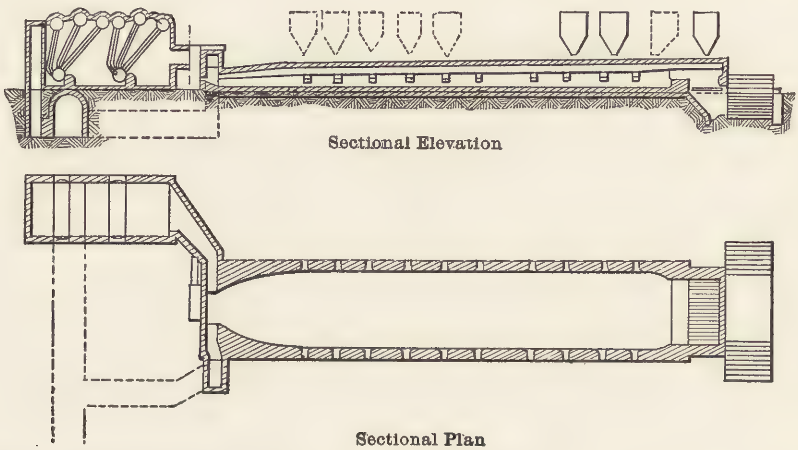

FIG. 8.—REVERBERATORY FURNACE NO. 1, HAVING 115.8 FT. LENGTH OF HEARTH, WITH WASTE HEAT BOILER SETTING.

openings, through which the coal is charged from coal hoppers, one being indicated in the elevation of Fig. 8.

There are ten 8x15 in. doors on either side with sills 18 in. above the skim-plate at the front. All the charge is dropped into the furnace by two charge-hoppers, shown in solid lines in the elevation.

In starting a furnace which has been shut down for repairs a fire of wood is placed in the fire-box, followed by an increasing fire of coal until, in 48 hours, the first charge may be dropped. Charges are added as fast as they melt until the furnace is full. The furnace takes 10 to 15 days to arrive at its full heat. Charges are put in on an average every 80 minutes, 10 tons being dropped through two charge-openings from the hopper near the fire-box and five in the other (90 ft. from the front of the furnace) so that the dust has a good opportunity to settle. The ore drops upon a bath

or layer of at least 12 in. of matte and slag, and hence does not stick to the hearth. The hot calcines spread out as they pour into the furnace, moving slowly toward the front in two streams or ridges, and gradually melting down as they take up heat from the flame and from the molten bath upon which they float. The interior of the furnace is filled with a clear, bright flame; and a steady temperature is kept up at the intensities given in Fig. 9. Skimming (or more correctly tapping) is performed every four hours, taking care, however, to do it not less than 45 min. after a charge has been dropped. It takes about 15 min., the amount run out being 45 to 50 tons, a rabble meanwhile being used to hold back and regulate the flow. To save any possible escaping matte, a settling trough, 24x18 in. cross-section and 7 ft. ft. long, is interposed in the flow, the slag being then granulated by water, and swept away by launder to the dump. It is aimed to keep 150 to 200 tons of matte constantly in the furnace; and this is tapped off from two tap-holes on the same side of the furnace through

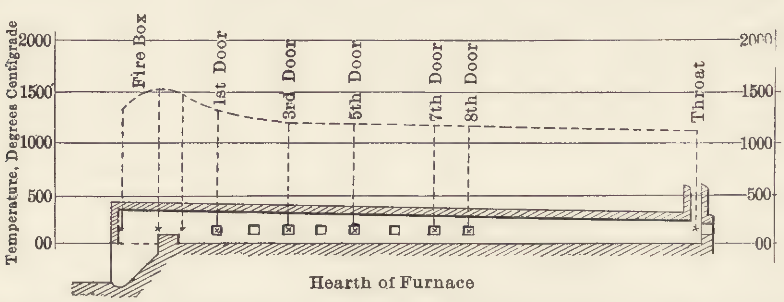

FIG. 9.—LONGITUDINAL SECTION OF REVERBERATORY, SHOWING CURVE OF TEMPERATURE THROUGHOUT THE FURNACE.

tapping-plates 12x12x2.5 in. thick. About 3000 ℔. of coal is dropped into the fire-box every 40 min. by the four openings in the roof, and the coal is carried as high as the bridge, while a clinker-bed is allowed to form on the bars. These are 2 in. square and are set 6 in. apart, permitting the dropping of a good deal of ashes, clinkers, coke, and even unburned coal; and, to add to the loss, the fire is freely grated. In this way it is possible to keep it in good condition without cooling the furnace during the grating, and to burn 44 ℔. coal per sq.ft. of grate-surface per hour. About once a month the furnace is patched or fettled near the fire-box end, where the temperature has been the highest, and the corrosion the greatest; and this work is done in the following manner: The slag is tapped and skimmed down close to the matte, and the latter is then withdrawn from the furnace, taking a train of 16 ladles. The fettling is now proceeded with, taking 20 tons of sand. The whole operation occupies eight hours. The doors are put up, the fire urged, and new charges are put in until 250 tons have been melted and the furnace has regained its normal working.

About once in eight months the furnace needs radical repairs. To effect these it is tapped dry and cooled off, and the corroded brick work renewed, mostly for 25 to 30 ft. at the fire-box end.

These reverberatories smelt 250 to 325 tons daily, and may be called 300-ton furnaces. They burn 55 to 65 tons of coal daily, or 21 per cent. of the charge, from which about 10 per cent. is recovered as coke droppings. At $4.50 per ton, the fuel cost would be $1 per ton of ore treated.

Skimming being done at the front, the furnace is not cooled, and a constant heat can be maintained. Care is taken to lute up all openings; and upon the furnace roof a layer of sand 3 in. thick is placed, to cover all cracks. A typical charge for these furnaces gives by analysis: SiO_2, 26.1; FeO, 31.3; CaO, 2.9; S, 8.1; and Cu, 9 per cent. The resultant slag contains SiO_2, 37.8; FeO, 38.6; CaO, 4.6; Al_2O_3, 6; and Cu, 0.37 per cent., while the volatilization loss of sulphur from the ore is 33 per cent.

Two 10-ton ladlefuls of matte are commonly tapped off at a time. The temperature of the escaping matte is 1035 deg. C., and when it has run by launder 80 ft. to the ladle it has cooled to 985 deg. C. The temperature of the escaping slag at the time of skimming is 1120 deg. C. The escaping gases at the neck or outlet flue have a temperature of 1100 deg. C.; just before entering the boilers 950 deg. C.; and after passing through the boilers 330 to 350 deg. C. The draft pressure at the fire-box is 0.5 in. and at the neck 1.25 in. of water.

During the first 10 minutes after the charge is dropped the escaping gases contain 2.28 per cent, of SO_2, and in the second and succeeding periods of 10 minutes the amount drops to 1 per cent. At the same time but a trace of SO_3 is to be found. We can conclude, therefore, that the reaction $FeS+3Fe_2O_3+7SO_2=7FeSO_3+SO_2$ is the principal one. Of the 8 per cent. of sulphur in the calcines $\frac{1}{3}$ or 2.66 per cent. is evolved, the remaining 5.33 per cent. going to form matte. The matte fall is 21.5 per cent. or 60 to 80 tons daily per furnace, and it contains 40 per cent. of copper.

Arizona Smelting Company.—The works of this company at Humboldt, Ariz., comprise three Edwards furnaces 100 ft. long, capable of roasting 115 tons each daily. They are oil-fired, the air supply being adjusted to give the desired degree of roast. Two oil-fired reverberatory matting furnaces, 100 ft. long, having each a maximum capacity of 350 tons daily, and yielding a matte of 40 to 45 per cent. The slag is granulated with water. Two stands of electrically operated barrel-type converters, producing a blister-copper of 95 to 97 per cent. from 100 tons of matte daily. These converters are tamped by a compressed-air power tamper handled by a jib-crane. The converter-building is commanded by a 40-ton Whiting electric traveling crane of 64 ft. span, and having a 15-ton auxiliary hoist.

At the centrally located power house there are three cross-compound Nordberg engines of 1600 h.p. aggregate, two of which are 500 h.p. each, driving two 480-volt, 60 cycle, three-phase generators which furnish the necessary current to all the motors about the works. There is also one 600-h.p. engine, direct connected to two Connersville blowers, giving the supply to the converters. Four 350-h.p. Sterling boilers are driven by the waste heat from the reverberatory furnaces.

The water supply is taken from the river by an 8-in. main suction pipe, placed horizontally in a shallow trench at the center of the river bed, with fourteen 8-inch pipe branches extending 20 ft. downward into the underlying gravel, and set 8 ft. apart on the main, so that a supply of water is obtained from the underground flow. The well above the main is 10 ft. deep with 3-ft. walls of brick laid in cement. The pump house, 15x20 ft. of steel and wood contains two 750 gal. electrically driven Swiss-Morris turbine pumps which raise the water to the 25,000-gal. tanks at the condenser house.

The consulting engineer, Cyrus Robinson, has designed and patented the following system of ore handling: Cars, provided with six-beam Howe scales (manufactured by Wonham & Magor of New York), receive and weigh the ore and flux for the charge as it is drawn from storage bins. These cars are driven by two 25-h.p. motors with an ordinary street-car controller, and are operated by two men: one as motorman, and another who makes up and weighs the charges. The ore, as received from the cars, is dropped into 10 hopper-bottomed receiving bins of 30 tons capacity each. These bins are discharged as desired by lever-operated gates upon a 12-in. belt conveyor 325 ft. long, by which delivery is made to hopper-bins set above the crusher in the sampling mill. Ten cars may thus be unloaded at once, or a single bin may be discharged by itself. At the sampling works the ore is crushed to nut size and elevated by a 10-in. Gates pocket elevator 52 ft. high to the coarse rolls which crush it to 0.25 in. A 1/10 sample is then taken out by a specially designed Vezin sampler, and the rejected portion sent to the storage bins by a belt conveyor. The sample passes to the fine rolls, set to crush to 20-mesh, when a cut of 1/10 is again taken by another Vezin sampler, the rejected portion being elevated by the belt conveyor just mentioned. The sample goes to sealed three-ton sample cars of three compartments. Deposited upon the floor of the finishing room, it is coned and quartered to a small bulk, and finally ground down on a sample-grinder to from 60- to 120-mesh. Duplicate samples, contained in the sealed cars, are preserved for check samples if necessary; and the machinery is so arranged that it is open to the inspection of the seller or his agent. There are 36 storage bins, each of 225 tons capacity, or 8000 tons in all. Delivery of ore from these bins is made either to the roasters or to the matting furnaces. It is estimated

that 50 per cent. of the ores must be roasted before being smelted at the reverberatory furnace.

The 10-ton cars, which receive the calcined ore from the Edwaıds roasters, are taken to the reverberatories by the motor-driven scale-cars. The matte from them is received in ladles handled by the 40-ton traveling crane.

Reverberatory Furnaces at Great Falls, Mont.[1]—These furnaces, 42.5x15.75 ft. hearth dimensions, are continuously charged with hot calcines, and discharge the slag and matte as fast as melted to settlers 16 ft. outside diameter by 5 ft. deep. In this way the copper loss in the slag is diminished and no time is lost in either charging or skimming, while the capacity of the furnaces is increased. These furnaces, on account of the nature of the coal used, are gas-fired.

The Garfield Works.[2]—This new plant of the American Smelters Securities Company, which is ultimately to be expanded to 5000 tons capacity per day, is situated on the shores of Salt Lake, far from the farming district of Salt Lake valley. The plant covers 30 acres, and is reached by three lines of railroad. The power-house, 130x256 ft., contains the steam and electrical power equipment needed for the operation of the plant. It is equipped with an electric-power traveling-crane of 80 ft. span for handling the heavy machine parts. The boilers are supplied by seven overhead coal-bins of 70 tons capacity each. The roaster building, 60 by 222 ft., contains sixteen 18-ft. McDougall roasters.

There are two sampling-mills, each 72x82 ft. in size. These are furnished with automatic sampling machines, and with troughed conveying belts, which carry the ore to storage bins for use at all the furnaces. The main building, within an area of 360x305 ft., contains three reverberatory furnaces, of 350 tons capacity each, four blast-furnaces of 500 tons each, four converter stands and a Huntington-Heberlein plant of 20 pots.

The reverberatory furnaces are 110x19.5 ft. hearth dimensions, with a fire-box of 8x14 ft. (112 sq.ft. grate area). The blast-furnaces are 42x240 in.; they have mechanical feed, and need 14 ft. fore-hearths. The slag is taken away by 110-ton slag-pots drawn by electric locomotives. The four converters, 96x138 in., are served by two 60-ton Shaw electric traveling-cranes, having 10-ton auxiliary hoists, and operate to the full length of the building.

The concrete main stack, which serves the works, is 300 ft. high, 30 ft. diameter, and has its base 200 ft. above the converter-floor, making a total draft hight of 500 ft. There are three flues, each 2300 ft. long, from the reverberatory furnaces to the stack which are 24 ft. high, with walls 13 in. thick, have hopper-bottoms, and needed two million brick to construct them.

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[1] *Eng. and Min. Journ.*, LXXXI, 92.
[2] *Eng. and Min. Journ.* LXXXI 509.

Upon the 13 miles of trackage needed including side-tracks, switches and spurs, there are 10,900 ft. of wooden trestles, and 1100 ft. of steel trestles. Besides this there are 3.5 miles of 3-ft. guage track for the electric locomotives, using 60-lb. rails, and having on it some 1600 ft. of trestles.

Detroit Copper Company.[1]—Besides two smaller furnaces, each 42x180 in., the plant has a large water-jacketed blast-furnace 48x264 in. hearth-dimensions, 14 ft. 10 in. from tuyeres to charging-floor, but carrying an ore-column of 7 ft., and having a capacity of 400 tons daily of a charge consisting of sulphide-concentrates. Matte of 60 per cent. copper is made.

A reverberatory furnace, 40 ft. long by 9 ft. wide, has been installed for the purpose of settling the slag and treating flue-dust. Slag flows into this furnace from the fore-hearth, and is tapped at the front into cars, clean enough to go to the dump, since it contains but 0.32 per cent. Cu. At the fire-box end 15 per cent. of flue-dust is fed in from hopper-bottom bins, and sprayed over the molten slag at that end of the furnace by means of an oil-burner blast. The dust, thus scattered over the slag, melts and disappears into the slag by the time it reaches the middle of the furnace. It costs $80 per day to run the furnace, including oil, blast, and labor, with 350 tons of slag treated daily.

Matte is tapped from the reverberatory furnace, and from the blast-furnaces, by launders to the converter set at a lower level. The converter is handled by means of hydraulic transfer-cars. Converter slag is poured into slag carts, taken away, cooled, broken up, and sent back to the blast-furnaces. Material for the converter is made of screened silicious ore and concentrates, the mixture averaging 4.4 per cent. Cu, which, without further grinding, is moistened and used for lining. A large slag dump of 400,000 tons, containing 4 to 6 per cent. Cu, is being resmelted.

Limestone and ores are stored in cylindrical steel bins, on top of which run the railroad tracks. These bins have sloping bottoms and draw-off gates delivering to two-wheeled buggies.

The Cieneguita Copper Company, Chipiona, Mexico.[2]—The plant of this company includes four wood-burning reverberatory smelting furnaces, two reverberatory roasters, and 50 roasting-stalls, together with a steam plant, gravity-incline, and several work-shops. Cord wood costs $3.50 per cord of 125 cu.ft. delivered.

Reverberatory Matte Smelting at Nijni-Tagilsk, Ural.[3]—Treating the self-fluxing copper ore in a 50-ton regenerative furnace, a matte of 70 to 75 per cent. copper was produced, the slag carrying 0.7 to 0.8 per cent. copper. After the application of a process for the extraction of copper from the silicate, the operation lasting three hours, the slag retained but 0.25 per cent. of copper. The fuel varied from 15.6 to 29.0 per cent. of the charge.

[1] *Eng. and Min. Journ.*, LXXXII, 105.
[2] *Eng. and Min. Journ.*, LXXXII, 630.
[3] *Eng. and Min. Journ.*, LXXXI, 766.

Powdered-Coal Firing.[1]—At the works of the Utah Consolidated Mining Company, Murray, Utah, the performance of the reverberatory furnace with ordinary coal firing was as follows: The furnace was 38x15 ft. hearth dimensions, and had a fire-box 5x8 ft. in size, and a stack 80 ft. high. The coal was carried to the depth of 3 ft. upon the grates, and the firing was done with a closed ash-pit, and with under-grate blast produced by an independent and electrically-driven fan. This arrangement necessitated intermittent grating, which was done just at skimming and charging. Taking a specific case, when the time of smelting was 4 hr. 27 min., the diagram below indicates the variation of temperature and undergrate pressure. Clinkers would form, both close to the grate bars, and then upon the side-walls of the fire-box, and, at the time of grating, needed heavy and arduous work to break them up, get them out, and put the fire in proper condition for the next melt. This operation would take from one-half to two hours, during which time the furnace would be practically idle, since the natural draft of the stack (without the assistance of the undergrate blast) was insufficient to maintain the heat. Besides this, the getting out of the clinkers involved the dropping and loss of 10 per cent. or so of live or even scarcely ignited coal. Moreover, at grating the firing was neglected, resulting in smoky and incompletely burned gases, and the wide variations of temperature were detrimental to the permanence or durability of the roof.

To insure a steady heat, complete combustion, and high temperature, it was proposed to use powdered coal, the preparation and firing of which was carried out in the following manner: Near the fire-box was arranged a centrifugal impact pulveriser of the Aero Pulveriser Company, New York, and on the same shaft a suction-fan running at 1400 r.p.m. The coal was fed into the feed-hopper, was drawn in finely powdered condition through the machine by the fan, and was delivered by pipes to the furnace through a flattened nozzle, flaring sideways, and set above the fire-bridge and 30 in. back of it. On the grate was carried a fire of coals to start the burning of the powdered coal, after which the incandescent brick-work was sufficient to re-start the burning, when, for any cause, it was interrupted. The quantity of coal thus burned was from 20 to 30 tons daily or equal to what would have been 50 lb. per sq. ft. of grate surface per hour. Additional air was also needed, and this was supplied by the forced draft of the ash-pit already alluded to.

Notwithstanding difficulties, due to the experimental nature of the run, the result of two months trial showed that the furnace smelted 30 per cent. more charge, and used 24 per cent. less coal per ton of ore melted. On the other hand, it took 30 to 60 h.p. to operate the machinery, but even when adding in the coal thus needed, the saving still showed a gain of 15 to 20 per cent. over the ordinary firing. Where the waste-heat is utilized

[1]*Eng. and Min. Journ.*, LXXXI, 274.

for steam-heating, even this deduction would not have to be made. At this time improvements were introduced in the construction and working of reverberatory furnaces, so that this system was no longer needed.

Converting.

Converters of the Washoe Plant.—The main floor of the converter house is commanded by two 65-ton traveling cranes, which handle the shells for ten stands of converters, the slag and metal ladles. The converter shells, newly lined, weigh 42 tons. On one side of the main floor is a bay where the matte ladles enter in order to pour their contents by launder into the converters. Below this floor are the wet pans which prepare the lining material for the converters. In the other bay are two copper-refining furnaces, one of 14 ft.x22 ft. 8 in., and one of 14x28 ft. hearth dimensions, each having a fire-box 5.5x7 ft. These furnaces are charged with molten copper from the converters poured in from 5-ton metal ladles, and the copper is cast in endless-chain anode casting machines. The converter slag as it is formed is poured by a 5-ton ladle into an endless-chain Hayden casting machine, which delivers it, suitably cooled by water sprays, to hopper-bottomed railroad cars.

The converters are of the barrel type, 8 ft. diameter by 12 ft. 6 in. long, each of nine tons average capacity. They are operated by hydraulic cylinders under a pressure of 400 ℔. per sq. in. When newly lined, the converter has a capacity of 80 cu. ft. The ganister consists of silicious ore of 70 to 85 per cent. silica, containing some gold, silver and copper, pugged with pond-slimes having some 60 per cent. of SiO_2 and 2.6 per cent. of copper, all slightly moistened with water. A silicious ore of 70 per cent. SiO_2, but containing 8 per cent. of Fe, is used—a much less efficient lining material.

For relining, the converter is opened, the crusts, etc. trimmed off, and the body relined with the ganister dumped into it and compacted with a compressed-air rammer of 32-in. stroke, weighing two tons, and suspended from a swinging jib-crane. The bottom is thus brought to within 6 in. of the tuyere level. The form or mold for the cavity is then set in place, filled round with ganister, and solidly rammed as before. The mold is now withdrawn and the tops, after being lined in the same way, set on, the joint being made with a softer mud. The weight of this lining is 16 tons per shell. The shells are dried out with a wood fire, followed by one of coal, the fire being urged by a blast entering through the tuyeres from a small Root blower.

When dry, the shell is set in its stand and the air turned on for a few minutes to burn up the remainder of the fuel. The ashes are then dumped out, and the converter set to charging position, receiving a first charge of 7.5 to 9 tons of matte, this taking 5 to 10 min. The times for working

through a straight single charge of reverberatory matte under ideal conditions may be given as follows: To the first skimming for slag 49 min., to the second skimming at the stage of white metal 63 min.; when blown to blister, 95 min.

The first charge having been worked through, 1000 ℔. of silicious ore is added to the converter, sticking more or less to the inside surface, and on this is poured the next charge. Blowing at once proceeds, the silicious ore saving the lining. As the charge becomes hotter, sweepings of slag, matte, scrap copper, etc., are put in. This controls the temperature, works off good material, and lengthens the blow on an average to 135 min. With 10 stalls in operation, 65 to 90 charges are put through in 24 hours. A converter will sometimes sustain as many as 15 pourings.

Allowing 75 charges of nine tons each daily, we have from the ten stands 675 tons of 40 per cent. matte, equal to 250 tons or 500,000 ℔. of blister-copper. When a converter is nearly exhausted, it will hold as much as 12 tons; and if it appears likely to break out, the charge may be poured into ladles, and transferred to other converters.

Regular charges are also treated in this way, blowing the matte to the first or second skimming, transferring to other converters, and charging again with 40 per cent. matte, so that it does not finish its own charge. The flue-dust made in blowing amounts to 25 tons daily, or to 5.5 per cent. of the matte treated. It is removed from the converter hoods, and later returned as sweepings to the converters. The copper ladles are lined 4 in. thick with clayey ore. Charges are seldom doubled.

Converters at Great Falls, Mont[1].—These are of the upright type, 7 ft. in diameter and will give an output of 15 tons of copper on one lining. A converter of elliptical cross-section, 10x9 ft. in the two dimensions, has lately been built which, filled to 16 in. above the tuyeres, possesses twice the usual capacity, and has a life of 50 tons per lining, while the time taken is no more than that needed for converting 15 tons in the smaller converter.

Copper Converter Melting Its Own Matte.[2]—At the works of the British Columbia Smelting Company, Greenwood, Boundary district, B. C., where they had no blast-furnace available for remelting 54-per cent. copper matte, the following plan was made use of. In an 84x126 in. converter a fire of wood was started, to which was then added 1500 ℔. of coke, and a light blast was admitted. When the fuel had become red-hot some three tons of matte were added, and the full blast turned on. The matte quickly melted, more was added, and the completely molten mass was skimmed. Matte was again added, and, this upon melting, was also skimmed, and so on until the converter was filled. The charge, consisting of

[1]*Eng. and Min. Journ.*, LXXXI, 92.
[2]*Eng. and Min. Journ.* LXXXII, 440.

some 15 tons, was then finished to blister-copper, the whole operation taking 3½ hours. It is to be understood that the method is not economical, and is a temporary expedient for treating a certain supply of matte without the necessity of first remelting it.

The Baggaley Process[1].—This consists essentially in the conversion of low-grade matte into white metal in a basic-lined converter, introducing at the same time into the converter silicious ore to satisfy the forming iron oxide from the matte. The work was conducted by Wm. A. Heywood at the Pittsmont Smelter, Butte, Montana, from Aug. 18, 1905, to Feb. 1, 1906.

The matte was smelted from a mixture of two-thirds of silicious ore, one-third pyrrhotite with converter slag and limestone, using about 9 per cent. of fuel. The ore and products were as follows:

	SiO_2	Fe	S	Al_2O_3	CaO	MgO	Zn	Cu
Silicious ore.........	52	14	17	7		1	2	4
Pyrrhotite 1st class ..	9	48	32	....	5	....	....	none
Pyrrhotite 2nd class .	30	31	13	....	23	....	....	none
Matte, In October...	......	......	....	....	......	....	....	20.9
Slag, In October.....	44	22.5	0.5	7.5	16.6	2.5	2	0.24

The matte-fall per ton of ore was 320 ℔., which gave 300 ℔. of converter slag with a total smelting and converting loss of 7 ℔. per ton of ore treated. The 8x13 ft. converter employed was composed of steel rings 11 in. thick and 24 in. wide, lined with 9 in. of magnesite brick, the interior dimensions being 4 ft. 9 in.x9 ft. 6 in. long. In operation, 1000 ℔. of silicious ore was placed in the converter and 4 to 5 tons of matte poured in upon it. The charge was then blown with additions of silicious ore till the matte was high. The slag was poured out and fresh matte added with more ore, and again blown. When it was desired to finish the white metal it was blown to blister-copper without the addition of more ore. With matte of 10 per cent. Cu 60 tons have been added during a period of 48 hours before finishing. With matte of 30 per cent. a charge has been finished in 3 hours. The same lining has been used during the whole run (5½ months) with but little repair. It is estimated that with the ordinary lining and for the 480 tons actually produced 96 linings at $10 each would have been necessary. The labor needed on the converter per shift was one skimmer, one puncher, and one helper. The amount of Butte silicious ore needed to form a converter slag of 30 per cent. silica and 60 per cent. FeO would vary from 9350 ℔. for a 5 per cent. matte to 6200 ℔. for a 30 per cent. matte.

The writer advocates the smelting of the 10,000 tons of the 3 to 4 per

[1]*Eng. and Min. Journ.*, LXXXI, 574.

cent. Butte ores there produced daily rather than to smelt 2500 tons of it as first-class ore and to concentrate 7500 tons by means of water at a loss of 20 per cent. and with the production of 3000 tons of ore which must be roasted and smelted in reverberatory furnaces because of its pulverulent condition. Indeed he claims this has been already done at the Washoe plant with the production of a 50 per cent. matte.

Electrothermic Smelting.

The ore is smelted in one furnace and then brought to homogeneous fusion in a second, thus obtaining a copper matte which flows out by a lower tap-hole while the slag comes away by the upper one.[1] In 1903, working upon 6 to 7 per cent. Chilian ore, and with a furnace of about 1000 h.p., 27.5 tons were smelted in 24 hours. There was used a current of 4750 amperes at 119 volts giving upward of 500 kw. The products were as follows:

Matte. Per Cent.				Slag. Per Cent.			
SiO_2,	0.8	S,	23.0	SiO_2,	27.2	MnO,	10.6
Al_2O_3,	0.5	P,	0.005	Al_2O_3,	5.2	S,	0.6
Fe,	24.3	Cu,	47.9	CaO,	9.9	P,	0.06
Mn,	1.4	......	....	MgO,	0.4	Cu,	0.1
....	....	......	....	FeO,	41.8	......	....

Treatment of Arsenical Products.

Treatment of Speissy Black Copper.[2]—At the Bruxlegg works in Austria anode slimes are treated as follows: They are mixed with saw-dust until of the consistence of honey, and put into a reverberatory furnace, at first at a low temperature. The heat is increased gradually to a just-visible red, carbonizing the woody contained matter. The slime is then treated in a cupelling furnace on a lead bath, the contained carbon reducing arsenic and antimony compounds. There is obtained a slag and a speissy copper-lead alloy, which may be termed "black copper," having the composition, Cu 45.6 per cent.; Pb, 28.8 per cent.; and carrying 46 oz. Ag and 0.64 oz. Au per ton.

This is charged into a small reverberatory furnace where the lead is sweated out at a softening temperature, and the dry residues (now freed from much of the lead) are sent to the refining furnace. This furnace has an oval hearth 10 ft. 8 in.x8 ft. 4 in. and a concave hearth lined with magnesite brick 9 in. thick, and is gas-fired. It is furnished with two tuyeres, one on each side of the bridge of the furnace, in order to heat it. A charge of 6.6 tons is put in the furnace through the side doors which are then luted up, this operation taking two hours. It is melted down

[1] *L'Electricien*, 1906 (2), XXXI, 276.
[2] Gustav Kroupa, in *Oest. Zeit. f. B. u. H.*, Feb. 10, 17, 1906.

gradually in ten hours, "roasting" with an oxidizing flame. Skimming follows, after which hot air is blown in through the tuyeres. Oxidation and slagging-off succeeds, while, to flux the oxides as they form, some powdered quartz ore is shoveled in. As the surface of the melted charge is lowered by skimming the slag, a fresh portion of 2.75 tons is added and treated like the first. Finally a third portion of one ton, either of a purer black copper or of anode scrap, is charged, it taking three hours to melt down. The copper is finally brought to pitch, poled, and cast into anodes. The total time of putting through a charge is 49 hours divided as follows: charging, two hours; melting the three portions, 18 hours; oxidation and skimming, 25 hours; casting, three hours. A total charge of 10.5 tons yields 5.8 tons of 22 per cent. copper slag; 0.8 ton of a final slag of 60 per cent. Cu, and 26 anodes containing Cu 95.4; Pb 0.2; Sb 2.2; As 0.05; Fe 0.4; Ni and Co, 0.16; and S 0.2 per cent.; also 158 oz. Ag per ton.

Treatment of a Speissy Copper-Lead Alloy.[1]—Copper-bearing ores containing lead and arsenic of the Rammelsburg district in the Harz, Germany, are smelted at Okerhütte. There accumulates in the blast furnace a crust of a speissy copper-lead alloy of the following composition: Cu, 25.9; Pb, 16.7; As, 11; Sb, 13.5; Fe, 22.2; Co, 1.1; Ni, 1.6; Zn, 3.3; S, 4.1 per cent., together with Au and Ag, 25 oz. per ton. This is treated in a reverberatory furnace with water-cooled walls, and having a hearth of three layers, the lower of marl, the second of fire-brick, and the upper of a finely ground mixture of 80 per cent. marl and 20 per cent. of a clayey-schist. The furnace is charged with five tons of the above-named alloy, all openings are closed, and the furnace brought to a red-heat, so that the charge is in a semi-plastic condition capable of being well oxidized. The temperature is then raised to the melting of the charge, with the formation of a spongy-iron scum. Blast, at 1.5 in. of mercury, is now turned on by means of two tuyeres set one on each side of the fire-bridge and sending converging jets to the center of the furnace. The spongy scum gradually melts and is skimmed off. As oxidation proceeds the iron is followed by the lead, antimony, and zinc, these bases forming silicates with the silica from the hearth. To remove the arsenic the furnace is cooled down until its contents assume a pasty condition at a clear red heat, dense fumes of arsenic acid coming off. As the fuming decreases the heat is raised with the evolution of more arsenic acid until action ceases. At this stage the metallic bath contains 75 per cent. of copper.

The furnace is fettled and the second charge, consisting of three tons of the speissy-alloy, is put in. The furnace is brought to a red heat, the charge roasted, and finally melted rapidly. After skimming, the blast is again turned on and oxidation and skimming alternate until the metal is refined. To note the progress of the refining a dip sample is taken,

[1]*Glückauf*, Sept. 26, 1905.

which is granulated by pouring into water. The granules are at first black, but as the refining proceeds the color changes to gray, and finally to a silver-white. As soon as the granules show the bright red surface of fine copper, refining is regarded as complete. The copper is now tapped off and granulated in water. These granulations are subsequently dissolved in sulphuric acid and used for making bluestone. The residual slime is separated by filter-pressing and worked on a cupelling-hearth to obtain the gold and silver.

Hydrometallurgy.

Extraction of Copper from Atacamite[1].—This mineral, consisting of various oxychlorides of copper, is found in great abundance in the Province of Atacama, Chile, hence its name. Coarsely broken, it is charged into large cast-iron retorts, and then being heated to 230 deg. C., is treated by steam under two pounds pressure. This transforms the copper compound into copper oxide and hydrochloric acid. In this form it can be smelted without fear of its loss, while, in the form of chloride, at least one-half would be volatilized in the furnace.

The hydrochloric acid produced is further used for the leaching of the low-grade ore in asphalt-coated tanks provided with stirring arms for agitation. This ore is easily treated by the acid which should be in quantity sufficient to dissolve any existing copper oxide. It is to be understood that there must be no calcum or magnesium carbonate present.

Filtration can often be conducted in the ordinary filter-tank, since the acid treatment does not produce the slimes so prejudicial to leaching. It is proposed to use the Elmore centrifugal filter, since it is claimed that, with a capacity of 100 tons in 24 hours, there will be needed but 8 h.p. The filtrate is returned to pass through the next charge in order to concentrate it, since copper chloride can be concentrated up to 60 per cent. of the salt. Because of the cost of making cement copper the author prefers to evaporate the solution somewhat, and to treat it with petroleum or other hydrocarbon, the copper being reduced to metallic copper by heating in a retort. The copper thus reduced is quite pure.

Wet Methods for the Recovery of Copper at Rio Tinto[2].—Details of the methods there practiced are given in THE MINERAL INDUSTRY, Vol. XII, p. 112, to which we add particulars as follows:

The mineral contains 2.5 per cent. copper on an average, varying from 1 to 3 per cent. and containing 45 to 48 per cent. of sulphur.

To extract the copper by natural weathering twenty years would be necessary[3]; roasting destroys the vegetation of the country; there, there-

[1]*Boletin* de la Sociedad Nacional de Mineria, Feb. 28, 1906.
[2]By M. P. Truchot, chief chemist, Huelva, *L'Echo des Mines et de la Metallurgie*, 1906, p. 482.
[3]The copper lost as sulphate from the Rio Tinto lode alone from the fall of the Roman Empire to the present day would amount to 80,000 tons.

fore, remain the wet methods where the reactions may be hastened by the use of a regulated supply of air and water.

In preparing the heaps for the oxidizing leaching of the copper, the fines amount to 80 per cent. the lump ore to 20 per cent. The two grades are placed in alternate layers and the top of the heap finished with fines to prevent too rapid filtration. Practice varies as to the temperature of the heap, which may be as high as 82 to 90 deg. C. where it is sought to promote oxidation and to increase the rate of leaching, while, in other cases, the temperature is not allowed to exceed 30 to 32 deg. C. Since the higher temperature is dangerous, one of no more than 45 to 60 deg. C. is recommended.

It takes six to seven years to exhaust one of these large heaps of 100,000 tons; the exhausted ore then containing 0.25 to 0.30 per cent. Cu only. The more permeable copper schists leach more rapidly, however, taking but three to four years. While the oxidation of chalcopyrite proceeds but slowly, the rate can be increased by finely pulverizing it and distributing it throughout the pile. It is surprising, when one examines a piece of massive sulphide, that it is so easily leached when thus in compact form. Yet there are in it imperceptible fissures into which the ferric sulphate solution penetrates, dissolving out the contained suphates of copper.

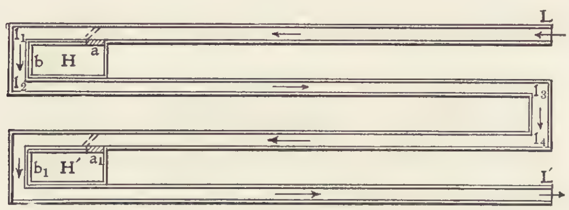
The copper solution from the heaps varies from 0.015 to 0.5 or even 0.6 per cent. It is of reddish-green color, containing ferric and ferrous sulphate, copper sulphate, free sulphuric acid, not to say bismuth and antimonial silver sulphates, and iron arsenate. To reduce the ferric sulphate so that it may not consume iron, the solution is slowly run over beds of newly mined pyrite which reduce it to the ferrous form. These beds are of fine ore, while the top, made up into squares of 5 ft. on the side, is finished with the finest mineral. The beds are disposed upon a sloping surface of 5 deg. toward a reservoir formed with a masonry dam, where the solution is brought together and has changed to a clear blue color.

Old-iron scrap is placed in the first deposition launders and there results a dark colored copper precipitate less pure than later on, then 250 to 300 yd. farther on it becomes brilliant and adherent to the pig iron. As one leaves this part of the launders the copper becomes granular, though always of a clear red color. Finally, when the solution is nearly exhausted, more and more arsenic, antimony and bismuth are deposited, giving products more or less black. The precipitate always contains graphite from the pig-iron.

When the water supply is abundant, and the free acid is consequently low, the consumption of pig-iron is about 1.3 to 1.5 tons per ton of copper precipitated. This seldom occurs, however, and generally we find that there is needed 1.75 to 2 tons per ton of copper extracted; that is, more than double the theoretical quantity, which is but 0.86 ton.

The highest-grade precipitate (cascara) is obtained from the first of the pig-iron, and contains 93 to 94 per cent. copper and a little arsenic, the granular copper contains 75 to 90 per cent. copper and about 0.5 per cent. of arsenic, the final, low-grade product (papucha) contains but 50 per cent. copper, large quantities of arsenic, antimony, bismuth and almost all the graphite.

Recovery of Copper from Mine Drainage[1].—Philip Argall gives the practice at the Tigraney mine, County Wicklow, Ireland, as follows. Fig. 10 and 11 are respectively the elevation and plan of the launders through which flow the copper-bearing mine waters, while at H are the sumps or hutches where the precipitate is collected. The launders, 15 in. wide by 12 in. deep, are set at a 10 per cent. grade. The launders were covered to a depth of 4 or 5 in. with iron scrap. At distances of 500 ft. along the course of the launders were set slightly lower than the launders the hutches H,



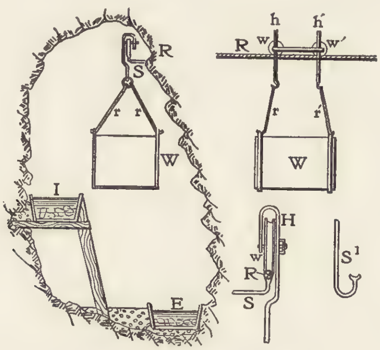
FIGS. 10 AND 11.—DIAGRAMS REPRESENTING THE DISPOSITION OF THE LAUNDERS.

5 ft. wide, 10 ft. long and 6 ft. deep. Twice daily the copper precipitate was removed. The water was turned off the 500 ft. section, and a man at once brushed the scrap iron with a stiff birch broom to dislodge the accumulated copper and, in part, the insoluble graphite. The water was then turned on with full force, washing the copper precipitate before it, the man managing to keep ahead of the onflowing water, and, upon reaching the first hutch, turned the stream into it. A second man turned the scrap over with a two-pronged fork, exposing other surfaces to a third man, who, following him, again brushed and dislodged more precipitate from the iron. This work about filled the hutch with water, when the stream was changed again to its course along the main launder. The other sections were treated in the same manner successively. The contents of the hutch were allowed to settle for 12 hours, and the supernatant water drawn off from above the copper which had settled to the

[1]*Min. and Sci. Press*, July 28, 1906.

bottom, leaving the hutch ready for the next sweeping. The hutches were cleaned out quarterly, and the precipitate put through a 5-mesh screen, the oversize being returned to the launders. The launders were also cleaned quarterly, each piece of iron being brushed to free it from the copper. The finer material was shovelled out and hand-picked, coarse copper going to one place, the clean iron returned to the launder, and the rust and scale rejected.

As the solution became weaker in copper there was a tendency to deposit ocher upon the iron. This could be measurably avoided by increasing the grade of the launders, or the velocity of flow. Water falling some distance on iron gives a coating of metallic copper, while mine-drainage water, flowing slowly, deposits its copper in granular and easily detached form. In places where plenty of fall was available the nearly spent solutions were turned down stairs, the steps of which were covered with heavy pieces of iron scrap, which in time became covered with co-



FIGS. 12, 13, 14 AND 15.

herent malleable copper, bringing the solution under the oxidising influence of the air.

Following the stairs were ponds, four in number, each 50 by 400 ft., in which reactions were completed and the ocher settled out. The solution then went through another series of precipitation launders, and so through cycles consisting of (*a*) precipitation of the copper, (*b*) oxidation of soluble salts and (*c*) precipitation or settling out of the ocher, and beginning the cycle over again with the solution considerably poorer in copper.

The precipitate was usually carefully dried on cast-iron plates heated by fire below so as not to burn the copper. The dry precipitate was barreled and shipped. Where it has been possible to use an abandoned drift in order to conduct precipitation in the mine because of the advantages of shelter and uniform temperature, this has been done as indicated in the illustrations Figs. 12–15. The mine waters are carried by the

launder *I* which starts near the top of the drift and descends on a regular grade until it reaches the floor, where it returns on the same grade to the entrance of the drift. The scrap-iron which is used for precipitating is brought in on a car suspended on a cable which runs along the top of the drift, this cable being supported by irons, *S*, driven into the wall, or by irons, S^1, driven into the roof of the drift. This "monorail" is well suited to the sinuous courses of the level.

Manufacture of Cupric Chloride.[1]—If, into a closed vessel containing copper-shavings and water, we introduce chlorine gas, the gas is slowly absorbed, at first transforming the copper to cupric chloride according to the equation $Cu + Cl_2\ Aq = Cu\ Cl_2\ Aq$. As the cupric chloride increases in quantity the reaction enters a new phase. The copper is attacked both by the chlorine and by the cupric chloride thus, $Cu\ Cl_2\ Aq + Cu = 2Cu\ Cl\ Aq$, the reaction becomes tumultuous, the mass rises to a temperature which may exceed 100 deg. C., and the copper becomes entirely dissolved as cupric chloride. The concentrated solution is now drawn off, crystallized and dried. Or again, it may be treated by sulphuric acid at 100 to 120 deg. C. to transform it into bluestone, and the latter salt manufactured. The gaseous hydrochloric acid, formed according to the reaction $Cu\ Cl_2 + H_2SO_4 = CuSO_4 + 2HCl$, is collected.

The Extraction of Copper from Pyrite Cinder or from Copper Ore.[2]—Well-roasted copper pyrites or copper cinders contain copper as: (1). Copper sulphate soluble in water; (2). Cupric oxide soluble in ferric sulphate; (3). Cuprous oxide capable of being oxidized by ferric sulphate which will then dissolve it; (4). Copper sulphate oxidizable by ferric sulphate which will then dissolve it.

The method consists then in lixiviating the cinders with a solution of ferric sulphate by which the copper salts are dissolved out and pass into the filtrate. This filtrate will contain ferric sulphate, ferrous sulphate copper sulphate and the sulphates of other metals when present, such as zinc, manganese, cobalt, nickel, and alumina. Ferric sulphate is very effective in bringing the copper salts into solution so that in burnt pyrites containing 0.8 to 4 per cent. copper, there is left in the residues 0.05 to 0.2 per cent. only.

The filtrate contained in a tank is brought to the temperature of 60 deg. C. and to it is added little by little ferric sulphate solution till by testing with ferro-cyanide solution the end point is reached. This may be, for example, 0.66 per cent. of a ferric sulphate solution of 25 deg. B. This oxidizing action will not exceed two days and for some kinds of cinders but a few hours. Air is now driven into the solution in presence of an alkaline base, such as milk of lime. The ferric sulphate is precipitated

[1]*Revue des Produits Chemiques*, July 15, 1906.
[2]Carl Millberg, in *Chemiker-Zeitung*, XXX, 511.

as an insoluble basic sulphate. This is filtered or decanted off and there remains in the filtrate the sulphates of copper and of other metals.

This solution is heated to the boiling point and to it is slowly added diluted milk of lime which precipitates the copper as an insoluble basic sulphate of a light green color and leaves the other sulphates in solution. The precipitation must be watched and the treatment stopped as soon as the copper has been precipitated. Should this have been pushed too far the excess can be corrected by the addition of copper sulphate solution which will redissolve the oxyhydrates of the other metals. Upon filtering the basic copper sulphate remains upon the filter while the solution contains the sulphates of the other metals which can be later separated as desired by well-known methods. The precipitate is washed on the filter, then dissolved in dilute sulphuric acid to obtain a concentrated solution of copper sulphate from which bluestone can be crystallized out.

Electrolytic Refining.

Direct Electrolytic Production of Copper from Matte.[1]—The matte is brought up to a grade of 78 to 80 per cent. in a converter, and cast into anodes. The cathodes are of pure copper and the electrolyte is the usual acid solution of copper sulphate. A current of five amperes per sq. ft. is used with a pressure of a little less than 1 volt. The undissolved residues are treated to extract the remaining copper and the precious metals. Nickel enters the solution and can be precipitated from it.

Electrolytic Copper.[2]—Electrolytic copper is, in conductivity, superior to nearly all brands of Lake copper, but for certain manufacturing purposes, as in the making of cartridge shells, the latter claims the superiority, due to a small quantity of arsenic which it still retains. Eventually, commercial copper is likely to include two grades, high-conductivity copper, including electrolytic and picked brands of Lake, and casting-copper of less than 98 per cent. Mathesson standard when annealed.

Acidulated copper-sulphate solution, the usual electrolyte, is made with the cheapest of commercial acids, and carries enough excess of sulphuric acid to make its specific electrical resistance low. There is a slight re-solution of the cathodes. Gold is untouched, and silver practically so. A small percentage of some soluble chloride is added to the electrolyte with the object of throwing down any silver and antimony which may tend to enter the electrolyte, and to make smoother cathodes.

In American refineries high-grade anodes are everywhere used. The usual compositions are: Cu, 98 to 99.5 per cent.; Ag, 0 to 3000 oz. per ton; Au, 0 to 40 oz. per ton; As, 0 to 2 per cent.; small amounts

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[1]German Patent, No. 160,046, Oct. 5, 1904.
[2]Lawrence Addicks, in *Journ. Frank. Inst.*, CLX, 421.
The Mineral Industry, XIV, 161; *Trans.* A. I. M. E. XXXVI, 19.

of Sb, Bi, Fe, Ni, S, Se, Te, and Si impurities, which form soluble sulphates, go into the electrolyte, and gradually increase in quantity. Selenium, tellurium, gold and silver go into the slime. We therefore have a threefold separation: copper at the cathodes; silver and gold in the slimes; copper and nickel (if present) as sulphates in the electrolyte. Selenium and tellurium are easily recoverable from the slimes, but have but little market value. Arsenic, antimony and bismuth go partly into solution, partly into the slime, depending upon the form in which they exist in the anode, and upon secondary reactions taking place in the electrolytic.

Arsenic is the most difficult impurity to deal with. If present in the anode to over 1 per cent. it rapidly grows in the electrolyte and is costly to keep down. With small quantities it seems to slime more rapidly, and the electrolyte needs no purification. Under favoring conditions an electrolyte of 2 per cent. arsenic will still permit of satisfactory cathodes, though this is above the customary figure. Antimony and arsenic seldom cause trouble. Various chemical methods of purification have been tried, but thus far unsuccessfully. There are few reagents which will precipitate arsenic in acid solution, and their action is unreliable. Thus Mr. Addicks has boiled electrolytes high in arsenic with metastannic acid and obtained a remarkable cleansing action, but on other trials found hardly any precipitation.

The anode-slime is of variable composition, a typical one being Ag 40; Au 2; Cu 25; Se and Te 5 per cent.; As and Sb 10 per cent.; and Pb, SiO_2, H_2SO_4 18 per cent. The presence of so much copper is objectionable, since, in cupelling the dried slimes, rich slags would be formed. After screening out the scrap-copper the remainder is removed by leaching it out with hot concentrated sulphuric acid, taking care not to carry the action so far as to redissolve silver. Other impurities, except tellurium are eliminated, and this must be burned off by prolonged furnace-treatment.

Cathode copper is exceedingly pure, usually running 99.93 per cent. Cu. The objectionable impurities are of two classes, those which decrease conductivity and those which make the metal brittle. Arsenic and antimony are in the first class; tellurium and lead in the second. As regards conductivity 0.0013 per cent. As, or 0.0071 per cent. Sb lower conductivity one per cent., and low conductivity is quite commonly due to the presence of one or both of these metals. Impurities of the second class rarely exist, and if present have probably got in mechanically. A third class of impurity is silver and gold, and cathodes usually contain from 0.1 to 1.0 oz. silver and a trace of gold, due to mechanical fouling.

Indeed, it may be that arsenic and antimony also get into the cathodes in this way. The slime is very finely divided, and, since the electrolyte must be circulated to prevent polarization, there results a slight turbidity.

The floating particles are doubtless hydrostatically attracted to the cathode.

The theoretical amount of copper deposited per ampere-day is almost exactly an avoirdupois ounce, but the actual amount is always less than this, due to re-solution of the cathodes, groundings, and short circuits between the electrodes. The re-dissolving of cathodes amounts to 0.5 to 1.0 per cent. of the amount deposited, and takes place chiefly at the surface of the bath, while, in presence of oxygen, copper is slowly oxidized to cupric sulphate in presence of the sulphuric acid, especially at the higher temperature, and the cupric sulphate has a solvent action on metallic copper. If the tanks are well insulated, and the circulation pipes are broken by rubber connections, and by letting the out-flowing solution fall, the current lost by grounding will be no more than 1 per cent. of the total. Short circuits between electrodes, or between the electrodes by way of the tank, will aggregate 5 per cent. Thus the net current-efficiency becomes 90 to 95 per cent. of the theoretical. The voltage at the switchboard needed to force the current through the tanks consists of metallic resistances, liquid resistance, and the counter-electromotive force. The metallic resistances should be so adjusted that the cost of the power lost by the resistances equals the interest of the investment in the copper conductors.

The liquid resistances of the solution depend on its chemical composition. In a pure electrolyte, copper should approximate to 3 per cent. (12 per cent. if figured to bluestone), and the acid may be raised to 13 per cent. for best results, since, if carried higher, polarization may offset gain in conductivity. In practice impurities cause the resistances to be 10 to 15 per cent. higher. Transfer resistance is a name given to a liquid resistance probably due to minute bubbl s of gas generated close to the electrode. If we make a series of measurements of voltage-drop at different current densities, and plot them, we obtain graphically a line which cuts the ordinate of zero-current at a point indicating a measure of the counter-electromotive force present. We find however upon analysis that the specific resistance of the electrolyte is greater than it should be, and the discrepancy is the greater, the closer the electrodes are to each other. The liquid resistance, due to temperature, varies inversely with its rise, but the practical effect is somewhat complex, the transfer-resistance rapidly dec easing, so that the resultant coefficient is 0.5 per cent. per degree F.

Contact-resistances are to be found at the joints of the main bars or conductors, and at the connection between these bars and two anodes. Main bars carrying 300 to 400 amperes per square inch of section give no trouble, but the contacts between the electrodes and the main bars are very variable. A single contact will run from 0.000005 to 0.0005 ohm according to the cleanness and pressure of the engaging surfaces.

The counter-electromotive force between electrodes is due to the greater

concentration or strength of the electrolyte at the anode than at the cathode, and may be estimated at 0.02 volt. The anode-slimes produce some polarization, which, if bulky, may start intermittent gasing, making the tank "crazy."

The total tank resistance is made up of various factors, namely: Metallic resistance, 15 per cent.; electrolyte (including transfer resistance), 60 per cent.; contacts, 20 per cent.; counter-electromotive force, 5 per cent.; total, 100.

In present American practice current density runs all the way from 12 to 35 amperes per sq.ft. To prevent polarization it is necessary to maintain an electrolyte of uniform composition, and this is done by circulating it from tank to tank. Were this not done, the heavy solution from the surface of the anodes would work to the bottom and form there a horizontal layer, of higher density. Different strata would have different conductivities, and the operation of the tank would be disorganized. The higher the current-density, the greater this tendency, and hence the greater the activity of the circulation. On the other hand, excessive circulation tends to stir up the slimes, and results in an increased loss at the cathodes. Roughly speaking, we may say that the silver contents of the cathodes, at 30 amperes per sq. ft., is twice that at 15 amperes. High current-density also causes rougher deposition, so that the cathodes must be more frequently renewed. Most of the refineries in the eastern United States use a current-density of 17 to 20 amperes per sq. ft.

Low efficiency is caused rather by the condition of the deposit on the cathode or by its tendency to sprout and form trees across to its anode (due doubtless to the effect of various impurities in it) than by local mechanical troubles between single pairs of electrodes. As many as sixty electrodes in a single multiple tank give no trouble.

Various chemicals have been tried to ensure a smoother deposit, and notably ammonium sulphate, but this salt increases the resistances of the solution. Gelatin also has a similar action, but it is not easily controlled. Often the character of the deposit may be changed by simply changing the temperature of the electrolyte.

But little cathode copper is melted into brass or copper-castings, though an ideal form for them. The conductivity of cathodes is lessened by remelting into wire-bars, due to the absorption of impurities from the furnace-lining, and partly to the mechanical adhesion of impurities during the electrolytic process, which particles do not get into the sample from which the wire is drawn. Cathodes are remelted in large reverberatory furnaces, air-rabbled to 6 per cent. cuprous oxide in solution, and poled to bring the copper to pitch, when it still contains 0.6 per cent. of Cu_2O.

Comparing the multiple- with the series-system of refining, we find the power needed in the former is one-half more than in the latter. The

series-tank has relatively few contacts or conductors, and the electrodes are close together, the anodes being specially hand-cast or rolled plates, and the grade of copper must be good; also there is less interest on the metal tied up in the process, and a less expensive plant. No starting-sheets are needed, but, on the other hand, it is harder than in the multiple-system to keep up the quality of the cathodes, and lead-lined tanks cannot be used. The fact that the large refineries are run on both systems indicates that either can be satisfactorily operated, though much more anode-copper is refined by the multiple system.

The Tacoma Copper Refinery.[1]—A tilting furnace receives blister-copper from the converter, and each day the accumulated copper is cast into anodes of 125 ℔. for treatment in the electrolytic department. The anodes are set 18 in a frame for transfer to the vats. Here the cathodes, when ready, are removed from the vats by a 20-ton electric crane, loaded on cars and sent to the re-melting furnaces to be cast into ingots, cakes, and wire bars. The output is 30 to 43 tons daily. The current for refining is obtained by long-distance transmission from Electron, Wash., in the foot-hills of Mt. Rainier. The current of 40,000 volts is stepped down at the works to 100 volts.

The Nichols Copper Works.[2]—At these works, at Laurel Hill, Long Island, N. Y., ores are smelted, matte converted and anodes electrolytically refined. The ores are smelted in two water-jacketed, Herreshoff, 46x120 in. blast-furnaces with nine tuyeres on each side, and with a smelting column of 7 ft. The blast-pressure is 16 to 24 oz. per sq.in. Each furnace is run with an open-breast and continuous overflow through a large settler to the slag-pots, and has a capacity of 210 tons daily, with a consumption of 8 to 9 per cent. coke.

The matte is tapped to sand-beds, and has a composition 40 to 60 per cent. Cu; 40 to 20 per cent. Fe; and 20 per cent. S. This matte, together with purchased matte, is melted down in a cupola-furnace with 9 per cent. of coke, and is tapped off, as needed, to the converter. There are four stands of hydraulically operated converters and twelve shells. The converters are lined with ganister of 85 per cent. crushed quartz and 15 per cent. clay mixed in wet pans, the lining, 18 in. thick, being put in by hand. The charge varies from two to six tons. The converters are blown at 8 to 10 ℔. pressure per sq.in. by two blowing engines of 100 and 120 h.p. respectively, and it takes on an average two hours per blow. In 24 hours the four converters (two are in continuous operation) produce 150,000 ℔. of blister-copper of 99 to 99.25 per cent. Cu, 50 to 100 oz. Ag., and 1 to 2 oz. Au per ton. The converter sustains 10 to 16 charges before requiring relining. The furnace and converter fumes are drawn off through a main

[1]*Eng. and Min. Journ.*, LXXXII, 147.
[2]*Eng. and Min. Journ.*, LXXXI, 314.

flue 1000 ft. long to a 300-ft. stack, and from the flue is recovered 100 tons of dust per month. This is briquetted and returned to the blast-furnaces.

Nearly 90 per cent. of the material worked up is blister-copper. After sampling, this it is remelted and refined to produce smooth anodes which are rolled, since under the series-system of refining the anodes need to be particularly smooth. There are 10 remelting furnaces each of 40 tons capacity, having sand bottoms. The casting is done through a ladle to a Herreshoff casting machine fitted with nine anode moulds, the machine needing 0.25 h.p. The anodes, 59 in. long, 16 in. wide, and 0.5 in. thick, weigh 67 lb. each. For furnishing the current there are 17 generators, each of 800 amperes at 155 volts, or of 167 h.p. each. There are eight cathode remelting furnaces, each of 35 to 45 tons capacity, by which the copper is cast mostly into wire-bars, and, to prevent sticking, the molds are painted with a mixture of lamp-black and benzine. It takes five hours to pour one charge. To determine conductivity three test-bars are taken during the dipping and drawn into wire.

Copper Refining at Perth Amboy.[1]—At the works of the American Smelting and Refining Company, the blister-copper for anodes is remelted in two furnaces of 125 tons aggregate daily capacity, and molded into anodes by Walker casting machines. The anodes are transferred to the tanks, 26 together. The tank house contains two series each, of 408 tanks. The current is furnished by two generators giving 4700 amperes at 105 volts. The cathodes are run for 14 days before replacing, and the anodes are dissolved to 15 per cent. of their original weight. For melting the cathodes there are two furnaces of 75 tons capacity, which, by means of Walker casting machines, mold it into ingots and wire-bars, and with a monthly production of 3200 tons. The electrolyte is gradually withdrawn from the tanks, and treated with granulated copper in baskets to neutralize the free acid. These liquors are then sent to be manufactured into commercial bluestone. The granulated copper is prepared from blister-copper, worked up from the leady copper-matte of the blast-furnaces.

Notes on the Electrolytic Refining of Copper in the United States.[2]—A loss of current occurs in the series system from the anode-slime at the bottom of the bath being a good conductor, and accordingly serving as a conductor between the terminal electrodes at the end of the tanks. In the multiple system, the voltage being low (0.2 to 0.3 volt), the tank may be of wood lead-lined, but for series work lead cannot be used, and accordingly slate or asphalt-painted tanks are employed. However, these soon become impregnated with the electrolyte, increasing their conductivity, and thus causing some loss of current. Another feature about multiple work is that one side only of the anode acts as such,.the other side receiving the deposit and acting as a cathode; while in the

[1] *Eng. and Min. Journ.*, LXXXI, 169.
[2] *Teknisk Tidscrift*, Stockholm, 1906.

series-system, both sides of the anodes are in process of solution, and hence a little less than twice the copper as anodes is needed in series as compared with that necessary in multiple work. This means a much greater quantity of money locked up in process of treatment, several thousands of tons of copper being constantly in use for anodes in a large refinery.

Attempts have been made to overcome this disadvantage by making the anodes thinner, but, since they must be withdrawn in sheet form and remelted the percentage waste is much greater, moreover, the plates must be renewed oftener at an increased labor cost. On the other hand, the volume of electrolyte in the series need be but one-fourth of that in the multiple system, and the electrolyte is carried at the higher temperature (48 deg. C.), thus decreasing resistances, and in these two ways the size of the plant is decreased. In series work, to lessen the risk of short-circuiting special attention must be given to remelting, poling, casting, and rolling the anodes, so that they may be smooth and homogeneous; and by the use of them it has been possible to decrease the electrode distance from 2 to 2.6 in. down to as little as 0.5 to 0.8 in. When the anodes are not homogeneous, or contain many impurities, they dissolve in an irregular way, the intensity of current becomes unequal, and a stronger deposit of copper occurs at the part of the cathode which is immediately opposite a point of the anode of higher intensity, forming a "button" which may furnish a point for short-circuiting.

When it has been attempted to hammer out the anodes the results have been less satisfactory then when they have been rolled. The cost of preparation of anodes for series work over-balances the gain attained by economy of power, the lessened interest, and depreciation on capital locked up in electrolyte and plant. In series work there are no special cathodes (stripping sheets) to be prepared, but against this is the difficulty of making a good separation of the cathode from the adherent anodes.

In general, one may say that the multiple-system is more economical than the series-system, the capital invested in copper is smaller, and the upkeep and depreciation smaller. According to Barrett, the costs are \$2.28 per ton less than in series work; and, when the anodes are cast directly from the converter, this difference is greater. He gives the total cost of refining at \$6.78 per ton, which, in modern works, has been brought as low as \$4.60 to \$5.32 per ton.

In the older practice, and carrying but 2 to 4 amperes current per sq. ft., electrolytic refineries needed to carry 75 to 100 times their daily production in stock, but in late years, with 15 to 17 amperes only one-fifth of this amount has been needed. The current-density depends upon the purity of the anode copper, so that, when 8 to 10 amperes would be prudent for an impure copper, for specially fine anodes as high as 40 amperes have been employed.

Current-loss has been cut down by careful attention to the arrangement of conductors, and by the careful insulation of generations, connections, and tanks. A. L. Walker's arrangement of conductors is made thus: Two tanks are connected by a $1\frac{1}{4}$ in. triangular bar on the common long sides, which takes from the main conductor 5 per cent. of the total current of 4000 amperes flowing through the bath. The cathodes and anodes of each tank are arranged alternately, so that the cathodes of No. 1 tank are connected to the anodes of No. 2, the anodes of No. 1 being connected to the main conductor. Thus the pairs of tanks are connected in series, saving in copper conductors. This, according to Mr. Walker makes a saving of some 83 per cent. of the total copper needed for the mains of the whole plant. The electrodes rest at one side upon the triangular rods, on the other on a china insulator.

Stripping sheets for cathodes are deposited in special tanks on lead electrodes thinly coated with grease and plumbago. When they have attained the proper thickness they are quickly stripped by means of a sharp cutter-bar, trimmed, from excresences, punched for hangers, either of wire or copper strips (made from the sheets themselves), fitted with rods, hung in frames, and handled by cranes to the tanks. The cathodes are remelted into ingots, cakes, and wire-bars, and the undissolved portion (scrap) of the anodes remelted with blister or other copper into anodes.

To keep down the percentage of copper sulphate, which would tend to increase in the electrolyte, one tank in each hundred is reserved for the removal of the copper on lead electrodes. With the electrolyte thus treated, according to Haber, in 30 months the contained iron rose from 0.091 to 0.296 per cent., and the antimony from 0.13 to 0.31 per cent., while arsenic fell from 0.07 to 0.05 per cent. But little silver passes over to the cathodes in normal working (0.2 to 0.3 oz. per ton), but as the acidity of the electrolyte diminishes, the silver thus transferred increases. To guard against this loss a little hydrochloric acid (0.003 to 0.006 per cent. Cl) is added to the electrolyte, when the solution is not acid enough, depositing the silver as Ag Cl. If too much HCl is added, there is great liability to short-circuiting due to the cathode copper depositing irregularly and forming needles on the cathode. The amount of arsenic in cathodes should not be allowed to exceed 0.001 to 0.002 per cent. It occurs in the anodes partly as metallic arsenic, partly as copper arsenide, the latter going into the anode mud. Metallic arsenic dissolves as arsenic acid, and remains dissolved in the electrolyte as long as the latter is acid and not saturated. Experiments have been made to obtain arsenic-free copper by adding ammonium sulphate up to 20 per cent. to the solution. An active circulation of the electrolyte is, however, more practical.

The addition of a small amount of HCl, as above mentioned, has been of much help in attaining higher current-densities, since the antimony

remaining in solution together with the arsenic, is hindered from depositing on the cathodes as long as the electrolyte remains normal. Selenium and tellurium exert no deleterious influence (as far as known) upon cathode copper; lead goes entirely into the anode slime. Nickel remains as a sulphate in the solution. When the acidity of the electrolyte drops off, or where the current-density is too small, cathode copper may become so brittle that it may be pulverized, due to the presence of cuprous oxide.

To keep up the purity of the electrolyte, a portion of it is removed periodically from the system. Its free acid is neutralized by treating it with granulated copper placed in baskets, which are alternately raised from and lowered into the solution, into which air is also blown. The solution is then concentrated, and bluestone crystallized out.

Finally the remaining copper is separated with scrap-iron and the impure residual liquid run to waste. At Chrome, N. J., the electrolyte is not entirely neutralized with copper, and, before crystallizing, the liquor is acidified with strong acid, heated, and let stand, so as to crystallize out sulphates, not of copper only, but also of nickel and iron. Upon filtering, and cooling the filtrate, arsenic acid crystallizes out, while antimony and antimonic acid are precipitated. The residual acid is utilized for dissolving other anode mud.

The circulating electrolyte goes back to the sump and to the main supply-tank, where it is heated to 50 deg. C. before again passing through the tanks. Blowing air into the electrolyte assists in depositing impurities, and colloid solutions (glue or size) help to coagulate suspended matter at the bottom of the tank, which otherwise would become mechanically entangled upon the surface of the cathodes. The anode slime is, as a rule, treated in a phosphor-bronze centrifugal machine, whereby the copper-scrap, amounting to 10 per cent. is separated and returned for remelting. The residual slime is mixed with 95 per cent. of its weight of 66 deg. Bé sulphuric acid, and boiled with steam for 12 to 14 hours in lead-lined vats. In this way 66 per cent. of the arsenic, all the iron and bismuth, and most of the copper is dissolved out. After settling and washing, the liquor is removed, and the residual slime filter-pressed, dried, and melted.

The Chrome Plant of the United States Metals Refining Company.

By Lawrence Addicks

The property at Chrome, N. J., formerly known as the DeLamar Refinery, is now owned by the United States Metals Refining Company, which is a subsidiary company controlled by the United States Smelting, Refining and Mining Company. It forms the copper refining link in the metallurgical chain operated by the parent company. Since the

change of control two years ago the refinery has been brought up to 200 tons a day capacity, and a complete copper smelting plant with blast furnaces and converters has been added, arranged for rapid expansion into a still larger unit, if found desirable. The entire plant as it stands to-day represents the most modern practice.

Shipments can be received by rail, by lighter or directly by steamer, a large steamship dock having been erected in connection with the smelter. Cupriferous material of every nature from lean ore to bullion is received. A three-track trestle with an ore bin capacity of about 8000 tons affords ample storage for properly distributing the different ores to as nearly as possible a self fluxing mixture, it being the intention to bed the ores as is commonly done in lead practice.

The Smeltery.

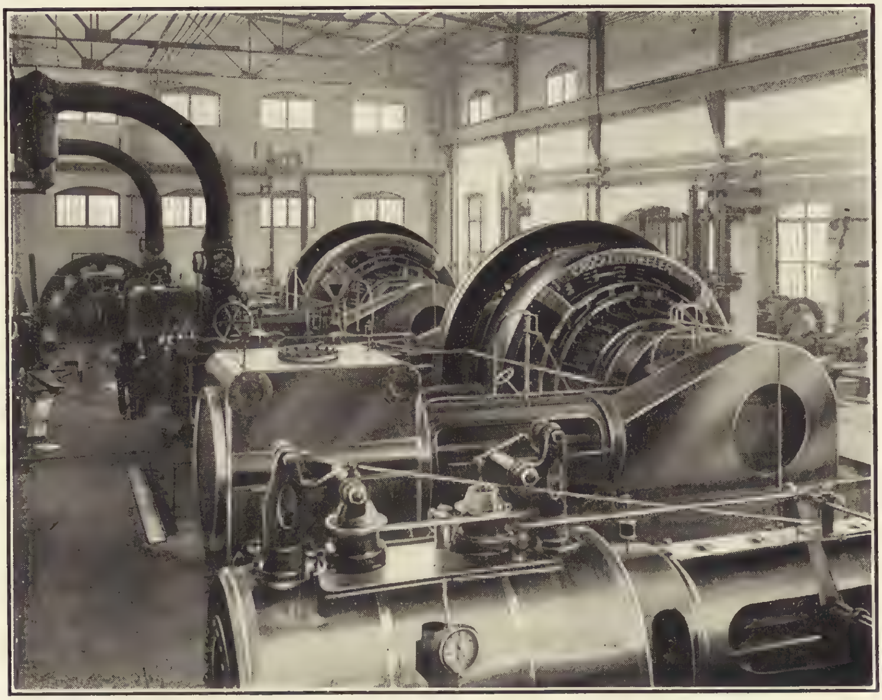
Ore Sampling.—At the head of the trestle is a well equipped sampling mill, designed to make the least fines consistent with accurate sampling. Ore is dumped from the car on a 5-in. grizzly and any lumps retained are broken by hand with a sledge. The grizzly delivers directly to a Brunton oscillating sampler, which cuts out one fifth which passes successively through three other samplers and appropriate crushing machinery, so that 1/625 of the original car load is delivered in 1/4 in. size upon the floor of the final sampling room, and 624/625 is returned as reject to the car from which it was taken and sent on to the bins. The first Brunton sampler which takes 5 in. lumps is the largest of the type that has been built. The capacity of the mill is conservatively rated at 100,000 lb. an hour.

From the ore beds the ore is transferred to an electric-driven charging-car carrying one trailer which runs on tracks at a lower level. The car and trailer extend just the width of the charging doors of the blast furnaces. The pair pass on over the scales to a Reedy electric elevator, installed in duplicate, and are raised with the motorman to the furnace charging floor and dumped directly into the furnace which is charged alternately from either side.

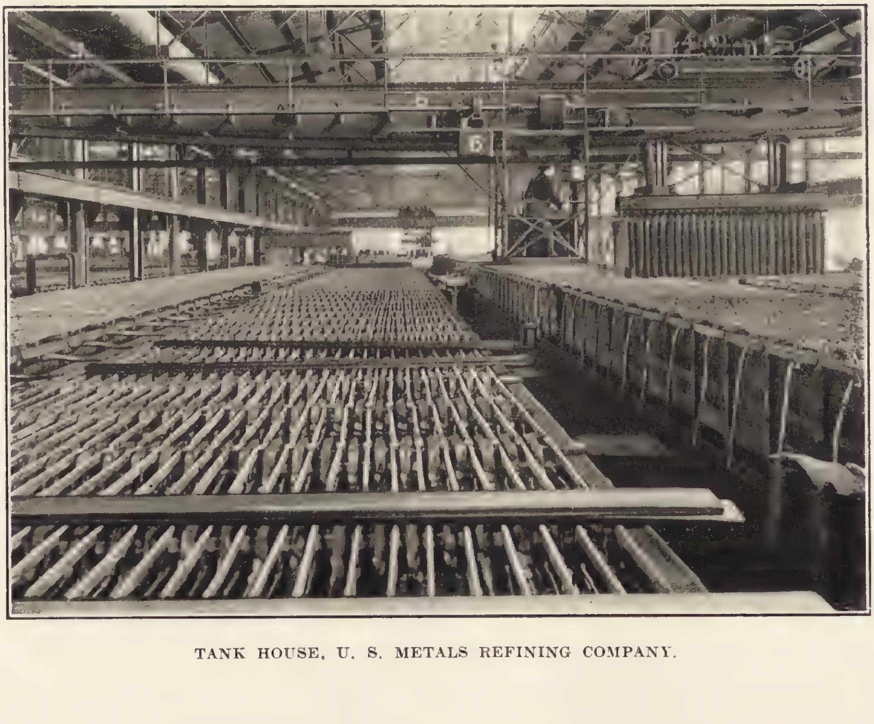
Blast Furnaces and Converters.—The blast furnaces are of the usual water jacketed type, the jacket sections being vertical. It is intended to install two large furnaces, but the plant is being started with a small 38 x 60 in. Traylor furnace, formerly in use at the refinery, and one large furnace. The building and equipment are all on the larger scale. Air is furnished at 32 oz. to the furnaces and jacket water is taken from the 5000-gal. circulating system in the refinery. The jackets were made by the Marine Boiler Works Company.

The blast furnaces produce matte assaying 50 per cent. copper, and a highly silicious slag. The latter is withdrawn in sectional cast steel

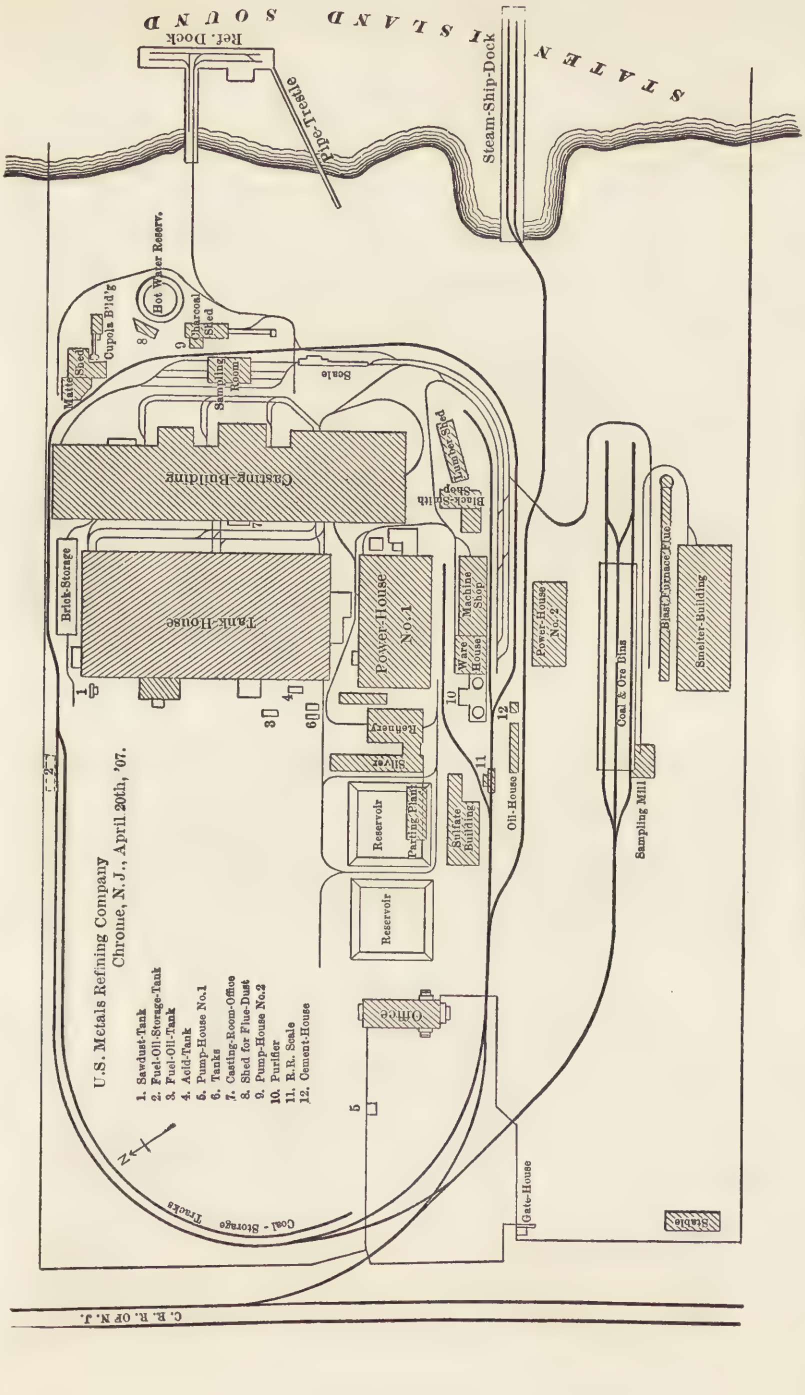
GENERAL VIEW OF SMELTING PLANT, U. S. METALS REFINING COMPANY, CHROME, N. J.



REFINERY POWER PLANT, U. S. METALS REFINING COMPANY.



TANK HOUSE, U. S. METALS REFINING COMPANY.



PLAN OF CHROME PLANT, U. S. METALS REFINING COMPANY

cars handled by electric locomotives. The matte is collected in 14 ft. settlers in the customary way and tapped out into ladles handled by a 40-ton crane and poured directly into the converters, of which there are at present two stands and six shells.

The converters are of the trough type 84x126 in., Allis-Chalmers, motor governed and supplied with air at 12 ℔. The blister copper is cast into bars and sent to the refinery, where it is treated just as outside bullion, a complete interaccounting of metals being made between the two plants.

Chimney and Flue.—Gases from the various units enter a long flue 12 x 13 ft. equipped with self discharging hoppers throughout, and finally pass into a 200 x 12 ft. Kellogg brick stack. No dust chamber has been erected, as the size of the flue is ample at present. The stack will easily handle four furnaces, and should these be erected later there is ample ground space for a dust chamber. The stack is furnished with a self-sealing discharge for flue dust and is built without a lining.

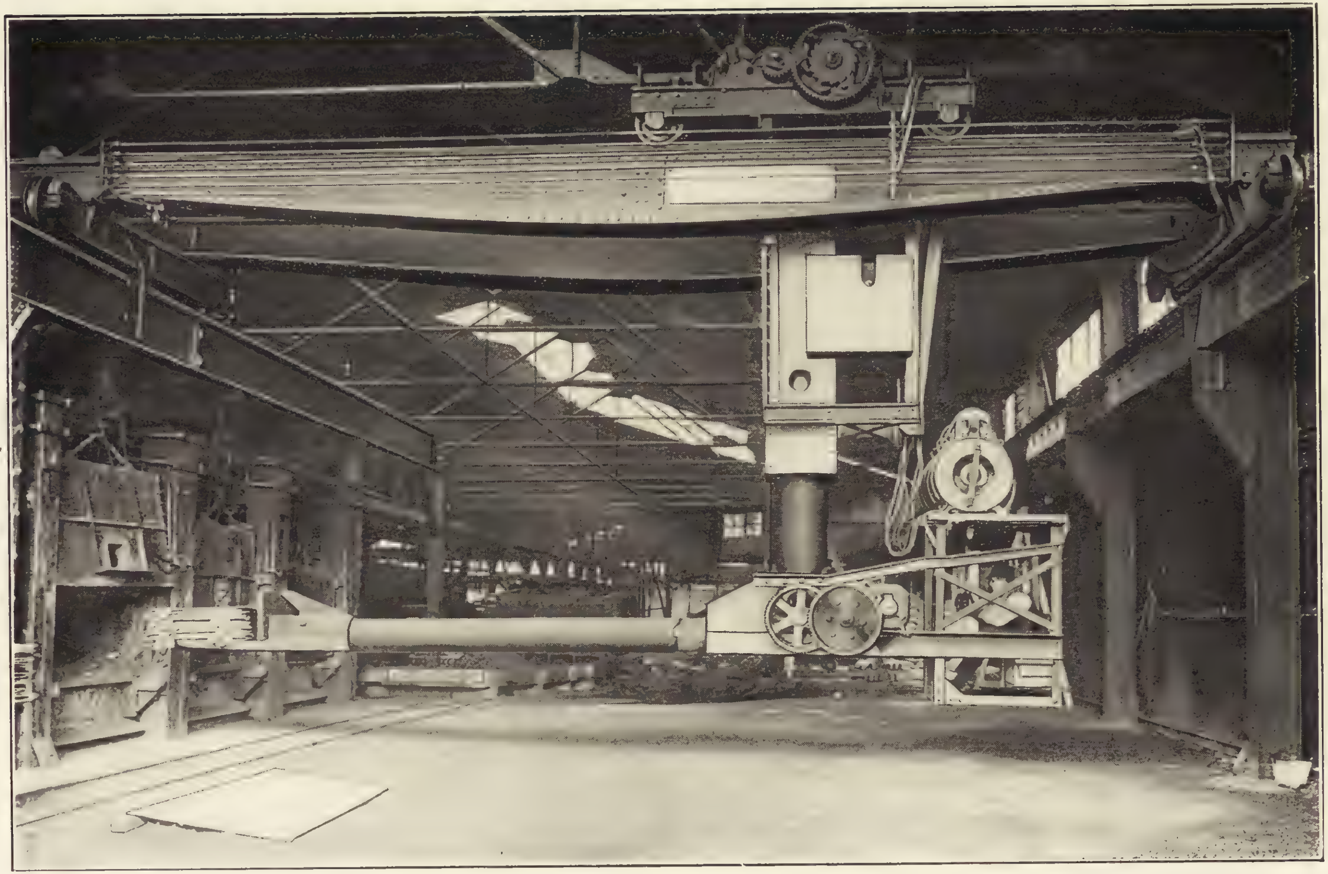
Power Plant.—Power for the smelter is supplied from a separate plant on account of lack of space for additional engines in the refinery power house. Steam and electric power, however, are taken from the refinery power house. Air for the blast furnaces is taken from a No. 9 Root blower, driven by a 11 x 24 x 36 in. Hamilton-Corliss tandem engine. The converters are supplied by a 13 x 28 in. and 30 x 30 x 42 in. Nordberg blowing engine. This power house also contains a 13 x 27 in. and 23 x 15 x 30 in. Ingersoll-Rand cross-compound two-stage Corliss air compressor, furnishing air at 80 ℔. pressure for general use, reserve units for this service being installed in the other power house.

The Refinery.

Sampling Pig Copper.—Following now the course of the converter bars into the refinery, they are taken on narrow gage cars by an electric locomotive to the pig-copper scales. Cars of five tons capacity are used. This gives a gage, which will take a curve of 25 ft. radius and also gives a car which can be pushed by hand, when desired, as all the cars are equipped with roller bearings. The scales are in duplicate, all material being weighed first on one, then on the other, one scale being used by the representative of the seller in the case of custom material.

From the scales the pig is taken directly to the sampling room, which is equipped with three high-speed drills and apparatus for rapidly handling the material, the capacity of each drill being 100 pigs an hour. As an average pig weighs 300 ℔. this gives a total capacity of 45 tons an hour.

Anode Furnaces.—From the sampling room the tracks lead directly to the anode furnaces, which are three in number. Two of them have a



CHARGING CRANE FOR CATHODE MELTING FURNACES.

DOCK AT CHROME PLANT, U. S. METALS REFINING COMPANY.

capacity of 100 tons in 24 hours, and the third 150 tons. Just short of 200 tons has been drawn from the large furnace in a single charge. This very large increase in size over the customary unit has been entirely successful. The furnace was built to be equipped with waste heat boiler and charging crane, but the arrangement is being tried out first on the wirebar furnace, which is an exact duplicate, and will be described in that connection. The two smaller furnaces are equipped with B. & W. type boilers. The anode copper is ladled into molds carried by a Walker casting wheel, each anode being 3 ft. square and weighing 600 ℔. As far as I know, this is the largest anode that has ever been used. The old plant had a capacity of about 115 tons a day, and when this was to be brought up to 200 tons, it was decided to build no new tanks but to increase the depth of the tanks 12 in., changing from electrodes 2 x 3 ft. to 3 x 3 ft. and increasing the current density about 20 per cent.

The anodes are cooled in a bosh of the same dimensions as a tank in the tank house and are kept in this space relation upon a special car and upon storage racks under the yard crane if not required at once in the tanks, so that all handling of anodes is mechanical and rapid.

Tank House.—The tank house contains 816 tanks in 24 groups of 34 tanks each, arranged in accordance with the Walker patent. The building is in three bays with two cranes in each bay. Twenty-six pairs of electrodes are used to a tank. At present, anodes are left in 28 days, and the cathodes fourteen. Experiments now being carried out indicate that the most economical relations will probably be 30 days for anodes and ten days for cathodes, as the present cathodes are exceedingly heavy, weighing about 300 ℔. apiece. The current density is about 21 amperes per square foot, which is higher than any other plant with the exception of the Boston & Montana. The normal current is 9500 to 10,000 amperes. The tank room is divided into three circuits of 272 tanks each. A current density of 500 amperes per square inch has been adopted for all bus bars on electrolytic work.

The slimes are sluiced through launders to a screen and sump from which they are pumped with a steam siphon to the top of the silver building. The circulation of electrolyte is accomplished by means of hard-lead centrifugal pumps in a pump house outside of the main building. The tanks are two in cascade and a circulation of about 4 gal. per minute per tank is maintained.

Wire Bar Furnaces.—The cathodes when unloaded are placed by a mechanical device in horizontal piles of 5000 ℔. each and sent to the wirebar furnaces, where they are picked directly from the car by the charging crane and put into the furnaces at the rate of 5000 ℔. a load. Charging cranes have been used for many years in connection with open hearth steel furnaces, but this is the first successful application to a copper refining

furnace. The design has been modified from the steel type, the load being pushed off the peel instead of being dumped from a shovel, and the furnace can be packed as tightly with cathodes as by hand. In view of the success of this experiment, it is intended to extend the use of the crane to all the furnaces. Patents on the arrangement have been applied for.

In order to accommodate the charging crane, it was necessary to build a furnace with one side practically all doors. Three doors 3 ft. 6 in. x 6 ft. 11 in. have been provided with only sufficient space between to afford room for a powerful buckstay. The hearth of the furnace is 14 ft. 5 in. x 34 ft. 3 in. and the walls are lined with magnesite brick to a point about two courses above the metal line, where a single course of chrome brick is laid and the wall continued with silica brick, the object of the chrome brick being to interpose a neutral layer between the basic and the acid brick. The bottom of the furnace consists of 9 in. of sand upon 9 in. of brick resting upon iron plates which are supported by brick piers.

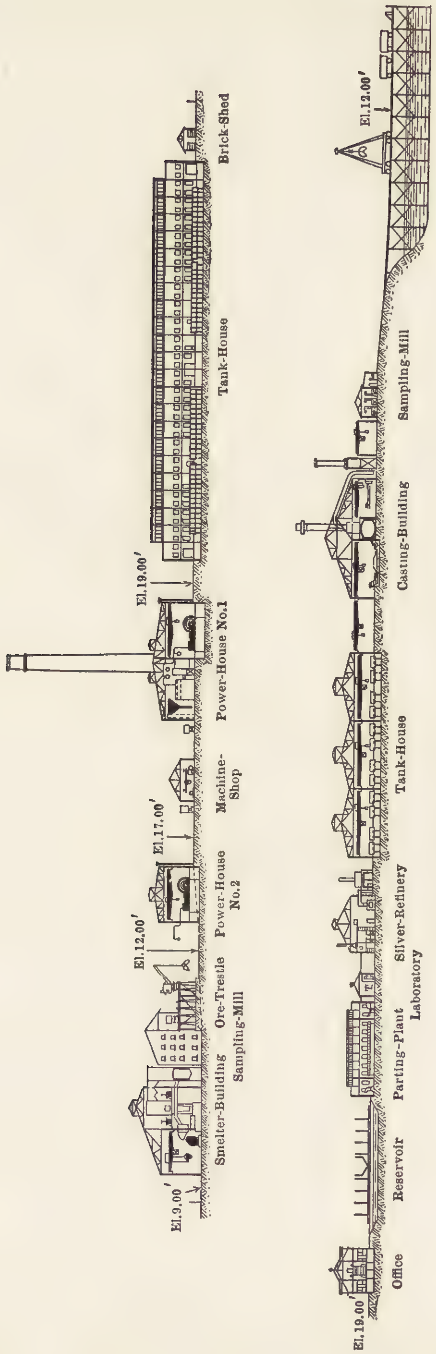
The gases pass up a 6-ft. brick-lined iron stack placed directly over the poling door without the usual connecting flue. The result is a marked improvement in the draft. The stack is carried by a tower of structural steel which straddles the end of the furnace. The staying is all direct, a yoke being passed around the stack to avoid the customary oblique stays. The fire box is stayed as a separate unit. The conker plate is water-cooled but all other parts are air-cooled.

The stack is capped by a damper and in this way closed off when the waste heat boiler equipment is in use. Then the gases are drawn from a point between the throat of the furnace and the bottom of the iron stack through a very short flue to a Worthington boiler. Draft is produced by a motor-driven fan. Evaporation tests have not yet been made on this unit but it is hoped to reach 6 lb. of water from and at 212 deg. F. per lb. of coal burned by the furnace.

The crane was built by the Morgan Engineering Works and is equipped with four motions, motor driven. It travels on overhead runways, passing the various furnaces. By recharging at night, which can be very rapidly done with the crane, 400,000 lb. can be brought round in one charge. In addition to the large furnace there are two 75-ton units which will probably be replaced by larger ones at an early date.

Power Plant.—The furnace coal and ash handling system is not yet completed at this writing. It is intended to handle all the coal through the roof trusses in electric larries; the ashes are to drop directly into cars which run outside the building through a tunnel to a shaft where they will be picked up by a yard crane and transferred to the narrow gage railroad system.

Anthracite buckwheat is burned under the boilers, run of mine bituminous in the refining furnaces, and fuel oil in the silver refinery. Coal is



SECTION OF CHROME PLANT, U. S. METALS REFINING COMPANY

stored in heaps by a locomotive crane, a method which has been found simple and economical. The loop tracks at the west side of the property give a storage of about 15,000 tons.

The buckwheat coal is unloaded at the power house from hopper bottom cars into an elevating and conveying system which delivers it to overhead bunkers in the boiler-room, carrying a day's supply, from which it is spouted directly to the boiler-room floor. The boilers are hand-fired, with forced draft supplied by fans. They are of the B. & W. horizontal type, furnished with McClave grates and carrying 150 lb. per sq. in. pressure. The flue gases are carried off by the natural draft of a 175 ft. brick stack 12 ft. in diameter. One economizer is installed and space is left for two more units.

Feed water is taken from the mains of the Middlesex Water Company and softened in a We-Fu-Go plant which delivers it to a suction well. Three outside-packed plunger pumps force it through primary and secondary closed heaters furnished by the Goubert Manufacturing Company, and then by way of the Green economizer to the boilers. Complete duplicate feed mains are installed and in addition there is a test service, feeding certain boilers next the superheaters. Large measuring tanks are provided and all piping and valves are arranged for frequent steam consumption tests on the various engines.

The steam is superheated in a separately fired Schmidt superheater to 475-500 deg. F., which has been found economical, after experiments ranging from the saturated temperature of 366 deg. F. to 750 deg. F. A duplicate system of headers is installed, arranged so that any engine or engines can run on either superheated or saturated steam and virtually forming a ring main. Cochrane separators are used to take care of priming, and all high pressure grips are handled by a Holly return system. Locke engine stops are used on all the larger units.

There are four electrolytic generating units, two 1500-h.p. Nordberg engines driving 1050-kw. Crocker-Wheeler generators, one 1125-h.p. Rice & Sergeant engine driving a 750-kw. Crocker-Wheeler generator and a 775-h.p. Hamilton-Corliss engine driving a 520-kw. Crocker-Wheeler unit. These are all horizontal, 100 r.p.m., full Corliss gear and valves, except the Rice & Sergeant, which has poppet valves on the high pressure cylinder, and the Hamilton-Corliss which runs at 105 r.p.m. Normally, three out of the four engines are in service and the 520-kw. unit is arranged to run in the place of any of the others.

The switches are of the multi-blade knife type, furnished by the Walker Electric Company, three 5000-ampere units being placed in parallel to give the full 15000-ampere overload capacity of the largest generators. Instead of circuit breakers, Eppensteiner engine stop relays are used. This is a relay attached to the ammeter shunt which operates the engine stops and shuts the engine down in case of a current overload. It was

developed at this plant and has recently been put upon the market under the above-stated name by the Weston Electrical Instrument Company. The bus-bars consist of two bars in parallel with a total cross-section of 20 sq. in.

Light and power are furnished by three 300 h.p. Nordberg-Corliss engines driving 200-kw. Westinghouse generators and a 100-kw. high-speed unit, all operating in parallel at 240 volts direct current.

The hydraulic system is quite elaborate. The condensing system consists of two 5000-gal. centrifugals, motor-driven, supplying three Worthington and one Alberger barometric heads with salt water. Two engine-driven 5000-gal. centrifugals handle the bosh system, circulating fresh water from reservoirs through the boshes and jackets and then through cooling sprays back to the reservoirs. A 50-lb. cold water, 100-lb. hot water and 180-lb. hydraulic service are also maintained. The 50-lb. is for miscellaneous general service; the hot water is from the condensation from the tank house heating coils; the hydraulic service is used in hydraulic cylinders. There is also a fire pump service. A number of the steam auxiliaries are arranged to operate either condensing or on back pressure, and a complete network of exhaust steam pipes is carried around the plant, live steam being replaced by exhaust wherever possible. Three air compressors furnish air at 80 lb. per sq. in. and air lifts and air-driven tools are quite generally used around the plant.

As an illustration of the varied service the modern works power plant is required to perform, it is interesting to note in tabular form the various pressures of air, steam and water actually carried. (See accompanying table.)

AIR, STEAM AND WATER PRESSURES.

Service.	Medium.	Pressure.	Service.	Medium.	Pressure.
Dry vacuum	Air	28 in. vacuum	Superheated	Steam	150 lb.
Induced draft	Gases	1 in. vacuum	Condenser	Water	10 lb.
Forced draft	Air	1 in.	Bosh	Water	20 lb.
Blast furnaces	Air	2 lb.	General service	Water	50 lb.
Converters	Air	12 lb.	Hot	Water	100 lb.
Condensing	Steam	28 in. vacuum	Fire	Water	150 lb.
Back pressure	Steam	5 lb.	Feed	Water	170 lb.
Live steam	Steam	150 lb.	Hydraulic	Water	180 lb.

Gold and Silver Refining.—The silver refinery has three departments, viz., the slimes, the furnace, and the parting. The slimes are delivered by the steam siphon pump in the tank house cellar to a receiving tank at the top of the slimes room. The usual process of boiling with sulphuric acid and niter and then filter-pressing, is employed. No agitating paddles are used, thereby keeping all machinery out of this room. The copper is reduced to almost a trace and in this way the slag returns to

the anode furnaces from the silver refinery kept down to a minimum. The filter press is made of hard lead plates on an iron frame and a lead lined cast iron egg is used.

In the furnace room are two 5 ft. 6 in. x 8 ft. reverberatories with a capacity of 100,000 oz. of doré each per charge. They cast one charge a week. The copper-slimes are melted and the impurities oxidized and slagged. It is an interesting commentary on the growth of copper refining to note that these furnaces, used for smelting by-product silver, are just about the size of copper-refining furnaces in use 25 years ago.

The doré is parted electrolytically in cells of the Thum type. A current density of about 50 amperes per square foot on 5 sq. ft. of anode surface per cell is used. The electrolyte is the usual silver-copper nitrate with small quantities of free nitric acid present. Power is taken from one of the tank-house circuits. The silver crystals are scooped up from the horizontal carbon cathodes, washed and melted in an oil-fired crucible furnace. The gold mud which is collected in canvas bags is boiled first with nitric and then with sulphuric acid and similarly melted. The various waste liquors are sent to the slimes boiling tanks. The parting plant was designed for an ultimate capacity of 1,000,000 oz. a month, so that outside doré can be parted.

Bluestone.—The sulphate building demands a word in passing. Here about 200,000 ℔. a month of blue vitriol is produced from tank house electrolyte and nickel sulphate is recovered from the mother liquors. The process is the customary oxidation with shot copper, concentration and fractional crystallization.

COPPERAS.

By Reginald Meeks.

The manufacture of copperas (sulphate of iron) has, of recent years, been carried on almost solely by iron and steel sheet and wire plants where it is obtained as a by-product in cleaning the sheets and wire. The Eastern Dynamite Company has suspended production of copperas at its Forcite works at Landing, N. J., and does not expect to resume operations. Wickwire Brothers, of Cortland, N. Y., have also shut down their copperas plant. The United States Steel Corporation was by far the largest producer in 1906, furnishing upward of 90 per cent. of the copperas made in this country. The production in the United States for 1895–1906 inclusive is shown in the accompanying table.

PRODUCTION OF COPPERAS IN THE UNITED STATES.
(In tons of 2000 lb.)

Year	Sh. Tons	Value.	Year.	Sh. Tons	Value.	Year.	Sh. Tons	Value.
1895	14,118	$69,846	1899	13,770	$108,508	1903	20,240	$121,440
1896	11,170	52,662	1900	12,374	96,517	1904	16,956	118,692
1897	11,924	56,565	1901	23,586	112,336	1905	21,103	147,721
1898	11,285	58,105	1902	19,784	118,474	1906	22,839	228,390

The above table includes only the copperas recovered as a by-product, and disregards the production as an intermediate product in the manufacture of venetian red by certain paint-makers.

Recovery of Copperas as a By-Product.[1]

To clean iron and steel from rust and scale it is immersed in a solution of acid in tubs or vats of shape and dimensions suitable to accommodate the material to be pickled. In tube works the pickling tubs are naturally long and narrow to accommodate the sections of tubing. In such works the material is pickled before annealing, in order that the latter process may have uniform and perfect results. Moreover, one of the last steps in manufacture is the passage of finished tubing through the pickling baths, after which it is rinsed and oiled, so that it may be shipped in perfect condition. In tin-plate, galvanizing and other work, where iron is coated with another substance, it is necessary that the iron should be clean and free from rust in order that the coating may properly adhere. It is therefore pickled and rinsed before coating.

[1] From "Steam Pollution by Acid-iron Wastes", by Herman Stabler, published by U. S. Geological Survey, Washington, 1906.

Pickling Bath.—The acids used for pickling are commercial grades of hydrochloric and sulphuric. Until sulphuric acid was cheaply produced, hydrochloric was used almost exclusively, but (except for special classes of work in which hydrochloric acid gives better results) it has now largely been superseded by sulphuric acid, a grade of 60 deg. B. (about 78 per cent. pure) being in most general use. The strength of solution varies somewhat with the work to be done. The best solvent for the sesquioxide of iron is a solution consisting of eight parts of sulphuric acid and three parts of water. A much weaker liquor is used in pickling, however, a 10 per cent. solution of the 60 deg. acid being a common strength for making up the fresh pickling bath.

Circulation is a primary requisite of good pickling. Were the material allowed to lie in a heap in the acid, some parts would be more thoroughly cleaned than others, the surface would become pitted, and the material would be seriously damaged in value. To avoid such results a rapid circulation of the liquid or a movement of the material pickled is necessary. This is customarily accomplished by machinery which automatically alternately thrusts the material into and draws it from the bath, or which forces the liquid around the material by a plunger. The pickling solution is kept at a temperature ranging from 140 deg. F. to a little more than 212 deg. F. in order to increase the activity of the acid. This is usually done by forcing steam into the bath from open pipes. Circulation and heating are accomplished by closed steam pipes of acid-proof material laid in the bottom of each tub. The liquor is thus heated to a state of violent ebullition, satisfactory circulation is obtained and rapid and thorough pickling made possible.

As previously stated, the fresh pickling bath contains about 10 per cent. of acid. As the process of pickling continues, the amount of free acid becomes less and less because of the combination of acid with iron. More acid is therefore added in a continuous stream, or by charges, in order to keep the bath working actively. Finally, when the liquor contains about 15 per cent. of iron sulphate and 2 per cent. of acid it is either discarded as useless, or else copperas is recovered.

Spent Pickling Liquor.—In 1230 gal. of discarded pickling liquor 1230 lb. of SO_3 were found, representing a value of $14 in acid used. This existed chiefly as ferrous sulphate, some ferric sulphate and about 2.5 per cent. of free sulphuric acid. About 2650 lb. of SO_3, or nearly two-thirds (more exactly, 65.8 per cent.) of the active principle in the acid used, is lost in the pickling tubs. Although a part of this passes off as fumes, it is presumed that nearly all this loss is due to splashing and leakage of the tubs and drainage from the pickled material.

Copperas Recovery.—A plant for the recovery of copperas was installed in August, 1904, at the works of the Shelby Tube Company, at Shelby, O.

This plant is exceedingly simple in construction and operation. The spent pickle is drained by gravity into a lead-lined wooden receiving or storage tank of rectangular cross section, having a capacity of about 6000 gal. From the storage tank the liquor is drawn by charges into an evaporator for concentration. The evaporator is a wooden tub of rectangular plan and trapezoidal elevation, having a capacity of a little more than 2000 gal., which is lined with ⅛-in. lead and heated by a steam coil. It is provided with a wooden hood and a draft for carrying off the vapor.

Iron or steel turnings are introduced into the evaporator to neutralize the free acid and to reduce ferric to ferrous sulphate. The liquor is concentrated to about 45 deg. B. (sp.gr. 1.453; Tw. 90.6 deg.) and then drawn off into wooden rectangular crystallization tubs, each of 675 gal. capacity. To assist in the crystallization and to provide a support for the crystals, wooden strips are hung in the tubs. The dependent crystals are gathered, drained, and barreled for shipping. The mother liquor, together with the more impure crystalline copperas from the sides and bottom of the tubs, is pumped back by hand to the storage tank or evaporator. Occasionally such matter is discharged into the stream with wash water from the tubs to dispose of accumulated impurities.

Analysis of the crystals produced gave the following results in per cent.: Acidity as H_2SO_4, 0.2; sulphuric acid as SO_3, 27.5; iron as Fe, 18.9. This approximates the formula $FeSO_4$, $7H_2O$ for ferrous sulphate crystals, but indicates the presence of small amounts of ferric sulphate and free acid.

Average figures for several months show that the copperas produced is, by weight, 50 per cent. of the acid used in pickling. For 6350 ℔. of acid used a product of 3175 ℔. of copperas would be obtained, having a content of 875 ℔. SO_3. This is 21.7 per cent. of the SO_3 in the acid used for pickling, or 71.1 per cent, of the SO_3 in the spent pickle.

There is therefore a loss—presumably taking place by leakage in the evaporator, storage tank, and crystallization tubs—of 8.8 per cent. of the active principle of the acid used for pickling, or 28.9 per cent. of the amount of such principle present in the spent pickle.

The estimated disposal of the SO_3, assuming an average daily use of 5000 ℔. of acid, may be summed up as follows:

LOSSES AND RECOVERY OF SULPHURIC ACID.

	Per Cent.	Lb. per day		Value as 60° acid.
		Acid	SO_3	
Loss in rinse water	3.7	185	144	$1.34
Loss in pickling tubs	65.8	3,290	2,566	23.80
Loss in copperas plant	8.8	440	343	3.18
Recovered in copperas	21.7	1,085	847	7.85
Total	100.0	5,000	3,900	36.17

Cost of Production.—Original estimates on cost of recovery per ton of product were as follows: Coal at $2.25 per ton, $1.08; iron turnings, at $13 per ton, $0.89; barrels, etc., for shipping, $1.90; total $3.87. It is thought that $2 will more nearly approximate the actual cost of fuel for evaporation with the evaporators used, although, owing to complication of heating arrangements, no exact figures are obtainable.

Accepting the other items as correct and adding $1 per ton for labor and $0.50 per ton as a 10 per cent. annual depreciation upon a plant valued at $2000, we have, for the production of one ton of copperas, the following figures: Average value of copperas (Twelfth U. S. Census) $9.64; fuel, at $2.25 per ton, $2; iron turnings, at $13 per ton, $0.89; miscellaneous supplies, $1.90; depreciation, $0.50; labor $1; total, $6.29; profit $3.35.

This is a very liberal estimates of costs, and will, without doubt, be more than sufficient to cover the actual conditions at Shelby. Were the plant arranged to save the liquor now lost and a better evaporator used, a much better showing could be made. It must be considered that the advantageous arrangement of the plant for gravity flow contributes in some measure to the low cost of the product. Where pumping is required, the cost would not in any case be more than $6.64 per ton, other items of cost remaining as above, leaving $3 per ton as a low estimate of profit under extremely adverse conditions.

Other Processes.—Several other processes for the disposal of sulphuric-acid pickle have been tried experimentally and a few have been applied on a commercial scale. The most prominent of these and their results are as follows: Electrolytic regeneration of acid, expensive; recovery of basic ferric sulphate and acid by aeration, expensive; recovery of "sugar copperas," practicable; chemical precipitation, expensive. So far as known, recovery of copperas in large crystals and as "sugar copperas" are the only processes that will pay for the cost of disposal of the spent pickle from sulphuric acid.

For spent pickle from hydrochloric acid containing iron chlorides a cycle of regeneration has been perfected in England by Thomas Turner. The method is described at some length by Naylor in "Trades Waste," pp. 262-263. In brief, the pickle is fed into a reverberatory furnace; iron oxide is retained and hydrochloric acid passes over with the fire products and is condensed in ordinary muriatic towers. The cost of 75 per cent. acid recovered is stated at about $2. In the United States it would doubtless be somewhat higher.

Uses of Copperas.

Sulphate of iron is used by packing houses in the coagulation of blood and mixing with waste products or tankage in manufacturing fertilizers.

It enters into the manufactures of all fertilizers where iron in some form is required, and is particularly adapted as a fertilizer in orange cultivation in California and Florida. Sulphate of iron also enters into the manufacture of cattle and poultry food, hog cholera cure and a number of medicines and tonics for the human system. It is used in the dyeing trade as a mordant for silks and in the manufacture of black ink. Also it is used largely for the manufacture of rouge (ferric oxide) and venetian red.

Water Purification.—If instead of sulphate of aluminum, sulphate of iron (copperas) is added to a turbid water containing bicarbonates of lime and magnesia in solution the sulphuric acid associated with the iron forms soluble sulphates with the lime and magnesia.[1] The iron reacts with the carbon dioxide of the carbonates to form a rather soluble bicarbonate of iron. The iron in both this compound and in the applied copperas is in the ferrous form, and there is a strong tendency to become oxidized to the ferric condition by the action of the oxygen dissolved in the water. The bicarbonate of iron gradually oxidizes and at the same time decomposes into ferric hydroxide and carbon dioxide, the latter being dissolved in the water and the former precipitating and coagulating.

The chief objection to this method of water purification is its slow reaction and this is remedied by adding caustic lime, or other alkali, which removes some of the carbonic acid and causes the insoluble ferric hydroxide to precipitate more rapidly.

The iron and lime process is the outcome of two old processes, viz. the Jewell sulphur and iron process and the Anderson process. The treatment of very soft water by the copperas and lime method requires much skill and a further objection against the process is that two chemicals are required instead of one as in the sulphate of aluminum process. The process presents the physical advantage of rapid sedimentation in turbid waters and has been in successful operation in several plants in the Middle West. Another point in its favor is the lower cost as compared to the sulphate of aluminum method, and this difference becomes more marked as the amount of sediment to be removed becomes greater.

Bibliography.

The statistics, trade and ordinary uses of copperas have been treated in The Mineral Industry, Vols. IX to XIV, inclusive. The method of producing copperas was described briefly in Vol. X; its use for water purification in Vol. XII; and the effect of copperas on plant growth, according to experiments conducted in France, was described in Vol. XII.

[1] From a paper read before the Central Water Works Association by Joseph A. Ellms.

CORUNDUM AND EMERY.

By Reginald Meeks.

The deposits in the United States furnished only a small part of the total corundum and emery consumed in this country in 1906, as in previous years. The following table shows the production and trade in these abrasives:

STATISTICS OF CORUNDUM AND EMERY IN THE UNITED STATES.

Year.	Production. (a)		Imports.				
			Grains.		Ore and Rock		Other Mfrs.
	Short Tons.	Value. (b)	Pounds.	Value.	Long Tons.	Value.	Value.
1897	2,193	$111,810	520,095	$20,022	5,209	$107,644	$2,211
1898	3,742	207,430	577,655	23,320	5,547	106,269	3,810
1899	3,970	228,570	728,229	29,124	7,435	116,493	11,514
1900	5,030	247,100	661,482	26,520	11,392	202,980	10,006
1901	4,305	146,040	1,086,729	43,217	12,441	240,856	10,926
1902	4,251	104,605	1,665,737	49,107	7,157	151,959	13,776
1903	2,542	64,102	3,595,239	109,272	10,884	194,468	17,829
1904	1,932	57,235	2,281,193	109,772	7,054	138,931	11,721
1905	(c)2,315	19,677	3,209,914	143,729	11,072	185,689	17,996
1906	(c)2,147	22,780	4,655,168	215,357	13,840	286,386	19,105

(a) Statistics of the United States Geological Survey for 1901–1903. (b) Values have not much significance owing to the wide variation in the quality of the materials combined in the totals. (c) Emery only.

Theoretically pure corundum contains only alumina, Al_2O_3; but almost invariably there are found small quantities of silica, ferric oxide and water. There are three kinds of commercial corundum, which are determined by the physical structure. The first is known as "block" and includes massive corundum, whether pure or associated with feldspar, muscovite, hornblend, etc. In some cases the separation from these foreign minerals is accomplished with difficulty. When free from impurities block corundum furnishes the best quality of mineral. The second variety includes all crystal forms of corundum, whether present in deposits of sand or block corundum; the third consists of small irregular grains and minute crystals.

Corundum Mining in the United States.

Georgia.—Deposits of corundum are found scattered through this State in a northeast direction. The mineral occurs in peridotite forma-

ions and has a good reputation. There are only two producers, one which is the Laurel Hill mine (owned by the International Corundum and Emery Company), situated at Pine Mountain, Rabun county.

North Carolina.—There were no new developments in the corundum industry during 1906. The mines continued in regular operation. The product from the Corundum Hill mine in Macon county forms a considerable portion of the output of the United States. Other mines, in North Carolina are: Sheffield and Reed, or Watauga, in Macon county; Buck Creek or Cullakeenee, Herbert, Isabel, Behr, and the Blue Ridge mines, in Clay county; Burnt Rock, Sapphire and Bad Creek, in Jackson county; and the Carter, in Madison county. Most of these deposits are smaller and unimportant.

Corundum in Foreign Countries.

Australia.—A discovery of corundum is reported in the ranges which divide the watershed between Lake Eyre and Lake Frome in the Farina district, South Australia, approximately four miles east of Mt. Pitts, on an unnamed creek. The rock formations consist of gneissic metamorphic silicious schist, quartzite, mica schist, and mica rock, trending east and west, and dipping at angles varying from vertical to 50 deg. to the south, varying in color from black to yellow and grey, and is massive and tough in structure. The corundum occurs in the form of segregated crystalline lumps, rough hexagonal crystals, and small irregular shaped pieces disseminated through the rock. In color it is blue, mottled and white and greenish and is associated with minute red crystals, chiefly rutile, which occur both in it and in the matrix. In some places the corundum content averages from 10 to 25 per cent. of the rock matrix.

Canada.—The production of corundum in Canada has been steadily increasing since mining began in 1900. The output in 1906 was 2274 tons, valued at $204,973, as against 1644 tons, worth $149,153, in 1905. The Canada Corundum company at Craigmont, Ont., produces

THE CORUNDUM INDUSTRY OF CANADA.

Schedule.	1900	1901	1902	1903	1904	1905	1906
Production, tons	60	534	1,137	1,119	1,665	1,644	2,274
Value	$6,000	$53,115	$83,871	$87,600	$150,645	$149,153	$204,973
Number of men	35	68	95	186	202	....	....
Wages paid	$10,000	$30,406	$34,674	$106,332	$139,548	....	....

about 300 tons monthly, and expects to enlarge its capacity soon. It is also likely that other properties will be more energetically developed, as the old regulations with regard to corundum lands and the establishment of works in Ontario have been revoked. The special agreements

made in 1889 and 1890 with J. N. Shenstone and B. C. Craig; in 1903 with the Canada Corundum company, and in 1904 and 1905 with the Corundum Refiners, Ltd., continue as heretofore.

The Hart Corundum Wheel Company and the Canadian Corundum Company, both of Hamilton, Ont., have amalgamated under the title of the Canadian Hart Corundum Wheel Company, Ltd., with a capital stock of $75,000. The Ashland Emery and Corundum Company, which operate the Burgess mine in Carlow township, Hastings county, Ont., employing about 35 men, had six concentrators in operation separating iron and corundum from spar by the process of dry concentration. The iron is then eliminated by a revolving magnet. Early in 1906 the company ordered $20,000 worth of new machinery.

Emery.

This material is the intimate mechanical admixture of corundum and either magnetite (Fe_3O_4) or hematite (Fe_2O_3). Its value for abrasive purposes depends upon the percentage of corundum contained in the mixture. Cases are on record where deposits of emery have been mistaken for iron ore, and blast-furnaces were erected, but smelting failed utterly.

All of the emery produced in the United States comes from either the deposits at Chester, Mass., operated by the Ashland Emery and Corundum Company or from the mines at Westchester, N. Y.

The trade-names for emery are Turkish, Naxos, Chester, and Peekskill, according to the locality from which it comes. It is supplied in the form of grains and flour and commands a price from 1¼@1¾c. per lb. for flour and from 2¼@4½c. for grain, depending upon quality and brand.

Massachusetts.—The only deposit of emery found in this State, with the exception of a prospect at Huntington, is in the vicinity of Chester. Of recent years the production from this district has greatly diminished and has been surpassed by that of the mines at Westchester, N. Y. For a detailed description of the emery industry at Chester, Mass., see THE MINERAL INDUSTRY, Vol. XIV.

New York. (By D. H. Newland).—The emery deposits near Peekskill were worked on about the usual scale during 1906. The output reported by the four producers that were active amounted to 1307 short tons, valued at $13,870. This compares with a total of 1475 tons valued at $12,452 in 1905 and 1148 tons, with a value of $17,220 in 1904. The valuation is based on the crude material as it comes from the quarries where it undergoes only a rough sorting before shipment. The lump emery is sent to outside points for grinding and manufacture into emery wheels, stones, cloth, etc.

The list of producers in 1906 includes the following: Blue Corundum Mining Company, of Easton, Penn., Keystone Emery Mills, Frankford,

Penn.; J. R. Lancaster, Peekskill, N. Y.; and the Tanite Company, Stroudsburg, Penn. The Keystone Emery Mills took over the properties formerly worked by H. M. Quinn. With the exception of J. R. Lancaster who sells his product to the Hampden Corundum Wheel Company, of Springfield, Mass., the companies mine the emery for their own consumption in connection with manufacturing plants.

Determination of Corundum.

The material is crushed in an iron mortar to pass a 14-mesh sieve and sampled. Treat two grains with concentrated HCl on a water-bath for two hours. Filter, dry and fuse the residue for 30 minutes with six grams of carbonate mixture (Na_2CO_3 two parts, K_2CO_3 one part) in a platinum crucible over a Bunsen burner. Digest with hot water and decant through a filter. The residue still remaining is treated with a large excess of dilute HCl and this solution is decanted through the same filter. The filter paper is dried and ignited in a platinum dish, the residue from the treatment with hydrochloric acid is added and the whole treated with hydrofluoric acid. Evaporate excess of HF and treat residue with hot water, finally transferring to an ashless filter paper. Dry, ignite in a weighed crucible, cool and weigh. The increase in weight is calculated as pure corundum.

CRYOLITE.

Cryolite is a double fluoride of sodium and aluminum and occurs in commercial quantities only in one known locality, namely, Ivigtut, South Greenland. It finds its chief uses in the manufacture of sodium salts, the manufacture of certain special kinds of porcelain and glass, and as a flux in the electrolytic aluminum process. Calcium fluoride results as a by-product in the process of making the sodium salts, and is saved and sold for use as a flux in open-hearth steel furnaces. In this form it is a substitute for fluorspar.

The cryolite deposit at Ivigtut is owned by the Danish government which leases it to the company that operates the quarry. Cryolite is discussed in Vols. I–IV of THE MINERAL INDUSTRY. The chief points of interest of those articles are summarized as follows:

In Vol. I there is a short discussion of the history of the discovery of cryolite in Greenland and a brief account of the methods of mining and shipping the product to the United States. In Vol. II the method of decomposing the crude mineral as practised in the United States and in Denmark is given, and the equipment of the Pennsylvania Salt Manufacturing Company is described. In Vol. III there is given a brief description of the Greenland deposit as to its situation in regard to shipping, and also a more complete account of the methods of mining.

The imports of cryolite into the United States during the last 10 years are given in the accompanying table:

IMPORTS OF CRYOLITE IN THE UNITED STATES.

Year.	Long Tons.	Value.	Av. per Ton.	Year.	Long Tons.	Value.	Av. per Ton.
1897	10,115	$135,114	$13.36	1902	6,188	$85,650	$13.84
1898	6,201	88,501	14.27	1903	7,708	102,879	13.35
1899	5,879	78,676	13,38	1904	959	13,708	14.30
1900	5,437	72,763	13,37	1905	1,600	22,482	14.05
1901	5,383	70,886	13.17	1906	1,505	29,683	19.72

FELDSPAR.

Feldspar is produced in the United States in Pennsylvania, Maryland, Connecticut, Maine and New York, which rank in importance in the order mentioned. Statistics of production during recent years are given in the following table:

FELDSPAR IN THE UNITED STATES. (*a*)
(In tons of 2000 lb.)

Year.	Crude		Ground.		Total.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
1901	9,960	$21,699	24,781	$198,753	34,741	$220,422
1902	21,870	55,501	23,417	194,923	45,287	250,424
1903	13,432	51,036	28,459	205,697	41,891	256,733
1904	19,413	66,714	25,775	199,612	45,188	266,326
1905	14,517	57,976	20,902	168,181	35,419	226,157
1906	....	....	....	....	72,656	401,531

(*a*) Statistics of U. S. Geological Survey.

Feldspar is a double silicate of aluminum and either sodium, potassium or calcium, known respectively as albite, orthoclase and anorthite. Orthoclase is the most abundant and occurs in the purest form. Feldspar is used chiefly as a flux for binding the different constituents of pottery in china and porcelain factories. It is also used for glazing china ware. In order to be of value in pottery work, feldspar should be free from iron oxide, mica and quartz. The last mineral, however, is not particularly objectionable, provided it is present in uniform amounts.

Market and Prices.—From January to July, 1906, the best ground feldspar was quoted at $9.75 @ $12 per ton. From August to December the price was $15 @ $20 per ton, and through December it was $12 @ $15 per ton. Common feldspar was steady throughout the year at $10 @ $12. These quotations relate to ton lots at New York. Smaller amounts are somewhat higher in price. Prices of crude feldspar at the mines is much lower and has averaged during recent years as follows: In 1901, $2.20; in 1902, $2.50; in 1903, $3.80; in 1904, $3.45; in 1905, $4. In 1906 the average was $4.10.

Industrial Conditions.—The recent tendency of the feldspar industry is strongly in the direction of concentration, the various small operators and grinders throughout the producing districts having leased their properties to larger firms, by which the industry has been much consolidated. In Pennsylvania, the Brandywine Summit Kaolin & Feldspar

Company leases property at Brandywine Summit and at Taugh Kennamon. The Eureka Flint and Spar Company, of Trenton, N. J., has likewise leased deposits in Pennsylvania, New York and Connecticut.

The Consolidated Feldspar Company of Middletown, Conn., is constructing a feldspar grinding plant of large capacity; it is hoped that it will be in operation in July, 1907. At Henryton, Md., there are deposits of feldspar upon which experiments have been made with a view to ascertaining their value as fertilizers for the tobacco industry of Connecticut. At the present time, carbonate of potash, which is a great tobacco fertilizer, brings at its port of entry $98 a ton, which corresponds to the price of 7c. per lb. for potash, but the low cost of grinding feldspar will make the potash contained therein available at a price of about 1.5c. per lb. if the results of successful preliminary tests upon the suitability of feldspar for fertilizing purposes are confirmed. The company owning the Henryton deposit contemplates the erection of a grinding mill.

Bibliography.

Reports on feldspar were included in The Mineral Industry, Vols. I, II, V, VII to XI, inclusive, and XIII. Vol. I describes briefly the composition and sources of feldspar and its uses in the manufacture of glass and pottery. Vol. II contains the most detailed account of feldspar in an article of 8 pp., by T. C. Hopkins, who describes the occurrence of feldspar and its mineralogical features and composition, uses, and methods of mining and preparing it for market. The distribution of feldspar throughout the United States is taken up state by state and the various deposits are enumerated. The other volumes give only short statements as to the production of feldspar in the United States, industrial conditions, market and prices.

FLUORSPAR.

By F. Julius Fohs.[1]

Western Kentucky and southern Illinois continue in the lead as producers of fluorspar, with central Tennessee and Colorado producing small amounts in 1906. While there was a larger amount marketed in 1906 than in 1905, the amount mined was less, due to a deficiency of production in southern Illinois, where the shipments exceeded the production by more than 5000 tons. The Kentucky production and shipments both fell below those for Illinois for the first time tince 1897, though

STATISTICS OF THE FLUORSPAR INDUSTRY IN 1905 AND 1906.
(In short tons.)

District.	Mined.	Ground Marketed.	Lump Marketed.	Gravel Marketed.	Total Marketed (a)	In Stock Dec. 31.	Companies.		
							Developing	Mining	Marketing
1905.									
Colorado and Tennessee ..	1,756	....	1,676	....	1,676	80	0	2	2
Southern Illinois..........	16,602(b)	2,313	3,816	9,206	15,335	6,762(b)	4	4	6
Western Kentucky........	20,560	5,220	2,470	11,461	19,151	10,583	2	12	4
Total..................	38,918	7,533	7,962	20,667	36,162	17,425	6	18	(c)10
1906.									
Colorado and Tennessee ..	620	....	660	....	660	40	1	2	2
Southern Illinois..........	19,532	3,995	4,533	16,053	24,581	2,975	4	6	8
Western Kentucky........	11,440	4,319	904	6,570	11,793	10,455	3	10	6
Totals.................	31,592	8,314	6,097	22,623	37,034	13,470	8	18	(c)14
Increase in 1906..........	....	781	....	1,956	872	....	2	0	4
Decrease in 1906..........	7,326	....	1,865	....	....	3,955	......	......	......

(a) This represents gravel shipped, of which less than half was mined gravel, the remainder having been mined as lump and crushed while all the ground was mined as lump. (b) Based partly on returns and partly on shipments. (c) The apparent errors in totals are not real, since companies shipping from two states were deducted.

Kentucky still ranks first in the production of ground fluorspar. In Kentucky, five shafts not regularly producing heretofore will be capable of supplying in 1907 more than the entire Kentucky production in 1906, while the old mines will be capable of turning out a larger amount of fluorspar than ever before. It may be seen therefore that the decrease in Kentucky production in 1906 is to be attributed not to lack of fluorspar,

[1]This review of the American fluorspar industry is abstracted, by permission of the Director of the Kentucky Geological Survey, from a bulletin by me, now in press, on "The Fluorspar Industry", to which the reader is referred for further details and a fairly complete exposition of everything relating to the subject. F. J. F.

but rather to the regulation of the supply so as not to exceed the demand; and further that Kentucky mines will be capable if called upon to supply a larger demand than at present.

There was marketed from American mines in 1906, 37,034 short tons of fluorspar valued at $211,231. This is an increase of 872 tons over 1905, although the value of the product in 1905 was about equal to that of 1906. Of the 1906 product, 8314 tons valued at $82,576 was ground mineral, 6097 tons valued at $34,398 was in the lump form, while of gravel there was 22,623 tons valued at $94,257.

FLUORSPAR OUTPUT OF THE PRINCIPAL PRODUCING COUNTRIES.
(In metric tons.)

Year.	France.	Germany.	Spain.	United Kingdom.	United States.
1897	2,722	23,232	2	302	3,973
1898	3,077	23,787	5	57	11,021
1899	5,140	24,306	310	796	21,806
1900	3,430	30,310	4	1,471	19,646
1901	3,970	28,741	*Nil.*	4,232	17,768
1902	2,650	(*a*)14,177	93	6,388	47,190
1903	2,447	(*a*)13,028	4,000	12,102	38,577
1904	2,047	(*a*)13,540	(*b*)	18,451	33,069
1905	2,434	(*a*)15,019	(*b*)	39,446	35,299
1906	(*c*)	....	(*c*)	36,280	28,657

(*a*) Exports. German statistics no longer report production. (*b*) Not reported. (*c*) Not yet available.

The production in 1905 reported in THE MINERAL INDUSTRY, Vol. XVI, and also that reported by the U. S. Geological Survey, was too high, because of the duplication or exaggeration of some returns from Illinois. The error has been corrected in the tables given with this article. The figures of shipments for 1906 are based on returns from the transportation companies carefully checked with the reports of producers. The reason for Kentucky shipments dropping to the second place may be summed up as follows: (1) Advantage of $1 per ton in freight rates in favor of the Illinois shippers. Rebates also are reported as being given in addition to this. (2) Illinois operators having cut prices in order to compete with the English, and also to undersell Kentucky operators. (3) The further invasion of the English shippers into the Western and Southern trade, previously largely controlled by Kentucky shippers. (4) The invasion of Illinois operators for the first time into Southern trade. (5) The partial retirement temporarily from the market of one of the large shipping companies, owing to caving of one of its Kentucky mines; hence, inability to produce.

The amount of galena concentrates secured as a by-product in fluorspar milling in 1905 in western Kentucky was 72 tons, which in 1906 amounted to 91 tons. In 1905, the amount shipped from southern Illinois was somewhat more than for west Kentucky, while in 1906 the galena shipments from southern Illinois amounted to about 430 tons, with a reported silver

content of 3.5 to 11 oz. per ton. The estimated amount of lead ore on hand at the end of 1906 in the same district was 170 tons. The zinc ore, baryte and calcite shipments from western Kentucky in 1906 were not strictly by-products of fluorspar mining or milling.

PRODUCTION OF FLUORSPAR IN THE UNITED STATES.
(In short tons)

Year.	Tons.	Value.	Per Ton.	Year.	Tons.	Value.	Per Ton.
1897	4,379	$36,264	$7.65	1902	48,018	$271,832	$5.19
1898	12,145	86,985	7.16	1903 (*a*)	42,523	213,617	4.28
1899	24,030	152,655	6.35	1904 (*a*)	36,452	234,755	6.44
1900	21,656	113,430	5.24	1905	39,600	232,452	5.87
1901	19,586	113,803	5.81	1906 (*b*)	34,683	201,481	5.78

(*a*) Statistics of the U. S. Geological Survey. (*b*) The statistics here reported for 1906 were collected independently for *The Mineral Industry*, and differ slightly from those collected by Mr. Fohs and used by him elsewhere in his article.—*Editor.*

Market.—The prices per short ton of fluorspar received in 1906 by American companies, in carload lots of 15 tons or more, were as follows: Ground $8.90 @ $15; lump $3.85 @ $8.70; gravel $2.50 @ $6. The differences in prices between American grades No. 1, 2 and 3 crude, and also between the different grades of ground, vary from 50c. to $3 per ton. Fluorspar of similar grade costs from 50c. to $1.50 more in lump form than as gravel.

Fluorspar for foundry purposes, in quantities from 50 pounds to ton lots, usually barrelled, was retailed in 1906 as follows: Crude, $10 @ $20 per ton; ground $20 @ $32 per ton. The smaller glass and enamel companies also purchase at times in small lots of from 0.5 to five tons. One company reported the sale of a ton of specimen fluorite at $60.

The average prices received for American fluorspar per short ton in 1906 were: Ground $9.93; lump $5.64; and gravel $4.16. The average price of all grades was $5.70.

Though commonly shipped in closed cars, some fluorspar was necessarily shipped in coal cars owing to car shortage in latter part of 1906.

The sources of competition of American fluorspar are three in number: (1) Limestone flux, (2) imported fluorspar, (3) cryolite and cryolite products. Owing to detergent and more powerful fluxing properties, the use of fluorspar has shown a steady but notable increase as compared with limestone flux for several years past. English fluorspar, of which about 30,000 tons are now imported yearly, is a keen source of competition, especially in the lower grades, but during 1906, American producers entered into active competition with the English, and the fight will continue earnestly in 1907. Imported cryolite, and its by-product, calcium fluoride, formerly a noteworthy source of competition, has been superseded almost completely by fluorspar, which is cheaper and better for all purposes except the production of sodium salts; the present importations of cryolite are used entirely for the latter purpose. (Refer to article on cryolite, elsewhere in this volume).

Uses.—About 20 per cent. of the crude fluorspar is marketed in lump form, the remainder as gravel. Of the total fluorspar consumption, fully 80 per cent. is used in iron and steel making, chiefly in open hearth charges, and 5 per cent. by other smelters. A small percentage of the lump and gravel (the crushed is included with the latter) is purchased by manufacturing chemists, enamel-ware makers and founders. The chemical (largely hydrofluoric acid) and glass industries consume 57 and 21 per cent. of the ground product respectively. The consumption for glass purposes shows an increase over previous years. Of the remaining consumption of ground fluorspar, 13 per cent. goes to the manufacture of enamels, glazes and fire-proof ware, and about 5 per cent. to founders and 4 per cent. for miscellaneous purposes. The enamel-makers are using less in the ground form and more lump than formerly. About 135 companies use fluorspar in greater or less quantity for the several uses; in addition, there are several hundred iron, steel and brass founders using it in small quantity. One company sells a patent flux consisting largely of fluorspar and red clay.

Fluorspar Mining in the United States.

Kentucky.—In the western Kentucky district, the Kentucky Fluorspar Company, with mines in Crittenden county, has to its credit the largest amount of development work. In a new shaft at the Brown mines a vein, more than 4 ft. thick, of fluorspar and zinc carbonate was secured. At the Hodge mines, exploitation on all three veins was in progress; a large central power plant is shortly to be erected. The Meeks incline, on the most southerly vein, has 6 ft. of white fluorspar and galena of the beautiful ribbon structure type. On the main Hodge, work was done in the Air shaft, zinc carbonate was mined from surface cuts and a considerable body of gravel fluorspar was mined at the Reed cut. On the Jones vein at the Holly mines, 4 ft. of fluorspar was cut in the main shaft. Gravel was mined on contract on the Reed and Klondike veins. A new vein of purple fluorspar was discovered 50 yd. southwest of the Klondike vein. The Memphis incline showed an ore shoot 14 ft. wide, largely No. 1 fluorspar. A well-designed plant has been installed and with aid of hammer-drills, the fluorspar is being mined at one-half its former cost. The present output will be increased upon the opening of new levels, sinking for which is now in progress. From maps and sections prepared by C. S. Nunn, president of the company, it is evident the Memphis ore shoots pitch to the right, looking down the dip, agreeing with Clayton's law. Slickenside grooves in the ore tally with the pitch of the ore shoots and serve as indicators; primary slickensiding, such as is seen on the walls, does not serve this purpose. At John and Nancy Hanks mines the ore shoots also pitch to the right, while at the Nine Acres, they pitch to the left. Immense gravel shoots on either side of the Kentucky mine were stoped; to the

south, the Milliken shaft was sunk almost entirely in gravel, getting white lump on one side. In sinking on the Yandell incline, excellent white fluorspar was encountered.

The main fault was cut at Glendale, showing at this and the Leona mine from 10 to 16 ft. of galena-charged fluorspar. New openings were made by the Blue Grass Fluorspar Company, on the Tabor and Asbridge properties, and small power plants were installed. The vein in these at 80 ft. depth varies from 4 to 12 ft. of gravel and lump mineral, showing locally a good deal of galena. At the Wheatcroft, a secondary vein of purple fluorspar was cut.

On the Columbia lode, a 3-ft. vein of zinc carbonate was secured on the Wilson land. At the Ada Florence, at 132 ft. a good vein of fluorspar-galena was found near the shaft, which changed to zinc sulphide and galena with only 2 per cent. fluorspar, a short distance southwest; a compressor, etc., were installed here. Arrangements are being consummated for re-opening the Columbia mine under new management. Developments were in progress on the Memphis lode; also at the Edwards, Thurman and Eclipse. The last has 2.5 ft. of blende and is equipped with a small air plant. Fluorspar 2 to 4 ft. wide, in contact with a mica-peridotite dike, was cut at the Will La Rue prospect. Veins of fluorspar showing well in lead were reported opened at the J. O. Belt and Ed. Flanary properties. At the Riley, several levels were operated and sinking continued below 200 ft. New shafts are being sunk at the Parish and Leander. At the Silver Star, a 3.5 ft. fluorspar vein was cut. A new vein of zinc carbonate was discovered at the Old Jim Sinkholes, while both carbonate and sulphide were secured at the Lady Farmer. Carbonate was mined at the Old Dad, and considerable 8 per cent. zinc dirt secured at the Red Hill.

In Livingston county, at the Evening Star mine, a body of ore 12 ft. wide was cut at 120 ft. depth, carrying 20 per cent. blende, and 5 per cent. galena, with the remainder chiefly fluorspar. The mine is now capable of an output of 50 tons per day. At the Nancy Hanks, to the south of the same lode, a large output of fluorspar and galena was had from the 168 and 180 ft. levels. A lump fluorspar shoot, 18 ft. wide was mined at the John mine, and considering that the mine and plant had to be entirely rebuilt, having burned, the output was unusually large.

No work was done in Lyon, Caldwell or Trigg counties, though operations in the last two will be resumed in 1907. Galena, blende and fluorspar were discovered for the first time in Christian county, at the McReynolds' prospect, and Eastern capitalists are preparing for considerable work in 1907. Further work was also done at the Freeman prospect.

The State Survey party headed by F. J. Fohs completed geologic mapping and investigations of the lead, zinc and spar deposits of the six counties of the district. Some important conclusions were reached for which

the reader is referred to the forthcoming report. Topographic mapping was done jointly with the Federal Survey in Crittenden and Caldwell counties. A resurvey of a proposed railroad was made in the western part of the field.

Milling.—The Kentucky mill continues a large producer of ground fluorspar; the only change made was the installation of an 80 h.p. Corliss engine to secure more adequate power. The Riley mill was operated but little. The Nancy Hanks became a regular producer and a large tube mill was added to increase grinding capacity; the newly patented Ratcliffe-Cohenour shaking screen is in successful operation here. The Old Dad zinc mill cleaned some carbonate ore.

In Caldwell county, the Union Central Mining Company erected a mill of the Joplin type, at the Stone mine, for the separation of fluorspar and galena, but it will not be operated until the spring of 1907. The Eagle Fluorspar Company continued experimental work on zinc-fluorspar ores at the Evening Star mill throughout 1906.

A successful separation of fluorspar and zinc sulphides is at last being accomplished, and by at least two distinct types of separators. One of these, a non-acid flotation process, is already in operation in the district, while tests with electrical separators are reported satisfactory. These processes are referred to elsewhere in this volume (see article on zinc).

In the central Kentucky district, the Monitor Mineral Company was able to do but little at its fluorspar properties, the Twin Chimney and the Fantail mines, owing to excessive water and lack of competent labor. These difficulties are to be overcome, and work will shortly be resumed. The Twin Chimney has reached a depth of 117 ft. below the tunnel. At the Million Calc mine, a tunnel is now being driven, and at 80 ft. shows 5 ft. of calcspar.

Southern Illinois District.—The Rosiclaire mine, which is 325 ft. deep, has been worked for 900 ft. on the vein at the 300-ft. level, the vein varyng from 3 to 28 ft. in width, of fluorspar and galena. A secondary vein striking W. 17 deg. E., intersecting the main vein not far from the shaft, was also worked. The Fairview incline, more than 330 ft. deep, was worked at four levels; for about 625 ft. it has been completely stoped out to the 300-ft. level; in the latter, the vein was from 6 to 14 ft. wide.

On Lead Hill, three mines, the Evansville, Robinson and Cave In Rock, worked two veins, bedded by replacement, 1 to 12 ft. thick (fluorspar, galena and zinc carbonate). Both beds were worked by entries at the Robinson, while at the Evansville only the lower one, where the entries were 150 ft. into the hill. At the Cave In Rock, at 18 ft. depth, they have gone across the lower bed for 240 ft., where it was cut off by a fault; some work was also done on the upper one. The same methods pursued

in relocating faulted coal beds will be of service where these beds are cut off.

An overthrust fault, pitching 35 deg., having from 3 to 8 ft. of fluorspar, galena and quartzite was stoped and further efforts made to intersect the master fault at the Mary mine. At the Rose, green fluorspar was mined, machinery having been installed, and a shaft was sunk 115 ft., encountering quartzite and fluorspar. The Pell was reopened, but not worked to any extent. A power plant was installed at the Hamp fluorspar mine, and both here and at the Mt. Carmel lead mine, development work was in progress. At the Hubbard mine, a broken down bedded vein of crystallized fluorspar, covering a large tract at grass roots, was operated. The Empire vein with gravel fluorspar and limestone, and zinc carbonate with a small percentage of fluorspar, in rather distant bands, was mined in a pit 50 ft. in diameter, and 45 ft. in depth. The main Empire shaft is reported to have 4 ft. of solid white fluorspar of the highest grade, with limestone walls.

Colorado.—Fluorspar was mined and shipped for the first time in Colorado, at Jamestown, Boulder county, in 1905. It was obtained from strong veins in Algonkian crystallines associated with andesite intrusions of later date; open cut methods were employed.[1] No shipments were reported for 1906 from the Boulder district; one of the operators reported his deposit, from which shipments were made in 1905, to have been exhausted. In Custer county, likewise in a region of Algonkian rocks and recent igneous intrusives, fluorite occurs locally in quantity in veins of the composite fault-fissure breccia-replacement type. It also occurs in other parts of the district more rarely and associated with gold tellurides. Near Silver Cliff, a fissure showing an average of 4 ft. of solid fluorspar is being developed by R. W. Blackett, by means of a tunnel. Vein continuity has been proved for 500 ft., and some shipments were made in 1906.

Tennessee.—The Rome incline at 120 ft., presents beautiful white crystallized fluorspar. At the Alcorn open-cut, on the line of the Southern Railway, the fluorspar increases in quantity among the limestone crevices, and shaft sinking is contemplated.[2] Here amber-colored crystallized fluorspar is obtained. No lead or zinc occurs at either property.

A 2.5 ft. vein of zinc and barite is being prospected in Smith county, by W. E. Minner. At the National Cemetery near Nashville, J. E. Wright reports that he opened a vein of fluorspar and barite, 1 to 2.5 ft. wide.

Bibliography.

The following references may be made to articles which deserve special mention among those contributions to the subject of fluorspar, which

[1] E. O. Hovey, "Fluorspar and Cryolite." "Mineral Resources of the United States," 1906.
[2] Private communication from D. C. Roberts.

have appeared in previous volumes of The Mineral Industry. In Vol. II, the article on Fluorspar, by P. L. Fearn, and "The Fluorspar Deposits of Kentucky," by Prof. Charles J. Norwood, are useful references. In Vol. VII, there was an article dealing with the consumption of fluorspar in the manufacture of hydrofluoric acid and a discussion of the advantages of fluorspar as a flux in smelting. In Vol. VIII, a description of methods of fluorspar mining in the United States, as carried out in the principal districts producing this mineral. In Vol. X, a complete account of the methods used in manufacturing commercial hydrofluoric acid. In Vol. XI, a brief description of the use of sodium fluoride for the purification of water. In Vol. XII, the uses and technology of fluorspar were discussed. In Vol. XIV, a full account of fluorspar mining in the United States, describing the principal producing districts—also a brief description of the plants for concentrating fluorspar and their equipment in 1905.

The Manufacture of Hydrofluoric Acid.

By Anson G. Betts.

No very useful literature on this subject exists to the best of my knowledge. Most chemists regard it as an extremely dangerous substance, and have presumably left it alone as much as possible. Yet hydrofluoric acid and fluorides have an extending use, for numerous purposes. Its preparation is easy, and safe if proper precautions are taken.

Experimental Work.—Samples of fluorspar may be tested by mixing

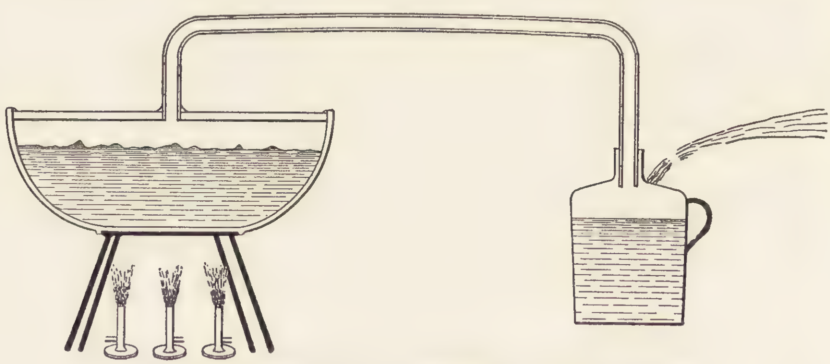


FIG. 1.—LABORATORY STILL FOR MAKING HYDROFLUORIC ACID.

say 50 grams with various proportions of 66 deg. sulphuric acid in small sheet-iron pans, and distilling under the hood. For preparation in small quantities for the laboratory, the apparatus shown in Fig. 1 gives good results if used out of doors. The retort is an ordinary cast-iron pot, perhaps 1 ft. in diameter and 6 in. deep. The cover is made by filling with sand to near the top, leveling it off and pouring in about ½ in. of lead. The

lead pipe is separate from the cover and passes over to a lead hydrofluoric-acid bottle containing water. The water must not come as high as the end of the lead pipe.

During distillation, the bottle is sprayed with water from a hose to keep it cool. A charge of about 2 kg. of fluorspar and 2.5 kg. H_2SO_4, (66 deg.) is stirred up in the pot. The fluorspar, for the most part, dissolves immediately on stirring in the sulphuric acid, without evolution of much fume, until heat is applied. The cover is put on and dry cement put over the joints as a lute, cement being suitable for this purpose.

The heating should be moderate at first to prevent too much frothing in the pot. Distillation takes 2 or 3 hours, and the end can be told by feeling of the lead pipe near the bottle, which is hot as long as acid is coming over. Very little loss is experienced and a yield of 80 per cent. or thereabout, is obtained.

Operation On Larger Scale.—On a large scale, the application of the same principles is successful. The general arrangement is shown in Fig. 2, for which a few explanations are necessary. The pot may be cast about

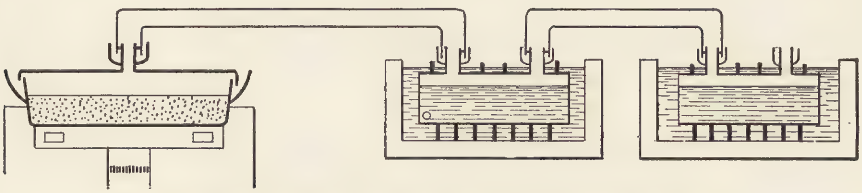


FIG. 2.—PLANT FOR THE MANUFACTURE OF HYDROFLUORIC ACID.

8 ft. in diameter, 3 ft. deep at the center, and 1 in. thick, with a slightly curving bottom to prevent cracking. For the pot a cast-iron cover 1 in. thick is used, dipping into the annular trough around the pot, which contains strong sulphuric acid as a seal. All the other seals are made in the same way, but water may be used for the joints on the condensers where the temperature is not so high. Lead retorts, and lead covers for the retorts are useless.

The condensers consist of a series of two or three lead boxes of about 1 cu.m. capacity, entirely submerged in a water tank, and partially filled with water or dilute HF. Condensers should be made of heavy lead, supported by wooden pieces to which the lead is attached by means of lead straps burned on. The lead delivery pipes may be about 5 in. in diameter. The condensers have an overflow so that the acid never can rise to the end of the delivery pipe. If this happens, a partial vacuum may result, and draw water back into the pot, where it would probably cause an explosion.

The charge may consist of 1000 lb. of ground fluorspar and 1000 to

1200 lb. of 66 deg. sulphuric acid. SiF_4 comes off first and deposits silica on the water in the first condenser, stopping absorption somewhat so that it is necessary to stir the water in the first condenser until most of the SiF_4 has come over. The pot may be charged in the morning and distillation finished by night. Coal is used for fuel, burned on a grate of about 3 sq.ft. The residue in the pot is comparatively hard, and, after cooling, is dug out with pick and shovel. The yield of acid calculated on the sulphuric acid used is approximately 80 to 90 per cent.

The cost of manufacture is not great, the principal items being the raw materials necessary. To produce 1 lb. anhydrous HF, about $2\frac{1}{2}$ lb. of fluorspar and 3lb. of sulphuric acid are necessary. Fluorspar and sulphuric acid are worth about \$10 to \$15 a ton, making a cost for raw materials, exclusive of coal, of approximately $2\frac{3}{4}$ to $4\frac{1}{8}$ cents per pound of anhydrous HF.

Method of Analysis.—The sample of acid is mixed with several times its bulk of nearly saturated and neutral potassium nitrate solution. This causes a precipitation of potassium fluosilicate in the solution. Phenolphthalein is used as indicator and the solution titrated with caustic soda in the cold. This gives the total of the HF and H_2SiF_6 present. The sample is then heated to boiling when it will be found that considerable more caustic soda may be run in to get another end point. In the first titration, the HF present and the HNO_3 liberated by the reaction of potassium nitrate and fluosilicic acid, are neutralized by the alkali. When titrated hot, the precipitated K_2SiF_6 is decomposed by the alkali. The following is the equation involved:

$$K_2SiF_6 + 4NaOH = 2KF + 4NaF + SiO_2 + 2H_2O.$$

The rule for calculating is, 1 gram NaOH used in the second titration =0.9 gram H_2SiF_6 in the sample. For HF present divide the number of cubic centimeters of NaOH used in the second titration by 2, and subtract the result from the c.c. used in the first titration. The remainder shows the HF, 1 gram of NaOH equaling 0.5 gram HF.

Shipment and Storage.—Hydrofluoric acid has been shipped in beer barrels with rosin lining, which are entirely successful and last for some time and for long shipments; also in rectangular lead carboys. Its storage in lead is not very satisfactory on account of the corrosion of the lead. Probably the presence of sulphuric and fluosilicic acids has some effect in the corrosion.

FULLER'S EARTH.

By Reginald Meeks.

There were several new discoveries of fuller's earth in 1906 and several new companies started to produce. There is still a prejudice in favor of the English product, but the domestic article should begin to assume its rightful position in the home markets. The production and imports are shown in the subjoined table:

STATISTICS OF FULLER'S EARTH IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.		Imports.		Year.	Production.		Imports.	
	Sh. Tons.	Value.	Sh. Tons.	Value.		Sh. Tons.	Value.	Sh. Tons.	Value.
1897......	17,195	$92,398	(a)	....	1902......	14,100	$109,980	15,135	$102,580
1898......	15,553	87,365	9,355	$81,044	1903......	20,693	190,277	17,100	120,671
1899......	13,620	81,900	11,558	69,460	1904......	29,480	168,500	10,221	74,000
1900......	11,813	70,565	9,154	64,790	1905......	25,745	157,776	15,181	105,997
1901......	14,112	96,835	12,061	80,697	1906......	28,000	237,950	14,827	108,696

(a) Not reported.

Markets and Prices.—The market was steady throughout the year and prices fluctuated within narrow limits. Ground fuller's earth was delivered on contracts at $14 to $15 per short ton, while single car loads brought from $16 to $17 according to quality and terms of sale.

Review by States

Alabama.—A deposit of fuller's earth, extending over 200 acres, occurs on Blakely island, near Mobile, which is reported to be of good grade and plentiful.[1] The discovery dates back to the Civil War, but little has yet been done to develop the deposits. A concern known as the Southern Commercial Company attempted to operate the deposits but did not succeed and its mill was burned a few years ago. Recently the property was acquired by the Standard Reduction Company of Chicago, which, it is reported, intends to erect a modern plant on the property.

California.—The California Fuller's Earth Company reports that the market for its product on the Pacific Coast is limited and that the freight rate to the East prohibits entrance into that market in competition with the English product which is carried as ballast and incurs a custom tax of $1.50 for the crude and $3 for the prepared product. The company has

[1] *Paint, Oil and Drug Review.* March 6, 1907.

operated by open pits, which is the usual form of development. There were no new producers in California during 1906, and the State produced only a small part of the total output of this country.

Florida.—The Owl Commercial Company of Quincy, Fla., operated continuously during 1906 and marketed its product entirely in the finished state. This concern is the most important producer in Florida. There are no developments to report, but it is understood that other operations are expected to start during 1907. The Southern Fuller's Earth Company whose plant is situated at Mt. Pleasant, Gadson County, Fla., operated during the entire year. No report of its production was received, and this is estimated at 5000 tons in arriving at the total given at the beginning of this article.

South Carolina.—As found in this State, according to Earle C. Sloan, State Geologist, fuller's earths are clays belonging to the Eocene and Neocene formations, and are of approximately the same composition as the English except in the water content, which is much lower. The color is white, gray, yellow, pink or black. The earth varies in specific gravity from 1.75 to 2, the lower limit being determined by included air spaces. The best varieties crumble in water, accompanied by a faint sizzing sound. The Eocene fuller's earth, when dry, adheres to the tongue. The earth of this formation extends in a broken sinuous band across almost the entire State, with its upper margin varying from 10 to 20 miles southerly from the "fall line." In some localities it approximates 40 ft. in thickness. The presence of marine shells is a bad augury for its serviceability, for implied salt water antecedents of the greater part of the bed of Eocene fuller's earth is frequently strengthened by the presence of sulphates in the form of alum, probably derived from the alteration of pyrite, which was previously formed by the action of organic matter on the salts of brackish water, in the presence of iron compounds. The presence of alum is very prejudicial to the value of fuller's earth employed for the treatment of the culinary or table oils and fats. The Eocene fuller's earth of the extreme upper portion of the coastal region, is generally free from sulphates in objectionable forms and quantities.

The beds of Neocene fuller's earth are detached patches dotted over the median and submedian portions of the coastal plain. As compared with the Eocene fuller's earth they afford superior satisfaction.

South Dakota.—For many years the deposits of fuller's earth near Custer, Custer county, have been known but it is only recently that any attention has been paid to them. The deposits are found in what is known as the Chadron formation and are reported to be the most extensive and important so far discovered in the West. They were first discovered at Fairburn and later at Argyle, Minnekahta and at other places.[1]

[1] *Mining Reporter.* June 7, 1906.

The Argyle Fuller's Earth Company, Custer, So. Dak., owns the bulk of the properties. It made no shipments, except samples, during 1906. The earth is of the following composition: Silica, 55.45; alumina, 18.58; water, 8.8; volatile matter, 5.35; ferric oxide, 3.82; lime, 3.4; magnesia, 3.5; manganese, trace. It is claimed that this analysis closely resembles that of many English earths.

Texas.—Important discoveries have been made near Somerville, in Burleson county, and two companies have organized to develop the deposits. These are the Texas Fuller's Earth Company and the Somerville Development Company. The former has ordered a plant which will cost about $35,000 and will have a location on the main line of the Gulf, Colorado & Santa Fé Railway about 75 miles north of Houston. The Somerville Development Company leased from N. W. Dunham the fuller's earth rights, water privileges and also all the fuel on the land for a period of 99 years. It is erecting a plant which will have a capacity when completed of 200 tons per day. These companies have, as yet, made no shipments.

Technology of Fuller's Earth.

Preparation.—Fuller's earth is prepared by first partly drying it in the air and then burning it at a low red heat on open iron plates which form the top of a long furnace. It is raked from the end nearest to the stack to the opposite end where it is cooled, and then is ground by buhr-stone mills to pass a 90-mesh screen. Ball mills and rolls are also employed for grinding.

Uses and Requirements.—[1]The chief use for fuller's earth is in clarifying oils and fats. A little is used by drug manufacturers, but this amount is insignificant. In refining cottonseed oil, pulverized earth, in proportions varying (according to the grade of the earth) from 5 to 7 per cent., is added to the crude oil, in a large vertical iron cylinder equipped with a peripheral steam coil and a central revolving agitator, which are employed respectively to heat the oil to 180 deg. F. and to stir the mass so as to bring particles of the earth in contact with all particles of suspended coloring matter. The mixture is then pumped under pressure into the filter press, causing at the same time the suspended fuller's earth to deposit on the inner surface of the canvas filter, and the oil to filter through this coating of the earth and the canvas; the aluminous earth mechanically entangles all suspended matter. The filtrate should be clear, colorless and without objectionable taste or odor. While an increase of the alumina content of fuller's earth enhances the bleaching effect it decreases the filtering quality. Therefore the essential attributes of a fuller's earth acceptable to the trade are:

[1] Abstract from Report of the State Geologist of South Carolina, 1906.

1. It must be finely subdivided, and free from such abrasive matter as might increase the wear of the pump cylinders, pistons and valves.

2. It must contain as much bleaching substance as is consistent with rapid filtration under a pressure of 60 ℔. for the reason that its value is proportional to the amount of bleaching power exercised consistently with the other requirements.

3. It is observed that where the alumina exceed one-fifth of the amount of the silica present, the critical point is approximated, beyond which an increase in the densely bedding aluminous matter prejudices filtration. The silica therefore serves to maintain the required porosity.

4. In the bleaching of cottonseed oils, lards and other organic oils or fats required for culinary purposes, the presence of matter affording foreign flavors or odors constitutes a fatal objection. In the treatment of mineral oils and of those organic oils used for illumination, fuel, lubrication, etc. the same fastidiousness is not essential.

GARNET.

By D. H. Newland.

The output of garnet for abrasive uses in the Adirondack region of New York, amounted in 1906 to 4729 short tons valued at $159,298. This output, it may be noted, is the largest ever made by this district. As the demand has been sufficient to absorb the supply, it would appear that the mineral is gaining favor with the abrasive trade. The large increase has been made possible by the recent developments undertaken by the North River Garnet Company, near Thirteenth Lake, Warren county. The company maintains its quarries in continuous operation, while the other companies cannot work to advantage during the winter months. The recently discovered deposit of garnet near Keeseville, Essex county, made no output except a small quantity for testing purposes.

PRODUCTION OF GARNET IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Short Tons.	Value.	Value per Ton.	Year.	Short Tons.	Value.	Value per Ton.
1897	2,261	$66,353	$29.34	1902	3,722	$122,826	$33.00
1898	2,882	82,930	28.74	1903	4,413	146,955	33.30
1899	2,565	72,672	28.33	1904	2,952	89,636	30.36
1900	3,285	92,801	28.25	1905	3,694	114,625	31.01
1901	4,444	158,100	35.51	1906	5,404	179,548	33.22

GLASS.

By A. Van Zwaluwenburg.

The manufacture of glass is classed among the metallurgical industries, its purpose being, however, not the separation of metallic products, but the production of a slag which is put to various uses in the arts. The importance of the industry as a consumer of mineral products is shown by the statistics of the materials produced for use in the manufacture of glass. According to the "Mineral Resources of the United States," 1905, the production of glass sand in the country during that year was 1,030,334 short tons, valued at $1,083,730; the production in 1904 was 858,719 short tons, valued at $796,492; consequently, 1905 showed an increase in quantity of 171,615 short tons and in value of $287,238.

The production of glass in the United States, according to the Census statistics, increased from $56,539,712 in 1899 to $79,607,998, in 1904, an advance of 40.8 per cent. To the total for the latter year should be added $9,663, the value of glassware produced by establishments engaged primarily in the manufacture of other products. The manufacture of glass was reported in 1904 by 399 establishments, the total capital of which was $89,389,151. They employed 63,969 wage-earners, paid $37,288,148 in wages and consumed materials to the value of $26,145,522, practically all of which consisted of mineral products. As compared with the totals for the industry in 1899, these figures represent an increase of 12.4 per cent. in the number of establishments, 47.2 per cent. in capital, 21.1 per cent. in wage-earners, 37.7 per cent. in wages, and 56.3 per cent. in the cost of materials.

The total value of the products for the year 1904 is distributed as follows:—Window glass, $11,610,851; polished plate glass, $7,978,253; tableware, $4,897,537; jellies, tumblers and goblets, $1,639,167; lamps, $1,247,628; chimneys, $3,061,334; globes, shades and other electrical and gas goods, $3,055,386; blown tumblers, stemware and bar goods, $2,928,198; cut glass, $987,556; prescription vials and druggists' wares, $5,638,508; beer, soda and mineral glasses, $7,927,287; liquors and flasks, $5,555,815; milk jars, $1,160,743; fruit jars, $3,436,047; patent and proprietary bottles, $3,709,510; packers and preserves bottles and jars, $2,989,557; all other products, $10,784,621.

The value of the materials consumed in the manufacture of glass during the same year is distributed as follows:—Glass sand (769,792 tons),

$1,547,147; soda ash (215,462 tons), $4,068,204; other materials $13,-488,363; natural gas, $2,777,157; oil, $526,868; coal, $2,748,766.

Quartz Glass.—At the high temperatures now available pure silica may be fused, and while in a plastic state it may be worked like ordinary glass. Quartz glass does not soften until high temperatures are reached. It may be cooled suddenly without cracking and possesses other properties which render it valuable for scientific and other purposes. Until recently, only a few specimens of tubes and small apparatus had been made by piecing together single crystals of clear quartz by means of the oxyhydrogen blow pipe. When a quantity of quartz is fused at atmospheric pressure the mass is rendered opaque upon cooling by recrystallization or by included air bubbles. In experiments recently undertaken by the Carnegie Institution quartz was fused and cooled under pressure, the result being a clear transparent glass.

GOLD AND SILVER.

By Frederick Hobart.

The year 1906 recorded a further increase in the great production of gold which has marked the last decade. The total exceeded $400,000,000, the largest amount on record. The Transvaal again led and showed the largest gain. The United States held the second place, with Australia third. These three producers together furnished about three-fourths of the total. For Russia, the fourth producer in rank, it has been impossible as yet to obtain final statistics owing to the troubles of the past year.

PRODUCTION OF GOLD IN THE UNITED STATES. (*a*)

States.	1903		1904		1905		1906	
	Fine Ounces.	Value.	Fine Ounces.	Value.	Fine Ounces.	Value.	Fine Ounces.	Value.
Alabama	213	$4,400	1,417	$29,300	2,008	$41,500	1,747	$35,900
Alaska	416,738	8,614,700	450,091	9,304,200	722,090	14,925,600	1,028,113	21,251,100
Arizona	210,799	4,357,600	161,761	3,343,900	130,203	2,691,300	155,965	3,223,800
California	779,057	16,104,500	924,427	19,109,600	928,742	19,197,100	901,495	18,633,900
Colorado	1,070,376	22,540,100	1,180,147	24,395,800	1,243,401	25,701,100	1,101,655	22,771,200
Georgia	3,000	62,000	4,688	96,900	4,586	94,800	3,106	64,200
Idaho	75,969	1,570,400	72,742	1,503,700	52,085	1,075,600	52,912	1,093,700
Maryland	24	500	116	2,400	818	16,900	....	(*c*)
Montana	213,425	4,411,900	246,606	5,097,800	236,541	4,889,300	221,858	4,585,800
Nevada	163,892	3,388,000	208,390	4,307,800	259,269	5,359,100	474,881	9,815,800
New Mexico	11,833	244,600	18,475	381,900	12,859	265,800	12,381	255,900
N. Carolina	3,411	70,500	5,994	123,900	5,990	123,900	5,931	122,600
Oregon	62,411	1,290,200	63,336	1,309,900	60,227	1,244,900	66,278	1,369,900
S. Carolina	4,872	100,700	5,892	121,800	4,601	95,100	3,609	74,600
S. Dakota	300,243	6,826,700	339,815	7,024,600	334,490	6,913,900	330,078	6,822,700
Tennessee	38	800	208	4,300	160	3,300	....	(*c*)
Texas	....	....	110	2,300	92	1,900	....	(*c*)
Utah	178,863	3,697,400	203,902	4,215,000	248,713	5,140,900	250,227	5,172,200
Virginia	654	13,500	184	3,800	242	5,000	....	(*c*)
Washington	13,589	279,900	15,862	327,900	17,900	370,000	17,155	352,600
Wyoming	175	3,600	793	16,400	1,147	23,700	1,277	26,400
Other States	468	9,700	....	....	....	....	20,770	429,100
Total	3,560,000	$73,591,700	3,904,986	$80,723,200	4,266,120	$88,180,700	4,648,385	$96 101,400

(*a*) The statistics in this table are as reported by the Director of the Mint those for 1906 being the preliminary figures (subject to revision), except that later official figures for 1906 have been used, when available. (*c*) Included in other States.

From the best available information, however, there was a small decrease; it was small, because most of the Siberian gold mines are isolated from the centers of commercial and administrative disturbance. Canada dropped to the sixth place, falling below Mexico; the decrease resulted chiefly from a lower output in the Yukon territory, which is still in the transition state. British India showed a loss in the Kolar goldfield, the largest mine in that field—the Champion Reef—having entered a zone of

lean ore, through which work is being pushed in the hope of recovering the main orebody at a lower depth.

The silver production of the world showed little change in 1906. The output increased with that of gold, lead and copper, in connection with

TOTAL PRODUCTION OF GOLD AND SILVER IN THE UNITED STATES.

Years.	Gold.	Silver.	Yrs.	Gold.	Silver.	Years.	Gold.	Silver.
	Dollars.	*Ounces.*		*Dollars.*	*Ounces.*		*Dollars.*	*Ounces.*
1792—1834	14,000,000	*Nil*	1883	35,000,000	35,730,000	1896	53,088,000	58,835,000
1835—1844	7,500,000	193,365	1884	30,800,000	37,800,000	1897	57,363,000	53,860,000
1845—1854	343,036,769	386,730	1885	31,800,000	39,910,000	1898	64.463,000	54,438,000
1855—1864	479,300,000	20,806,518	1886	35,000,000	39,685,513	1899	71,053,000	54,764,000
1865—1874	454,950,000	154,390,609	1887	33,000,000	41,721,592	1900	79,171,000	57,647,000
1875	33,400,000	24,533,993	1888	33,175,000	45,792,682	1901	78,666,700	55,214,000
1876	39,900,000	30,010,054	1889	32,800,000	50,000,773	1902	80,000,000	55,500,000
1877	46,900,000	30,783,509	1890	32,845,000	54,516,300	1903	73,591,700	54,300,000
1878	51,200,000	34,960,000	1891	33,175,000	58,330,000	1904	80,723,200	57,786,100
1879	38,900,000	31,550,000	1892	33,000,000	64,900,000	1905	88,180,700	56,101,600
1880	36,000,000	30,320,000	1893	35,955,000	60,000,000	1906	96,101,400	56,183,500
1881	34,700,000	33,260,000	1894	39,500,000	49,500,000			
1882	32,500,000	36,200,000	1895	46,610,000	55,727,000	Total	2,882,348,469	1,673,483,422

Note.—To the end of 1872, the statistics are those of R. W. Raymond, United States Mining Commissioner; subsequent statistics are those reported by the Director of the Mint.

PRODUCTION OF SILVER IN THE UNITED STATES. (*a*)

	1903.		1904.		1905.		1906.	
	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.
Alabama	....	....	200	$116	300	$181	....	....
Alaska	143,600	$77,544	210,800	122,264	169,200	102,116	191,700	$128,038
Arizona	3,387,100	1,829,034	2,744,100	1,571,578	2,005,700	1,672,592	2,747,800	1,835,283
California	931,500	503,010	1.532,500	144,313	1,082,000	653,009	1,564,500	1,044,946
Colorado	12,990,200	7,014,708	14,331,600	8,312,328	12,942'800	7,811,239	12,248,100	8,180,628
Georgia	400	216	1,500	870	900	543	1 000	668
Idaho	6,507,400	3,513,996	7,810,200	4,529,916	8,125,600	4,903,962	8,257,300	5,515,132
Michigan	50,000	27,000	127,800	74,124	253,000	152,690	286,800	191,557
Montana	12,642,300	6,826,842	14,608,100	8,472,698	13,454,700	8,120,181	11,478,700	7,666,738
Nevada	5,050,500	2,727,270	2,695,100	1,563,158	5,863,500	3,538,740	6,742,900	4,503,650
N. Mexico	180,700	97,578	214,600	124,468	354,900	214,189	356,200	237,909
N. Carolina	11,000	5,940	14,800	8,584	13,200	7,966	10,900	7,281
Oregon	118,000	63,720	133,200	77,256	88,900	53,653	100,100	66,858
S. Carolina	300	162	500	290	200	121	....	....
S. Dakota	221,200	119,448	187,000	108,460	179,000	108,030	157,500	105,196
Tennessee	13,000	7,020	59,200	34,336	95,400	57,576	(*d*)	....
Texas	469,600	245,376	469,600	272,368	417,200	251,788	(*d*)	....
Utah	11,196,800	6,046,272	12,484,300	7,240,894	10,319,800	6,228,205	11,538,000	7,706,346
Virginia	9,500	5,130	6,700	3,886	200	121	(*d*)	....
Washington	294,500	159,030	149,900	86,942	119,400	72,060	140,500	93,841
Wyoming	200	108	4,400	2,552	2,700	1,630	1,300	869
Other States	97,400	52,596	....	....	(*c*)13,000	7,846	360,200	240,581
Total	54,300,000	$29,332,000	57,786,100	$33,515,938	56,101,600	$33,858,438	56,183,500	$37,525,521

(*a*) The statistics in this table are reported by the Director of the Mint, those for 1906 being the preliminary figures (subject to revision); except that later official figures for 1906 have been used, when available. (c) Includes Maryland, 100 oz.; Missouri, 2,900 oz. (*d*) Included in other States.

which most of the mining of silver is done. The United States and Mexico remained the chief producers.

The large production of gold recorded in the tables given on the following pages was at once incident to, and an important factor in, the extra-

ordinary commercial and economical activity of the year. An unusually large proportion of the gold mined undoubtedly entered into and increased the world's supply of capital and circulating money; and this condition

GOLD PRODUCTION OF THE WORLD.

Countries.	1904			1905			1906		
	Oz. Fine.	Kilo-grams.	Value.	Oz. Fine.	Kilo-grams.	Value.	Oz: Fine.	Kilo-grams.	Value.
America, North:									
United States..	3,904,986	121,445.1	$80,723,200	4,265,742	132,682.0	$88,180,700	4,648,375	144,584	$96,101,400
Canada........	793,420	24,675.4	16,400,000	700,863	21,800.0	14,486,833	581,709	18,093	12,023,932
Newfoundland .	6,242	194.0	129,022	4,550	141.5	94,049	4,475	139	92,500
Mexico (*a*).....	620,209	19,291.1	12,819,720	779,181	24,236.0	16,107,100	805,000	25,039	16,639,350
Central America	54,218	1,686.3	1,120,700	73,212	2,277.0	1,513,400	(*e*) 58,321	1,814	1,205,500
America, South:									
Argentina......	446	13.9	9,200	(*e*) 446	13.9	9,200	(*e*) 731	23	15,100
Bolivia........	147	4.5	3,000	(*e*) 147	4.5	3,000	(*e*) 1,064	33	22,000
Brazil.........	98,354	3,058.8	2,032,984	(*a*)117,396	3,651.5	2,426,575	(*e*)120,948	3,738	2,500,000
Chile..........	30,812	958.4	636,900	(*e*) 30,812	958.4	636,900	(*e*) 30,963	966	640,000
Colombia......	95,520	2,970.8	1,974,400	(*e*) 95,513	2,970.8	1,974,400	(*e*) 95,791	2,980	1,980,000
Ecuador.......	6,430	200.0	132,900	(*e*) 6,430	200.0	132,900	(*e*) 6,430	200	132,900
Guiana (British)	70,661	2,198.0	1,460,580	82,300	2,559.9	1,701,141	79,682	2,478	1,647,031
Guiana (Dutch)	25,778	801.8	531,831	34,442	1,071.3	711,916	38,162	1,187	788,820
Guiana (French)	86,532	2,691.5	1,788,800	(*e*) 86,532	2,691.5	1,788,800	78,028	2,427	1,612,863
Peru..........	17,406	541.4	359,782	(*e*) 17,406	541.4	359,782	(*e*) 17,900	557	370,000
Uruguay.......	1,227	40.0	25,368	(*e*) 1,227	40.0	25,368	(*e*) 1,210	38	25,000
Venezuela......	(*e*) 14,514	451.4	300,000	(*e*) 14,512	451.4	300,000	(*e*) 14,998	466	310,00
Europe:									
Austria........	2,283	71.0	47,190	(*e*) 2,283	71.0	47,190	(*e*) 2,225	69	46,000
Hungary.......	117,949	3,668.7	2,438,006	(*e*)117,949	3,668.7	2,438,006	(*e*)117,078	3,828	2,420,000
Germany (*c*)...	88,029	2,738.0	1,819,518	126,446	3,933.0	2,613,639	135,094	4,202	2,792,439
Italy..........	325	10.1	6,718	(*e*) 325	10.1	6,718	(*e*) 319	10	6,600
Norway.......	350	10.9	7,234	(*e*) 350	10.9	7,234	387	12	8,000
Portugal.......	40	1.3	827	(*e*) 40	1.3	827	(*e*) 48	1	1,000
Russia.........	1,212,055	37,700.0	25,053,177	1,063,883	33,402.3	22,197,155	1,087,056	33,812	22,469,432
Spain..........	257	8.0	5,316	(*e*) 257	8.0	5,316	(*e*) 266	8	5,500
Sweden........	1,958	60.9	42,235	(*e*) 1,958	60.9	42,235	1,984	62	41,000
Turkey........	1,400	43.5	29,000	(*e*) 1,400	43.5	29,000	(*e*) 1,403	44	29,000
United Kingdom....	17,537	545.5	359,719	(*e*) 13,584	422.5	280,781	12,849	400	266,500
Africa:									
Madagascar....	233,199	2,024.1	1,345,121	66,258	2,060.9	1,369,553	56,585	1,760	1,169,608
Rhodesia......	3,779,621	7,238.3	4,820,223	348,518	10,840.4	7,203,865	479,089	14,902	9,902,873
Transvaal......	94,815	117,551.4	78,124,766	4,897,221	152,324.1	101,225,558	5,786,617	179,988	119,609,373
West Coast....	....	2,949.1	1,959,826	165,844	5,158.4	3,427,995	199,432	6,203	4,122,260
Asia:									
Borneo (British)	42,745	1,329.5	883,539	(*e*) 42,745	1,329.5	883,539	(*e*) 42,332	1,317	895,000
China (*e*)......	217,688	6,771.0	4,500,000	85,918	2,673.0	1,776,100	(*e*)217,688	6,771	4,500,000
East Indies (Dutch).....	50,797	1,580.0	1,049,967	68,426	2,128.0	1,414,500	55,636	1,419	1,150,000
India (British).	557,007	17,325.6	11,513,340	576,889	17,943.7	11,924,308	533,658	16,599	11,030,711
Japan (*g*)......	217,707	6,771.6	4,500,000	161,105	5,011.0	3,330,300	217,688	6,771	4,500,000
Korea.........	(*e*)48,379	1,505.0	1,000,000	(*e*) 58,055	1,805.7	1,200,000	(*e*)120,948	3,771	2,500,000
Malay Peninsula....	18,990	590.7	392,522	(*e*) 18,990	590.7	392,522	16,933	527	350,000
Australasia (*d*).	4,220,690	131,281.2	87,241,662	4,159,220	129,369.2	85,970,779	3,978,579	123,751	82,237,228
Other Countries	72,570	2,257.2	1,500,000	72,570	2,257.2	72,570	(*e*) 72,570	2,257	1,500,000
Totals.....	16,888,367	525,255.0	$349,088,293	18,360,945	571,422.5	$378,411,754	20,090,251	613,246	$407,658,920

(*a*) Figures based on exports and coinage. (*c*) Includes output from domestic ores only. (*d*) Six States and New Zealand (*e*) Estimated. (*f*) Includes Servia, Persia, West Indies, Formosa, British New Guiana and Philippine Islands. (*g*) Exclusive of Formosa.

Note.—The value of gold is $20.67 per troy ounce, which is equivalent to $664.55 per kilogram.

in turn stimulated the operation of known districts and the search for new goldfields. The improvements in mining practice and in methods of beneficiating ores continued to be important factors, a considerable

part of the gold production of recent years having been derived from old mines reopened or extended in their workings, after having been given up as unprofitable under earlier methods.

Silver was in demand throughout the year, and its price advanced to the highest point for several years. It shared in the general prosperity and the continued requirement for supplies.

SILVER PRODUCTION OF THE WORLD.

Country.	1905			1906		
	Oz. Fine.	Kilograms.	Value.	Oz. Fine.	Kilograms.	Value.
America, North:						
United States	56,101,600	1,745,318.0	$33,859,169	56,183,500	1,747,543	$37,525,521
Canada	5,974,875	185,843.7	3,605,957	8,568,685	266,522	5,723,110
Mexico (*a*)	65,040,865	2,023,418.0	39,254,309	68,500,000	2,130,638	45,751,835
Central America	1,361,449	42,355.0	821,697	(*e*) 670,000	20,838	447,500
America, South:						
Argentina	(*e*) 70,000	2,177.0	42,245	(*e*) 70,000	2,177	46,754
Bolivia	(*e*)6,600,000	205,287.7	3,983,100	(*e*)6,650,000	206,843	4,441,602
Chile	(*e*) 850,000	26,438.2	512,975	(*e*) 850,000	26,439	567,724
Colombia	(*e*)1,000,000	31,103.5	603,500	(*e*) 980,000	30,482	654,552
Ecuador	(*e*) 40,000	1,244.2	24,140	(*e*) 40,000	1,244	26,716
Peru	(*e*)5,000,000	155,521.0	3,017,500	(*e*)5,100,000	158,631	3,406,341
Uruguay	(*e*) 1,093	34.0	660	(*e*) 1,000	31	668
Europe:						
Austria	(*e*)1,279,633	39,801.4	772,284	(*e*)1,270,000	39,502	848,246
Hungary	(*e*) 525,723	16,352.4	317,284	515,000	16,019	343,974
France	(*e*) 617,955	19,221.0	372,948	19,500	607	13,024
Germany (*c*)	12,535,238	389,898.5	7,565,267	11,649,160	393,442	7,780,590
Greece	(*e*) 855,912	26,622.5	516,500	(*e*) 820,000	25,505	547,686
Italy	(*e*) 801,917	24,943.0	438,973	(*e*) 800,000	24,883	534,328
Norway	(*e*) 257,200	8,000.0	155,225	170,000	5,288	113,545
Russia	172,912	5,378.2	104,356	120,241	3,740	80,310
Spain	(*e*)3,774,989	117,418.0	2,278,281	3,825,000	118,974	2,554,756
Sweden	(*e*) 20,923	650.8	12,627	(*e*) 21,500	669	14,360
Turkey	(*e*) 572,342	17,802.2	345,420	550,000	17,107	367,351
United Kingdom	159,689	4,967.0	96,376	162,500	5,054	108,535
Asia:						
Dutch East Indies	182,889	5,690.0	110,386	185,000	5,754	12,356
Japan	2,409,879	74,971.0	1,454,437	(*e*)3,250,000	101,089	2 170,708
Australasia	14,362,639	446,738.4	8,667,853	13,519,410	420,510	9'029,749
Africa	619,620	19,276.0	374,154	(*e*) 495,000	15,397	330,615
Other Countries (*d*)	(*e*) 50,000	1,552.2	30,176	(*e*) 50,000	1,555	33,396
Total	181,338,362	5,638,183.3	$109,382,829	185,035,496	5,786,483	$123,475,852

(*a*) Statistics compiled from export and coinage. (*c*) Silver produced from domestic ores only. (*d*) The output is mostly from China and Persia. (*e*) Estimated.

Note.—Unless specified to the contrary, the statistics have been taken from official sources. The average commercial value of silver for 1903 was 53.45c. per ounce, equivalent to $17.18 per kilogram; for 1904 it was 57.221c. per ounce, or $18.40 per kilogram; and for 1905 it was 60.352c. per ounce, or $19.40 per kilogram.

Gold and Silver in the United States.

The gold production of the United States reached a total of $88,180,700 in 1905, the largest figure ever recorded up to that date. In 1906, according to the statistics of George E. Roberts, director of the mint, the total output was $96,101,400. The increase was largely due to developments in Alaska and the new mines of Southern Nevada. The latter are remarkable from the fact that they are in a region which was prospected and passed over many years ago, and then neglected until three years

ago, when the finds at Tonopah drew a multitude of prospectors into the country.

Colorado continued to hold the first place among the States as a gold producer, notwithstanding a loss in its total production. The increase of over 40 per cent. in Alaska carried that Territory to the second place, putting it before California, which showed a production nearly stationary. Nevada, with an increase of about 80 per cent., took the fourth place, South Dakota, Utah, Montana and Arizona following in order. There were no other changes of importance.

Colorado led in silver production in 1906, Utah taking the second place, while Montana dropped from first to third. Idaho and Nevada stood fourth and fifth, as in 1905.

Alaska.—The large production of 1906, and the great increase over the previous year, was due in part to the opening of several new districts and in part to more systematic operation in the older and better known fields. Nearly all parts of the Territory contributed to the gain. In the American Yukon, in Nome and the coast districts production continued large; while Fairbanks, the Copper river country and other newer regions contributed to the output. The great mines on Douglas Island worked steadily. Much progress was made in the introduction of machinery and more systematic operation.

Arizona.—As heretofore the gold production of Arizona was largely made in connection with the output of copper, which is now the most important mining industry of the Territory, and is reviewed elsewhere.

California. (By C. G. Yale.)—The immediate effect of the disaster of April, 1906, was to stop development at many mines in California; also in other places where San Francisco capital was employed. Those properties, however, which were on a producing basis were not in the least affected by the disaster. It was only those mines in process of development which were closed down. Most of these closed because no further funds were forthcoming and others because it was impossible to obtain machinery or have extensive repairs made. The latter feature inconvenienced even the productive mines. However, this effect on the mining industry was only temporary. The development of the new districts in Southern Nevada had an influence on California mines, by drawing away many men, and making labor scarce. Beyond this there are few matters to note regarding mining in California during 1906, aside from the usual progress.

The bulk of the gold continues to come from quartz mines, although the greatest increase of output occurred in dredging, which is the most progressive of all the forms of gold mining. There were upward of 50 dredges at work in the State, the largest number being in Butte county, in the Oroville field. The increase from this source was about $750,000,

which will be enlarged in 1907 as the new machines lately put in operation have so much greater capacity than the old ones. Some have been started in the Yuba river field, which are digging to a depth of 100 ft., the ground in that vicinity being very favorable. The only other important field is near Folsom in Sacramento county, where several of the largest machines are now at work. Dredges are also operating in Calaveras, Siskiyou and Trinity counties, but the fields at these places are not so extensive as in Butte, Yuba and Sacramento counties, where they work the bottom-lands near the mouths of the Feather, Yuba and American rivers.

The hydraulic, drift and surface placer mines do not evince any increase of output, and in fact are turning out less gold each year, though the deficiency is more than made up by the dredgers. The dredges now turn out more gold each year than the hydraulic, drift and surface placers of the State put together. The greatest gravel-mining counties for the old-style work are Siskiyou and Trinity, which yield half the output of the State from the hydraulic and placer mines. They have no restrictions in that section to hamper hydraulic mining, the tailings going into the Klamath river and tributary non-navigable streams. There are more active mines in Siskiyou than in any other county of California, but most of them are small ones. The most productive county in gold is Nevada, where quartz mines predominate, followed second by Butte, where auriferous gravels and sands are worked by the dredges.

Colorado. (By George E. Collins.)—The year 1906 was for Colorado, on the whole, uneventful and prosperous. No new bonanza of the first rank was opened, and the long expected new camp still remains undiscovered. The undoubtedly improved condition was due to two causes: the absence of labor troubles and the higher prices of metals, and especially of silver. New enterprises were relatively few in number. The only seriously disconcerting features in the outlook for the future are, first, the decadence of prospecting, due largely to the more alluring attractions of Nevada, and, secondly, the scarcity of skilled labor.

The production of the Cripple Creek district was about the same as for 1905. The Portland is still by far the largest shipper, followed by the Elkton, Vindicator, Granite, El Paso, Stratton's Independence and Golden Cycle. Several good orebodies were opened up by lessees, of which the Little Clara lease, on the Work property, is the most important. The remaining properties of the Woods Investment Company, including the Gold Coin, have been absorbed by the Granite company. The Findley was shut down, pending completion of the Golden Cycle mill; this mine and the Golden Cycle commenced a large output in the spring of 1907.

As to ore treatment, the proposed mill to handle the Independence dump by raw cyaniding is still in abeyance, and little progress has been made with the treatment of the large low-grade bodies of semi-

oxidized ore, and the dumps, by this process. The old Telluride mill was purchased by the Golden Cycle interests, and is being rebuilt as a roasting-cyanide mill. The Dorcas mill was destroyed by fire, and is not to be rebuilt. The projected co-operative drainage tunnel was not commenced until well into 1907, owing to disagreement among many of the principal interests involved as to the basis on which the expense should be apportioned.

There was a large output from Leadville. Toward the end of the year shipments were greatly hindered by shortage of cars. The properties of the Western Mining Company produced the largest tonnage, followed by the Moyer mine of the Iron Silver company, the various mines included under the administration of the Yak Tunnel Company, the Ibex (still operated mainly by lessees) and the Reindeer. The New Monarch mine was also a prominent shipper, and it is believed that its ore was of far better grade than the average. Developments on the Owers property in Iowa Gulch are reported to have been unfavorable, the shaft sunk through the wash having struck granite without encountering the ore-bearing limestone series. Perhaps the most salient single feature was the striking success of the Yak tunnel, which partly by supplying power and transportation of ores, but principally by leasing and working the properties opened up along its course, has become a very remunerative enterprise. Its success affords an excellent object-lesson to the many other tunnel schemes throughout the state.

The list of misfortunes in the San Juan region of rugged mountains and heavy snowfall was again added to. In the spring the Camp Bird mill was destroyed by snowslide and fire, while about the same time a snowslide carried away the buildings of the Shenandoah mine, near Silverton, with a heavy loss of life. Several other mills near Silverton were damaged by the same agency. The Silver Lake mill, also near Silverton, was totally destroyed by an incendiary fire. All of this damage, however, is being, or has already been, repaired; other properties are beginning to produce largely, and there is every reason to hope for a period of steady prosperity.

The Telluride district was more prosperous than for many years. The leases on the Smuggler-Union produced largely and profitably. The Tomboy company's Argentine mine continued to make a large output, and the grade of the ore considerably improved, so that the proportion of profit was higher than at any time since 1902. The Liberty Bell settled down to steady production, after the interruptions due to remodeling the mill. The vicinity of Ophir, on the other hand, was less active.

At Silverton, owing to the loss of the Silver Lake mill, the Gold King and the Sunnyside were the leading mines. The most prominent newcomer was Gold Prince, which erected at Animas Forks a mill of 500 tons daily

capacity, in the construction of which no expense was spared. With the aid of cheaper shipping costs, as a result of the extension of the Silverton Northern railroad to the same point, and a solidly built tramway from mine to mill, it is expected that the extensive low-grade orebodies opened up in the Sunnyside Extension ground last year can be profitably worked. It is certain that this district contains many other veins of almost or quite equal size, which will be successfully operated some day, whenever working costs have been reduced to a sufficiently low level; and it may be that the enterprise of the Gold Prince and Grand Mogul companies will prove this time to have arrived already. Prominent among the new producers near Silverton are the Old Hundred and Green Mountain in Cunningham Gulch.

At Ouray, the enforced suspension for seven or eight months of the Camp Bird and the continued inactivity of the American-Nettie were severely felt. On the other hand, the reopening of the old Red Mountain mines by the agency of the Joker tunnel, and the reworking of the Saratoga, led to a feeling of hopefulness for 1907. At the Revenue tunnel good discoveries in the lower workings resulted in a profitable summer's work. The Bachelor was idle, excepting for the work of a few lessees.

Among the minor districts, Breckinridge was benefited by the advent of a market for zinc ore; and several mills were in operation on zinc-lead ores. The Revett dredge on French Gulch had a successful season; and further developments in dredging are anticipated. Creede continued to ship a considerable tonnage, principally of low-grade silicious ores, Lake City and Kokomo are still awaiting a revival. The large zinc orebodies of Red Cliff and Rico shipped a small tonnage, which is likely to be increased in the near future. At Aspen the principal production was made by the Percy, La Salle and Smuggler mines; there was little, if any, increase in activity following the better price for silver.

Among the northern counties, the situation in Gilpin remained unchanged; except in the Russell Gulch district a large proportion of the best mines were idle. The Old Town is still the largest and most profitably producing mine; but the Pewabic is also handling a large tonnage. Nothing has yet been done toward carrying out the extensive program of development projected for the Gregory-Bobtail consolidation, with a view to proving the famous old Gregory, Bobtail and Fisk veins at considerably greater depth. The Gunnell changed hands, and the new proprietors re-erected the shaft house (which was destroyed by fire), and re-timbered the upper portion of the shaft, which was ruined at the same time, preparatory to reopening the mine. On Quartz hill all the principal mines were idle; the future of this district is bound up with that of the Newhouse tunnel, which is now believed to be within less than 2000 ft. from many of the principal veins, and is advancing at the rate of over 250 ft. monthly.

The neighboring Idaho Springs district continues to pin its faith to its deep cross-cut tunnels, planned to unwater and develop the old producing veins to the north and southwest of the town. Among the most important of these are the Newhouse, Central and Lucania, the ulterior objectives of which are various parts of Quartz hill, and the McClelland tunnel, aiming to strike the old Freeland property in depth. The Gem and Lamartine are at present being worked only by lessees and the Stanley is carrying out only development work. The largest producers were the Little Mattie on Chicago creek, and the Specie Payment, on Bellevue mountain.

At Georgetown there is today no bonanza mine, such as the Dives-Pelican, Colorado Central or Terrible of former years. Several mills are at work on the dumps and stope fillings left by former workers, and others are being built. Pending the discovery of new bonanza ore bodies, the milling of low-grade silver and zinc ores gives employment to a considerable number of men, so that the camp is by no means inactive. Considerably increased interest is shown in the veins of the Argentine district, which was opened up during 1906 by a narrow-gage railroad (the Argentine Central).

In Boulder county there was no increase of production, as compared with the previous two years; but much is promised for the future. The new mill under construction for the Wano mine, at Jamestown, is nearly finished; and as the preliminary testing was thorough and satisfactory, it is safe to predict that a success will be scored, if the ore bodies in the mine prove of the size and grade expected. A much larger mill is being erected to cyanide *en masse* a huge rhyolitic dike near the South Boulder, which is claimed to run on the average a couple of dollars or more. It has been known for years that some of the remarkable systems of dikes in this country contained gold, but the gold has hitherto been found too erratic in occurrence for economic work.

In placer mining some work was done by the dredges built a couple of years ago in the Clear creek valley near Golden. The successful operation of the new dredge on French Gulch near Breckinridge is referred to above. Hydraulic mining is carried on at the Twin Lakes placer, near Leadville, and the Keystone Placer, below Telluride; but in the latter case extensive caves of surface soil, resulting from the heavy snowfall in the spring, caused the suspension of operations.

There was no strike or difference of opinion between employers and employees in 1906, a condition of affairs which can only be attributed to the comparative decadence of the Western Federation of Miners in every camp throughout the State, excepting Silverton and Ouray, where there is still a leaven of old time independence and common-sense in the local unions. As a whole, it cannot be denied that wages and con-

ditions of labor are satisfactory throughout Colorado. The demand for miners everywhere far exceeds the supply, and the opportunities for advancement open to steady and sensible men, in contracting, leasing or by promotion to positions of responsibility, are excellent. There has as yet been no formal advance in the wage scale, excepting at Aspen, where the operators, in accordance with a pledge made years ago, raised the minimum daily pay from $2.50 to $2.75, in consequence of the improved profit margin resulting from the advance in silver. This ($2.75) is now the lowest wage paid to miners anywhere in the State. Any general advance would perhaps be a grave mistake, as the majority of mines find it difficult to make both ends meet even now. But indirectly, in the case of obtaining employment, and in the increased wage paid to the more competent men, labor is getting its full share in the present condition of relative prosperity, as compared with two or three years ago. At Cripple Creek and elsewhere, business men (storekeepers and saloonkeepers) complain of slack business; but business men are apt to forget that the mines in a camp should be run for the benefit of the mine-owners, rather than that of the town.

At Leadville the card system has been abolished, and some are of opinion that its abolition at Cripple Creek and Telluride will follow. Undoubtedly the system has been abused in some cases, and its essential object of protecting and providing constant employment for the steady and competent men has been lost sight of, but in my opinion it would be a great mistake to do away with the system, merely because a few errors have been made in its administration.

Idaho. (By Robert N. Bell.)—The gold output of Idaho in 1906 did not show a material increase. One of the main producers of the State, the old De Lamar mine, was practically out of commission, owing to the installation of a new method of treatment which was not completed early enough to make much of a show in production in 1906. The mine was developed right along, however, and shows bigger resources of ore (but probably not of as high grade as formerly) than at any time during its career. The most important part of Idaho's gold production will still have to be credited to placer operations and a continued increase may be anticipated from this source, as several new placer enterprises are afoot, involving extensive mechanical equipment.

A remarkable revival of interest occurred in the old Atlanta gold camp, in Elmore county, which produced ore to the value of $5,000,000 from its secondary enriched surface horizons 40 years ago. The ore, on development, showed a slight mixture of antimony, and proved difficult to treat, and the camp has been quiet for a number of years. The Monarch mine, at this camp, is installing a hydro-electric power plant, an aerial tram three-fourths of a mile long and a new mill of 200 tons daily capacity

adapted for amalgamation, concentration and cyaniding; the ore is said to be suited to this treatment, and will make a good yield, even though one-fourth of its value is in silver. Most of the machinery is on the ground, and the management intends to have the plant in full operation early in 1907; the mine has 1,000,000 tons of developed ore, worth $5 a ton and upward. The Minerva mine, in the same district, operated a 10-stamp mill throughout 1906, producing from $7000 to $10,000 per month by plate amalgamation, while the tailings, which are being saved for further treatment, are said to run from $5 to $7 per ton. The Pettit mine, in the same district, has $1,000,000 of developed ore with average of $10 gold per ton, and the owners are installing a 50-ton mill.

The Golden Sunbeam mine, near Bonanza, Custer county, developed a free-milling orebody 10 ft. wide and something over 200 ft. long, that yields $30 to $50 per ton. It will shortly be equipped with a new mill, and should become an important producer of gold. The gold in this ore, which is a volcanic tuff, is very coarse and free and shows a saving of 85 per cent. by simple amalgamation. The mine produced $60,000 with a small prospecting mill last summer after the strike was made.

The output of silver in Idaho for 1906 was derived chiefly from the Cœur d'Alene lead ore but included 500,000 oz. from the Snowstorm copper ore, and 750,000 oz. from the silver-gold ore of the Trade-Dollar mine at Silver City.

The Old Ramshorn mine, at Bayhorse, in Custer county, has a rich deposit of dry silver ore with large resources blocked out. The ore of this mine is a siderite gangue carrying gray copper, and yields on an average 100 oz. silver per ton. This property, together with a number of important lead-ore deposits in the same neighborhood, was carefully studied during the autumn, with a view to forming a combination and building a smelter, for which the district presents a fine opportunity. It was formerly a big producer of silver and lead, but has remained practically idle excepting for small leasing operations since the slump in silver in 1893.

Montana. (By B. E. St. Charles.)—The output of gold in 1906 was satisfactory, every county contributing to the aggregate. As usual, a large part of the production came from the smelters at Butte. The Cable lease and Southern Cross companies were banner producers. Both operated almost continuously; the former treating the output of its mines in its own mill and the latter shipping to the Washoe works part of the time and using its own plant the remainder. Fergus county was the most productive, the Barnes-King, Gilt Edge, Kendall and other mines in that county (in which there are immense bodies of fair grade ore) yielding continuously. The Barnes-King property was sold in November at a figure said to have been $1,250,000, Butte and Eastern men taking

it and organizing the Barnes-King Development Company. Up to April 30, 1906, Barnes-King had yielded $1,527,126 gross, of which sum $547,-800 was paid out in dividends at the rate of about $144,000 a year on 210 tons daily treatment. The average yield per ton is $5.95, and the cost of mining, etc., $2.80 per ton. Total extraction for 1906 was 49,384 tons. The quantity of ore in sight on April 30 was 400,281 tons.

The Green-Campbell Consolidated Gold Mining Company started its new mill about Oct. 1. This mine is located in the Silver Star district, 55 miles southeast of Butte. During the three months' operation it crushed about 3600 tons of ore that averaged $10.50 a ton.

Gold dredging became more prominent in the State in 1906 than ever before. Five new boats were put in operation, one by the Conrey Placer Mining Company, operating at the mouth of Alder gulch, Madison county; one by the Supreme Gold Dredge No. 1, on Hughes creek, in the western part of the State; one by the Trustees Gold Company, on Basin creek, Jefferson county; one by a Kansas City syndicate on Cottonwood creek, Deer Lodge county, and the other by the Gold Creek Company, Ltd., on Gold creek, Granite county. Another was built for work on the Jefferson river, 45 miles southeast of Butte, but faulty construction prevented its use. The Conrey company operated three large boats within half a mile of each other and extracted a large amount of gold. Three companies suspended work as soon as the cold weather set in, but will resume in the spring. The boat on Cottonwood was designed for saving gold and sapphires, both of which the gravel contains.

Nevada. (By W. H. Shockley.)—There was more stock gambling in Nevada in 1906 than at any other time in the history of the State. In a single day over a million shares were sold in the San Francisco Stock Exchange. Much of the money has been furnished by Easterners who have found that Nevada mining shares offer as good a chance of profit as horse races or other sports. The great fortunes made by some of the speculators will attract capital for a long time to come.

The great feature of the year was the rich strike in the Mohawk mire at Goldfield. Ore was discovered by Hayes & Monnette on their lease at a depth of about 250 ft. in April; this strike caused a rush for other leases, and now the whole hillside is covered with hoists and looks like an active oilfield with its forest of gallow-frames. These leases mostly ended on Dec. 31, and work was pushed as fast as possible, in some cases with six-hour shifts paying miners $5 per diem. Electric hoists are thought better than gasoline, but an electric pump was discarded in one lease owing to dangerous shocks given workmen. A few air-hammer drills used gave great satisfaction to the owners but not so much to the men, the dust being very trying; they are locally called "six-shooters." From April, 1906, to Jan. 1, 1907, the Mohawk mine produced $6,020,000; nearly all of this

was extracted after August 1, 1906. Nevadans claim this as a record production for the first eight months after ore discovery. The best shipment was that made by Hayes & Monnette, less than two carloads netting $700,000.

The Mohawk and the adjoining mines—the Jumbo, Red Top and Laguna—have been merged into the Goldfield Consolidated by Senator Nixon and George Wingfield. This company has 156 acres and a capital of $50,000,000 in 5,000,000 shares of $10 each. Messrs. Nixon and Wingfield charged the modest commission of 250,000 shares for making the combination. The shares are selling now for about par. The Combination mine has also been purchased and will be consolidated with the above mines. The engineers of the Goldfield Consolidated estimate the blocked out reserves at $12,338,000, excluding low-grade ore. Looking into the depths they see a production of $30,000,000, gross assay value, from the ore shoots already known, with a reasonable expectation of an immensely larger amount from the three acres near the Mohawk, without mentioning the remaining 153 acres; from all this it is likely the Goldfield Consolidated will give some surprising returns.

Goldfield is one of the most active places on earth, but much of the activity is misdirected. It has increased greatly in population—the present estimate is 15,000. There was much suffering in 1906 because of lack of fuel. Owing to the lack of rooms many of the poorer men slept in the saloons or in unfinished buildings. The death rate was high; as many as 20 deaths were reported in a day, but there is much difficulty in learning the true state of affairs.

Goldfield is the scene of the operations of the L. M. Sullivan Company, whose personnel and methods would make a startling history. From the sale of their widely advertised stocks their receipts reached a maximum of $140,000 in a single day. These great receipts were overbalanced by their heavy expenses, of which advertising was the largest item, and in January, 1907, their drafts were protested and the company reorganized. The new president is T. B. Rickey of the State Bank.

"High-grading" is regarded by many miners as a legitimate perquisite analogous to secret commissions. Mohawk leasers without the rich streak found it difficult to keep miners who constantly tried new employers till they found work near the rich ore. Premiums were offered for work in the Hayes-Monnette lease where a man could easily carry away $1000 in his coat pockets. The high graders were greatly aided by wagons gathering up shipments at dead of night. The Mohawk management attached a large shipment of ore in the possession of Wells, Fargo & Co. and another shipment at the Selby smelter, San Francisco, but the dispute about this ore has not been settled.

In January, 1907, a strike was feared, the proposed establishing of

"change houses" by the mine owners being seriously objected to by the miners as a reflection on their honesty. In these change houses the men will be required to change their digging clothes for ordinary clothes under supervision, thereby lessening the chance for high-grading. The settlement of this dispute finally allowed change houses in a modified form, the wages of the miners being increased to a minimum of $5 per eight-hour shift.

In August, 1906, there was trouble with the Independent Workers of the World and the newspapers. This was settled by somewhat forcible measures. The terms of the settlement were secret, but it was currently reported that one of the conditions was that no more men should be run out of town by the miners. However, men were driven out as late as January, 1907. Many men were driven from Goldfield; all were men from Colorado who had taken the part of the mine-owners in the labor war at Cripple Creek.

No important discoveries seem to have been made in Goldfield except in the immediate vicinity of the Mohawk. There were some small strikes in the Great Bend, Daisy and other Diamondfield mines.

The Bullfrog district was rather quiet during 1906 and nothing phenomenal was reported. The Gold Bar mine is said to have a very wide ledge of rather low-grade ore. The banner mine is still the Montgomery-Shoshone, where work has been pushed. Here the visitor is shown 65 ft. of white talc said to average $240 per ton. It is a characteristic of nearly all these Southwest-Nevada gold mines that the ore shows but little evidence of its value and would be thought worthless by most expert miners from other places. Men are now grading at the Montgomery-Shoshone for a 100-stamp mill, which is expected to be running in March. Several other companies are talking of putting in mills, and the Bonny Claire mill will soon be running. Up to date the Tramp Consolidated seems not to have decided about its mill.

A few miles north of Beatty was the camp of Transvaal, where there was a lively excitement for a few weeks in April and May. Town lots were sold for $100 to $400 and for a time there was a population of several hundred. The camp is now almost deserted.

A little work is being done in many camps south of Tonopah. A rush was made for the Saline valley copper deposits in September, but nothing seems to have come of it. The silver camp of Klondyke seems to be coming to the front; a number of claims are being opened there. This camp antedates Tonopah. Rich bunches of silver ore are found in strong quartz ledges.

Tonopah has gone on steadily; a threatened strike of the miners was avoided by wise management in the summer, and wages are now settled for three years on an eight-hour $4 basis. The 100-stamp mill of the Tonopah Mining Company, at Millers, is now running, and the 40-stamp

mill of the Montana Tonopah will soon be in operation. The camp has been free from litigation for a long time, but an apex suit has been started by the West End Consolidated against the MacNamara.

Manhattan was started in December, 1905, and by April 1906 was a town of 3000. The mines here were largely financed from San Francisco and the earthquake caused great depression; later things picked up a little on the report of the coming of a strong company bringing in water and building a large custom mill. There are some promising properties in Manhattan, but they are cursed by litigation; the richest part of the camp is well named Litigation hill, as all of the mines are tied up with from one to seven suits. Nevada laws and custom particularly favor litigation; 90 days is allowed for doing location work, and the claims are usually marked only by the center monument. This gives a chance of swinging the claim, should any neighbor make a good strike; then the prospectors have a pleasing way of destroying monuments and location notices. Great hardships are caused by the loose system of taking up ground. But little ore was shipped from Manhattan in 1906; the average ore is too low grade to stand the long haul to the railway. About 20 miles north of Manhattan is Round Mountain, where there have been some rich gold discoveries. Great things are promised for this camp.

On Oct. 29, there was a lively scene at Walker Lake. The Indian reservation was thrown open on that day and some 4000 people took part in the rush; most of them found men already in possession of the best claims. Much litigation will result should any good discoveries be made; so far, copper prospects are giving best promise. It is said that 2000 claims located on Oct. 29 will be abandoned.

The Blair mines at Silver Peak were sold during 1906. A 20-mile railway connecting with the Tonopah & Goldfield Railroad, at Blair Junction, has been built and the town of Blair started. This should be a good camp, for the Blair mines have been long and favorably known. The ore exists in large amount; the average is said to be a little below $10 gold, and with the wide ledges the costs should not be over $4 per ton, leaving a good margin of profit. The Valcalda mines, near the Blair mines, are said to have $2,000,000 in sight.

Thirty-five miles northeast of Luning are the camps of Goldyke and Atwood; a 10-stamp mill is being erected at Goldyke on a wide low-grade ledge about which all sorts of optimistic predictions are made.

Fairview has some good mines, the Nevada Hills being a specially good property. The town of Wonder, near Fairview, is young, but has some rich silver mines. All speak well of this camp and prophesy it will soon outrank Tonopah and Goldfield.

Seven Troughs and Rosebud are among the new camps; both are near Humboldt. Rosebud shows white porphyry with specks of ruby and

stephanite; assays of croppings give up to $4000. Rosebud is said to be more promising than was Tonopah at the same age. Phonolite is also very new, the mines having been found in August; this is 12 miles north of Ione, an old camp. The Silent Friend mine averages $35 for a 4-ft. ledge, with streaks giving higher assays.

Many railways are projected for Nevada; the San Pedro, Los Angeles & Salt Lake Railroad is already at Rhyolite and the Tonopah & Tidewater Railroad will soon reach the same place. This is Borax Smith's road, and will be continued to join the Nevada Central at Austin, probably touching Manhattan and Tonopah en route.

New Mexico. (By Charles R. Keyes.)—There were notable developments in the mining of placer gold in 1906, but the output of the lodes remained about the same as in former years. It may be expected that the great increase of production of the complex ores of the baser metals will soon add largely to the gold production. Rich placers were demonstrated and several undertakings for their exploitation were begun. The most important new developments are in the Rock mountains, near Elizabethtown, in the Tuerto mountains, near Golden, in the Caballos mountains, at Apache cañon, and in the foothills of the Mimbres range, east of Hillsboro. Many other points of minor note might be mentioned. About one-half of the total gold product is now taken from placers.

West central New Mexico continues to be the chief lode-mining district. Mogollon and Rosedale are the principal gold-mining districts in Socorro county. At Hillsboro much new work has been done recently. In Grant county most of the gold comes from the vicinity of Silver City. Near Jarilla, in Otero county, some good quartz veins are yielding gold. Operations were lately instituted to explore more extensively than ever before the deep mines of Lincoln county, particularly the Old Abe, now down 1500 ft., which is the deepest mine in New Mexico.

Recent investigations appear to show conclusively that in nearly all of the many mountain ranges of New Mexico gold appears in workable grade. In many of the districts good placers are also to be found, and the problem of a sufficient water supply for exploiting these placers has now been solved. Another noteworthy circumstance, and one which should greatly stimulate mining, is that the geology of New Mexican gold has been carefully worked out, so that most of the uncertainties as to occurrence and probable extent of deposits have been removed.

The output of silver showed an increase over the previous year; but, considering the former records over of New Mexico, there is not nearly the activity that there should be. Western Socorro county continued to be far in the lead, with Grant county second. Doña Ana, Luna and Sierra counties are also among the principal producers. Most of the mines which a dozen years ago produced high-grade ores still remain

closed. Large augmentations of the silver output may, however, be expected from the complex ores of the baser metals now being so largely developed, as ores of this class almost invariably carry enough silver and gold to warrant recovery.

Oregon.—A good deal of prospecting and development work was done in Eastern Oregon, and some of the older mines around Baker City were worked on an enlarged scale. Generally, however, there was no special incident, but production showed an increase.

South Dakota. (By John V. N. Dorr.)—The expectation of an increased production of gold in South Dakota in 1906 was not fulfilled. In fact, the production fell off slightly. This was due, among other things, to the fact that the Limestone district dropped out as a producer; the one mill operating there in the fall of 1905, that of the Spearfish Gold Mining Company, shut down in the spring and the two 200-ton mills of the Eleventh Hour and Victoria companies, which were just being completed a year ago, shut down after a short campaign.

The chief event of 1906 was the change of the Homestake Mining Company to an eight-hour basis. The whole force of more than 2000 men, including miners, mill men, teamsters, and wood-choppers, were given an eight-hour day with the same wages previously paid them for 10 and 12 hours. It is understood that the management does not expect that the change will increase the cost of operating materially, as most of the miners actually worked little more than eight hours and with the shorter hours for mill men the number of men per shift can be decreased. The output of the Homestake company for 1906 showed an increase, although the slime plant which was expected to start in the fall was delayed on account of the impossibility of getting material.

The Maitland, Imperial, Dakota, Wasp No. 2, and the Lundberg, Dorr & Wilson mills, treating silicious ore, were operated throughout the year and the Golden Reward until the fall, when it was shut down to change from dry to wet crushing. The Gild-Edge Maid mill was changed from fine wet crushing to coarse dry crushing and is reported to be making some profit on the low-grade ore. The Kildonan mill of the Mogul Mining Company, which was changed to a wet crushing mill, started up, using four 6-ft. Monadnock Chilean mills to do the grinding, with one 5x18 ft. Abbé tube-mill to regrind the sands from the highest-grade ore treated. The Reliance company, holding property in the Portland district, completed a 150-ton works in which Huntington mills are used. As the Golden Reward put in both Chilean and Huntington mills, the comparative value of these two grinders on the hard silicious ores will probably soon be determined.

No new discoveries of great importance were made during 1906, although several companies did considerable development and were able to block

out large quantities of the low-grade ore. There was considerable prospecting in the southern hills, but no heavy production was made from there. The smelter at Rapid City was in operation and afforded a nearer market for the blue ores produced in the district, but cannot compete with the cyanide mills on oxidized ore unless it runs over $20 in value.

It is impossible to foretell the effect of the anticipated change to eight hours on the mines outside the Homestake, and what its influence will be on the production in the Hills. If no trouble ensues, the additional mills now in commission and the operation of the Homestake slimes plant should cause a material increase in the production in 1907.[1] It is true in this district as in many others, that the increase in production for the last few years has been due almost entirely to the advance in metallurgical practice and the consequent reduction in the cost of operation, for, leaving out the Homestake, four-fifths of the ore now being treated could not have been handled at a profit with the milling costs of five years ago.

Utah. (By L. H. Beason.)—The principal gold mining in 1906 was done by the Consolidated Mercur Gold Mines Company at Mercur and by the Annie Laurie and Sevier Consolidated mining companies, in the Gold Mountain district of Sevier county. The Consolidated Mercur up to Dec. 1, 1906, mined and milled an average of a little more than 800 tons of ore daily at a total cost of $2.48 per ton. In November the directors authorized the management to proceed with the installation of a new slimes plant at an expense of about $25,000. The tailings have in the past averaged close to $1 a ton; the average during 1906 was 95c. By the new process, the management expects to bring the tailings down to as low as 50c. per ton.

The only other mine operated in the Mercur district during 1906 was the Sacramento, which produced about 6000 oz. of gold and over 1200 flasks of quicksilver. This is the only quicksilver producer in the State. It is reported, that the high-grade cinnabar ores are nearly worked out and, while the mine still contains large bodies of low-grade ores, there will be a falling off in the revenues from this source in 1907 unless other bodies are developed. The Overland mine, which is equipped with a good mill, has been closed, owing to the refractory character of the ore.

In the Gold Mountain district, Sevier county, the Annie Laurie company carried on a vigorous campaign of development and kept its cyanide mill in operation. The Sevier Consolidated placed its new 125-ton mill in operation and is said to be working successfully. The Sevier Consolidated ore averages $10 per ton.

Alta, in the Big Cottonwood district, promises to be a large contributor of gold, silver, copper and lead ores from now on. The Columbus Con-

[1]This prediction will not be fulfilled, owing to the disastrous fire in the Homestake mine in April, 1907, which necessitated the flooding of the mine and the closing of all its mills. *Editor.*

solidated is the best developed property in the camp, but large bodies of mill ore have been proved in the Albion and City Rocks mines. Both mines contain considerable high-grade ore and shipments are being made. The old Flagstaff and Emma mines were sold to Salt Lake and Eastern syndicates, resulting in the formation of the Consolidated Flagstaff and the Emma Copper companies. A consolidation of the South Columbus and Alta Quincy mines, under the title of the South Columbus Consolidated, was effected.

Placer mining was carried on in a limited way in Grand and San Juan counties.

Washington.—The chief incident in mining in Washington was the working of several mines in the northern part of the State by some of the large companies in the Boundary district of British Columbia, which find the ores of the Republic and other districts advantageous for use as a flux in smelting. There was a good deal of prospecting in some parts of the State, but the production did not increase.

Production of Gold and Silver in Foreign Countries.

Africa.

The three large gold producing regions which make the African continent at present of great importance are the Transvaal, Rhodesia and the West Coast. The attempts to revive the ancient gold mining industry of Egypt and the Sudan have so far resulted in very little, and it does not appear probable that they will attain any importance. The French exploitation of Madagascar has also been attended with only a moderate degree of success.

Transvaal. (By W. Fischer Wilkinson.)—The value of the gold won in the Transvaal in 1906 was £24,579,987. The Transvaal production

PRODUCING SECTIONS OF THE WITWATERSRAND.

Section.	No. of Producing Mines.	No of Stamps.	Percentage of Value of Production.
Randfontein to Champ d'Or	11	840	10
Princess to Langlaagte Central	9	720	7
Langlaagte Estate to City and Suburban	15	2,110	31
Meyer and Charlton to Witwatersrand	22	3,080	37
East Rand Mines	7	1,145	15
Totals	64	7,895	100

is mainly derived from the Witwatersrand area, the outside districts contributing only about 4 per cent. of the grand total. It is also interesting to note that 68 per cent. of the Witwatersrand output comes from

the section between the Langlaagte Estate on the west and the Witwatersrand mine on the east, containing a continous line of mines covering a distance of 12 miles. The accompanying table, compiled from one month's returns, shows the relative importance of different sections of the Witwatersrand goldfield.

The large increase in the production of 1906 over 1905 is due more to the increased scale of operations than to the accession of new mines. In January, 1906, there were 63 producing companies in the Witwatersrand district, and in December, 65, the two new companies being the Cason of the East Rand, and the Luipaardsvlei of the West Rand. The number of stamps dropping in January was 6920, while in December the number was 8070. The use of tube mills largely increases the milling capacity of the stamps, and the tonnage treated is now the most trustworthy basis to use for measuring progress. During January, 1906, 1,008,929 tons were milled in the Witwatersrand district, and in December 1,268,089 tons.

During 1906 some of the mines decided to stop the practice of keeping gold reserves for the purpose of equalizing the monthly returns, and gold to the value of £186,608 from this source is included in this year's returns. The accompanying table gives the figures of production and tonnage of the Transvaal from the commencement of gold mining.

TRANSVAAL GOLD PRODUCTION BY YEARS.
(Chamber of Mines Returns.)

Year.	Witwatersrand District.			Outside Mines Value.	Transvaal Total.
	Tons. Milled	Value.	Value per Ton Milled		
		£	s.	£	£
1884–9	1,000,000	2,440.000	48.8	238,231	2,678,231
1890	730,350	1,735,491	47.4	134,154	1,869,645
1891	1,154,144	2,556,328	44.2	367,977	2,924,305
1892	1,979,354	4,297,610	43.4	243,461	4,541,071
1893	2,203.704	5,187,206	47.0	293,292	5,480,498
1894	2,830,885	6,963,100	49.2	704,052	7,667,152
1895	3,456,575	7,840,770	45.2	728,776	8,569,555
1896	4,011,697	7,864,341	39.2	739,480	8,603,821
1897	5,325,355	10,583,616	39.74	1,070,109	11,653,725
1898	7,331,446	15,141,376	41.3	1,099,254	16,240,630
1899	6,872,750	15,067,473	43.84	661,220	15,728,693
1900	459,018	1,510,131	65.82	....	1,510,131
1901	412,006	1,014,687	49.25	81,364	1,096,051
1902	3,416,813	7,179,074	42.00	74,591	7,253,665
1903	6,105,016	12,146,307	39.79	442,941	12,589,248
1904	8,058,295	15,539,219	38.46	515,590	16,054,809
1905	11,160,422	19,991,658	35.82	810,416	20,802,074
1906	13,571,554	23,615,400	34.80	964,587	24,579,987

The average yield of the Witwatersrand mines, in 1906, amounting to 34.5s., exclusive of the gold reserves, per ton milled, again shows a decrease. The average reduction in grade is due to the increased scale of operations, and the consequent reduction in the working costs.

Classified according to the value of recovery the September returns show the following figures: A yield of 20 to 25s. to three companies; 25 to 30s. to 21; 30 to 35s. to 20; 30 to 40s. to eight; 40 to 45s. to nine; and upward of 45s. to four.

The cost of development, which figures in the costs as development redemption, becomes proportionally reduced as the tonnage is increased. It may be well to explain that it is customary on the Rand to charge all development work, except main shafts, to capital account, and to write off to working costs a round figure per ton milled, calculated on the available ore reserves. These figures have in the past been estimated on a conservative basis, and it will be evident that by including ore, which was formerly considered unpayable, the average cost of the ore reserves will be considerably reduced.

As to how development expenditure should be treated in the accounts has been the subject of discussion—some arguing that when a mine reaches the milling stage the development account up to that date should be closed, and that only the current development, which should of course be kept up to the scale of milling, should be charged to working expenses; others maintain that the development account should be gradually liquidated by charging an amount to working costs equivalent to what the ore reserves stand at in the books. As the development costs of a mine are usually highest in its early years, this system makes the working costs higher than they actually are when milling commences, and is, on this account, misleading.

The result of charging to working costs a higher figure for development than what it actually costs when the producing stage is reached, is to create a sinking fund for the redemption of capital expenditure. If this policy is right, the redemption of all capital expenditure should logically be provided for and not only of the money spent on development.

During 1906 several amalgamation schemes were carried through with the object of increasing the working area, and of obtaining the advantages of large plants, and of reducing management expenses. Some of the smaller outcrop mines are now approaching exhaustion, and the ground remaining can, in certain cases, be worked with advantage by the neighboring deep level. Among such amalgamations may be mentioned those of the Wemmer and the Village Main Reef; the Henry Nourse and the Nourse Deep; the Geduld Group; the Klip Deep and South Wolhuter, and the South Geldenhuis Deep, South Rose Deep, Rand Victoria and Rand Victoria East, which four companies will in future become the Simmer Deep.

These amalgamations have on technical grounds much to recommend them. In the early days of the Rand the parcelling out of the ground

was done on too small a scale, and the boundaries were in many cases irregular and unsuited for economical working. But exception has been taken to some of the amalgamations on the ground that the shareholders in the different mines have little say as regards the terms, which are practically arranged for them by one or other of the big financial houses. Take, for instance, the case of the Henry Nourse and Nourse Deep amalgamation. The former mine was one of the old-established outcrop mines, which had for years paid regular dividends and had an assured future before it of about 17 years. Judging by its past results and by the favorable statements of the directors as regards the future, the mine appeared a particularly sound mining investment. The controlling house, however, suddenly proposed and carried out a scheme for amalgamating the company with another neighboring mine also under its control, and while in theory the property of the Henry Nourse shareholders was improved by the fusion with another mine, due to possible advantages of longer life and cheaper working costs, the practical result to them has been that, instead of their customary dividend of over £100,000 a year, their share in the profits of the combined concern in 1906 was only £56,250. In the face of these figures it seems that the objections raised to this amalgamation were not without weight.

In some cases economy of working has been sought by erecting a joint plant for two mines, each having its own stamps, tubes, and cyanide plant, but sharing in the power station. The Knights Deep and Simmer East have a joint plant of 400 stamps and three tube mills, this being the largest reduction plant on the field. The Crown Deep and Robinson Central also have a joint mill, and the East Rand companies are also laid out on the coöperative principle.

The use of tube mills was rapidly extended. In August, 1906, there were, according to *South African Mines*, 58 tube mills on the Rand. Their use enabled higher extractions to be obtained, and an increased tonnage to be put through the battery, as coarser screening can be used. The Knights Deep, for instance, put through in September 7.24 tons per stamp per 24 hours, while the Luipaardsvlei mill deals with about eight tons, using 400-mesh screening. Both mines have stamps weighing 1550 ℔., and maintain an extraction of over 90 per cent. As regards fineness of ore reduction, W. A. Caldecott, the consulting metallurgist to the Goldfields group, states that now 97 per cent. of the total ore passes a 60-mesh screen (0.010 in. aperture), and that the stamp battery's economic limit appears to be 92 per cent., the rest being done by tube mills.

There is not much variety in the methods of milling and cyaniding. Denny Brothers, however, introduced at the George Goch and Meyer & Charlton mills a new method in which the use of cyanide solution

in the battery and filter presses in the extractor house are the chief features. A detailed account of these plants was given in a lengthy paper read before the South African Association of Engineers. It is too early yet to say whether the new process is more economical than that in common use. The inventors may, however, be congratulated in having struck out a new line, and in having erected a plant which, as regards first cost, is said to compare favorably with the standard plant of the field. Among metallurgical improvements may be recorded the use of zinc fume for precipitating gold dissolved in acid washes from the clean-up of zinc gold slimes, a process which is to be tried on working solutions. Belt conveyors for ore and sand have come largely into use.

A new departure as regards management which is being tried at one of the Goldfields mines is in placing the whole of the reduction plant under one manager instead of having separate managers for the mill and cyanide works. This change has been successful, and it is probable that the system will be extended.

As an example of the present-day costs of mining and milling on the Rand the costs of the Simmer & Jack mine for the year ending June 30, 1906, are given: *Data*: Tons milled, 624,507; sorting, 16.83 per cent.; number of stamps, 320; no tube mills. Yield of fine gold per ton, 7.572 dwt. Value, 31s. 10.453d. *Cost per ton milled*: Mine expenses, 10s. 4.513d.; hauling expenses, 1s. 5.838d.; pumping expenses, 5.796d.; transport of quartz, 2.717d.; crushing and sorting, 7.322d.; mill expenses, 1s. 8.328d.; cyanide expenses, 1s. 5.573d.; slimes expenses, 6.691d.; development redemption, 2s. 6.000d.; maintenance of buildings, 0.937d.; roads and surface improvements, 1.156d.; general charges, 9.891d.; total, 20s. 4.762d. Of the ore mined 47 per cent. was broken by hand and 53 per cent. by machines.

The report of the Rand mines group of nine mines for 1905 gave the following average figures based on 2,023,239 tons milled. The figures are of interest as showing the wide difference there is in the grade of the different mines.

Item.	Highest.		Lowest.		Average.	
	s.	d.	s.	d.	s.	d.
Revenue per ton	53	10.534	28	2.051	35	3.732
Expenditure per ton	29	1.680	19	3.407	22	9.824
Working profit	32	2.822	1	0.335	12	5.908

The report of the Consolidated Goldfields gives some interesting figures as regards the amount paid in wages on the Transvaal gold

mines during the year ending June 30, 1906. They are: White men, £6,341,706; Chinese, £2,742,239; Kafirs, £1,079,908. Gold to the value of £22,100,707 was recovered during the same period, and was distributed as follows: To working costs, £14,637,043 (66.23 per cent.); to dividends, £5,234,750 (23.69 per cent.); to profits tax, £475,000 (2.15 per cent.); to reserve fund, debenture redemption and interest, machinery, renewals, etc., £1,753,914 (7.93 per cent.); total, £22,100,707 (100 per cent.). The working expenses of the producing mines of the Goldfields companies were approximately in the following proportion: White wages, 34.5; colored wages, 19; stores, 39.5; sundries, 7; total, 100 per cent.

The profits earned by the mines show a general increase over 1905. Thirty-nine companies in the Witwatersrand district paid dividends amounting to £5,565,969; and three in the outside districts £169,192, or a total of £5,735,161.

DIVIDENDS OF GOLD MINES OF THE TRANSVAAL.

Year.	£	Year.	£	Year.	£	Year.	£
1887......	12,976	1892......	879,320	1897......	2,707,181	1902......	2,121,126
1888......	112,802	1893......	955,358	1898......	4,848,238	1903......	3,345,502
1889......	432,541	1894......	1,527,284	1899......	2,946,358	1904......	3,927,830
1890......	254,551	1895......	2,046,852	1900......	....	1905......	4,857,539
1891......	334,698	1896......	1,513,682	1901......	415,813	1906......	5,735,161

Of the 65 producing companies in the Witwatersrand district 39 paid dividends during 1906. The mines of the Central Rand are the largest profit earners, seven mines distributing 42 per cent. of the total dividends of the Witwatersrand. The Robinson mine heads the list with a distribution of £550,000.

The finance companies did not do so well as the mines, owing to the heavy fall in the value of South African mining shares, necessitating the writing off of large sums for depreciation of securities. According to the *Investors' Chronicle*, the market value of 44 leading companies, which was £154,509,000 on Dec. 12, 1904, had fallen on Nov. 26, 1906, to £70,308,489. Goerz & Co., for the year ending Dec. 31, 1905, wrote off £331,851. The General Mining and Finance Company for the same period £230,164, and the Consolidated Goldfields of South Africa, for the financial year ending June 30, 1906, wrote off £700,000. All these companies passed their dividend, as did also the East Rand Mines.

The mining industry continued to suffer from an insufficient supply of unskilled labor. The accompanying table gives statistics of the labor supply during 1906 and previous years.

PERSONS EMPLOYED IN GOLD MINES AT END OF MONTH.

	White.	Colored.	Chinese.		White.	Colored.	Chinese.
1902—July	8,162	32,616	....	1904—December	15,023	83,639	20,885
December	10,292	45,698	....	1905—June	16,939	104,902	41,340
1903—June	11,825	66,221	....	December	18,159	93,831	47,267
December	12,695	73,558	....	1906—June	17,959	90,882	52,352
1904—June	13,413	74,632	1,004	December	17,495	98,156	52,917

In December, 1906, the labor employed on all mines, including gold, coal and diamonds, was: Whites, 18,827; colored, 116,389, and Chinese, 52,917.

The present labor force is concentrated on the best producing properties, while the less promising properties, and those which are some distance from the productive stage, have to remain comparatively idle. Owing to the uncertainty as regards labor and in some cases to the difficulty of raising fresh capital, work on the unproductive mines has been reduced since the beginning of 1906. In January there were 57 developing companies, using 173 machine drills; while in September there were 48 companies, using 146 drills.

The Chinese supply has ceased, and during 1907 about 20,000 will be free to return home on the expiration of their three years' engagement. Re-enlistment may take place, but in any case it seems that this labor will be a decreasing quantity. The Chinese now represent about a third of the unskilled labor supply, and their repatriation must necessarily mean a considerable contraction of mining work and of gold production, and the dismissal of a large number of whites.

There is some hope that an important economy of labor may be brought about by the introduction of small machine drills, which can be put into the hands of Kafirs. But the solution of this problem has not yet been obtained, although the experiments made in this direction appear to be meeting with success. The mining correspondent of the *African World* (Nov. 17, 1906) says that experiments at the Robinson mine with the Gordon stope drill have shown that a Kafir can put in eight 3-ft holes with this machine against one by hand. It seems probable that a reliable stope drill will before long become an accomplished fact. Its adoption must bring about a great economy of labor, and will go a long way toward solving the labor problem.

Further, a reduction in the cost of living, which time will no doubt bring, will also tend to alleviate the position by making it economically possible to employ more Europeans. But a reorganization of the methods of working on these new lines cannot be done in a moment, and it does not seem that there is much hope for any rapid expansion in the industry, while if the Chinese are repatriated, a serious set-back will certainly take place.

Rhodesia. (By W. Fischer Wilkinson.)—The gold production in 1906 was 553,985 oz. bullion, equal to 479,089 oz. fine gold, or $9,902,570. The production, in bullion ounces, since the commencement is given in the accompanying table.

GOLD PRODUCTION OF RHODESIA.

Year.	Oz. crude.	Year.	Oz. crude.	Year.	Oz. crude.
To Sept. 1, 1898	6,471	1901	172,062	1905	409,836
Sept. 1–Dec. 31, 1898	18,085	1902	194,169	1906	553,985
1899	65,304	1903	231,872		
1900	91,940	1904	267,737	Total to 1907	2,011,461

Besides gold there are other metallic products of minor importance, the total of which was as follows, up to the end of 1906: Silver 298,354 oz.; copper, 36 tons; lead, 1804 tons; chrome ore, 3647 tons; wolframite, 17 tons; diamonds, 1849 carets.

The most noticeable feature in the gold returns for 1906 is the large number of small producers having batteries of five stamps or under. Whether this source of gold will be permanent seems doubtful. As pointed out in the president's address to the Chamber of Mines in reviewing the work of the financial year ending March 31, 1906, many of the small workers and tributors are working on properties partially developed by companies that have exhausted their working capital. Quartz mining is not generally profitable for the man of small means, and a time must come when development is necessary in order to obtain grist for the mill. The president does not anticipate that the production of gold by the small producer will be maintained at its present figure.

An unfavorable feature of the gold-mining industry of Rhodesia is the lack of dividends. During 1906 only two companies, the Globe & Phœnix and the Giant, declared dividends. The latter is a new company, which commenced crushing in December, 1905, with a mill of 15 stamps and two tube-mills. The value of the output is about £9000 per month, of which about £5000 is profit. This company paid its first dividend of 7.5 per cent. in July. The capital is £250,000. A wide reef is worked and the filling in of stopes is practiced. The Globe & Phœnix mine, which has a 40-stamp mill, continues to make good returns and to pay dividends.

The Killarney-Hibernia mine has a 20-stamp mill and a tube mill. The monthly value of its output is about £5000, but the company has not yet paid a dividend, and the development of the mine in depth has deen disappointing. The Selukwe mine with 40 stamps is one of the principal producing mines, but it has not paid a dividend since 1903. The Wanderer, which is more of a quarry than a mine, crushes the largest number of tons per month of any Rhodesian mine. It has a dry-crush-

ing mill equivalent to 100 ordinary stamps. The value of its monthly output is in the neighborhood of £9000. For the year ending March 31, 1906, it crushed and treated 174,975 tons at a cost of (exclusive of development redemption) 6s. 9.38d. per ton. Nine other Rhodesian mines operating 195 stamps crushed 360,018 tons during the same period at a cost, exclusive of development redemption, of 21s. 5d. per ton. Other mines treating over 4000 tons of ore a month are the East Gwanda with 60 stamps, the Sabiwa with 40 stamps, the Ayrshire with 60 stamps and the Penhalonga with 50 stamps. Attention is now being paid to the mining of low-grade iron-stone and schistose ores as distinct from ordinary quartz reefs.

West Africa. (By W. Fischer Wilkinson.)—The gold production of West Africa in 1906 showed an increase of about 30 per cent. over the previous year, the total being 221,591 oz. of bullion, equal approximately to 199,432 oz. fine gold, or $4,122,260. This includes the Gold Coast Colony and Ashanti.

The gold mines of the Gold Coast Colony may, for convenience, be grouped into three divisions: Mines working the auriferous conglomerates (banket); quartz reefs; and alluvial deposits (dredging).

An analysis of the gold production of 1905 shows that the different districts produced gold in the following proportions: Banket mines, 32.5 per cent.; quartz mines, 61.8 per cent.; dredging companies 5.7 per cent.

The banket mines are situated in the district of Taquah, a village which lies about 40 miles from the coast, and which is connected with the port of Sekondi, having fair facilities for landing cargo, by a railway of 3 ft. 6 in. gage.

The outcrop of the conglomerate beds, which have been extensively worked by the natives in the past, has been traced for a length of about 20 miles in a line more or less parallel with the coast. The whole length of outcrop is portioned off into blocks or concessions, of which four are being worked on a large scale at the present time. The Taquah mine has a 10-stamp mill, crushing development rock, and a new 50-stamp mill in course of erection. The Abbontiakoon, Block 1, is crushing about 5000 tons a month with a dry-crushing plant. The Abosso and Wassau mines have each a 30-stamp mill and cyanide plant.

The old workings of the natives consist of numerous small circular pits sunk close together along the outcrop. Even with this primitive method of mining, a considerable tonnage of ore has been removed. The old workings indicate the position of the best ore, and it is on these old workings that the leading mines are located. Although it is not certain that the absence of old workings is conclusive proof that no payable banket occurs, it is probable that this is the case, or at all events that the reef is in such places of low grade. The results obtained from

surface, adit levels and borehole samples tend to confirm this opinion.

The banket bears a close resemblance to the Transvaal ore, and is a simple ore to treat by the ordinary amalgamation and cyanide processes. The banket reef or reefs (in some cases only one reef is being developed, in others two) vary considerably in thickness, but have on an average about 3 ft. of banket. The greatest depth so far reached by the mine workings is 1000 ft. on the incline, which depth has been attained at the Abosso and Wassau mines, and the continuation is further established by boreholes. The formation inclosing the reef is sandstone, which forms fair walls, so that stoping presents no difficulties.

As to the value of the banket, the records of the crushing mines form the best index. The Abosso mine recovered during 1905 from 31,259 tons 68.6s. per ton. During the first six months of 1906 the recovery was 66.2s. per ton. The Wassau mine in 1904 recovered 65s. per ton, and 48s. per ton milled in 1905; the Abbontiakoon has put through 47,840 tons, yielding about 37s. per short ton. The Taquah mine has been crushing unsorted dump ore for some months, and getting recoveries of 48s. to 68s. per ton.

Working costs are at present considerably above the Rand level, but they are rapidly coming down. The high cost of administration and of white labor is the main factor that makes the coast compare unfavorably with the Rand. For the purposes of comparison, working costs may be regarded as made up of three main items—supplies, native labor and white labor. As regards supplies, there is, with the exception of fuel, no great difference between the prices paid on the West Coast and in South Africa. Coal, which represents about 10 per cent. of the working expense, costs 46s. per ton, as against 14s. on the Rand. But the majority of the mines still burn native wood, which they claim is cheaper than coal.

Of native labor the West Coast has at present a sufficient supply. The average pay of a native, skilled and unskilled, is 1s. $9\frac{3}{4}$d. The average native on the West Coast is not so efficient as the average native in South Africa, but on the West Coast the industry has not been so long established, nor is the labor so well organized or supervised. The material is on the whole good, and the West Coast has the advantage of being able to employ the native on skilled work, which is prohibited in South Africa. Many of the natives have been taught trades at the mission schools, and make good mechanics, carpenters, drivers, etc. This is an important point, because the country cannot support, for climatic reasons, a white industrial population as on the Rand.

At the present time the lower costs, including development and London expenses, in the banket district on a 30-stamp basis are in the neighborhood of 40s. per ton. As regards the health of the white employee,

attention paid to providing good accommodation in camps well cleared of jungle and fresh food from the steamers has had a marked effect. Where attention had been paid to these points the health of the camp was good.

The Gold Coast Railway is now in fair working order, but high rates continue to be charged. The railway is handicapped by the heavy cost of construction, the rates being framed to pay interest on the capital expenditure as well as the running and maintenance charges. The policy appears to be to get as much out of the mines now working as possible, and not to look for increased profits from the expansion of the industry that cheaper rates would encourage.

The principal quartz reef mines are the mines of the Ashanti Gold Fields Corporation, the Akrokerri, the Bibiani mine, the Prestea and the Broomassie mines. Quartz mining in the colony promises to be profitable. Up to the present the mining companies have not done well, owing to the lack of railway communications and insufficient development. Now that these difficulties are being removed, the best mines ought, before long, to become dividend paying.

The dredging companies are as yet of small importance. They work on the Ankobra and Offin rivers. For the year 1905 they won gold to the value of £37,292.

GOLD PRODUCTION OF WEST AFRICA.

Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.
1895......	25,416	1897...	23,555	1899...	14,250	1901...	6,088	1903...	70,763	1905...	165,844
1896......	23,940	1898...	17,733	1900...	10,557	1902...	29,880	1904...	94,815	1906...	199,432

Asia.

Outside of Siberia, which is included in Russia, the chief Asiatic gold producers are British India, China and Korea. The production of gold in China is largely an unknown quantity, owing to the entire absence of statistics. Estimates are based chiefly on exports and similar indications. The director of the United States Mint, on this basis, estimated the gold production of China, in 1905, at $1,776,088, equivalent to 85,918 oz. fine gold; but this is very much below the amounts estimated by other authorities, which vary from $4,000,000 to $6,000,000. The former figure appears to be nearer to the facts, so far as they can be ascertained. Some further remarks will be found below.

British India.—The production in the Kolar goldfield of Mysore was 627,700 oz. bullion in 1905, and 576,287 oz. in 1906; a decrease of 51,413 oz., or 8.2 per cent. The bullion reported in 1906 was equal to

518,658 oz. fine gold, or $10,720,661 in value. The decrease in 1906 was due to the fact that the Champion Reef, the most important mine in the district, entered early in the year into a zone of lean ore, reducing its production largely, in comparison with preceding years. Active development work was continued, however, in the hope of recovering the richer orebody at greater depth.

China.—In gold mining attention is directed at present chiefly to Manchuria. The following data on that province are obtained from a United States consular report, based on Japanese information. The northern part of Manchuria abounds in minerals, principally gold, both quartz and placer, silver, copper, lead and iron. Thus far the average annual output has amounted to $10,000,000, some of the mines being operated by the Chinese officials, but most of them by native companies and small capitalists. A large part of the deposits has not yet been touched.

The territory drained by the Huifaho and to the north of that river is rich in gold, both quartz and placer, silver, lead, copper and coal. This is true also of the district to the east of the Tunghuachiang (Sungari river). This mineral wealth is made accessible by the Yalu, Tunghuachiang (Sungari), Nenchiang, Hailungchiang and Liao rivers and their tributaries. The construction of the Changchun-Kirin Railway will also greatly facilitate the development. The operation of the mine at Tienpaosan was about to be undertaken by Americans, but the project was abandoned on account of the Russo-Japanese war. The viceroy of Kirin then canceled their claim and the mine is now being operated by the bureau of mines of Hungchung with Chinese capital. The same is true of the gold mine at Chiapikou, which was originally operated by the Russian government, 20,000 coolies being employed, but is now in the hands of the Chinese.

Korea.—The greater part of the 1906 production of gold was made by the Oriental Consolidated Mining Company, a New York concern, which is operating mines in the Wunsan district. Its latest report deals with the operations in the year ended June 30, 1906. Although the Russo-Japanese war ended in October, 1905, a large amount of trouble was experienced at the mines through scarcity of competent labor and lack of transportation facilities. This resulted in a reduction in tonnage crushed in the mills of 21,230 tons as compared with the previous fiscal year. The total amount of ore mined was 233,157 tons valued at $1,177,487, or an average of $5 per ton. In addition to the above the mines operated on tribute, produced 3260 tons of ore valued at $20,150. The average mining cost, including all underground development work, was $1.14 per ton. The operating profit was $367,882. Milling costs varied from 0.45c. to 0.80c. per ton.

Australasia.

The total gold production of Australasia—including the Commonwealth of Australia and New Zealand—in 1906 was 3,978,579 oz.; a decrease of 177,615 oz., or 4.3 per cent., as compared with 1905. New Zealand showed a substantial gain, but in the Commonwealth of Australia, the only State showing an increase was Victoria. Part of the loss was due to the division of labor from gold mining to other mineral industries, chiefly copper and silver.

Eastern States. (By F. S. Mance.)—The gold yield for 1906 does not come up to that for the previous year, the decrease amounting to 177,615 oz. Particulars are given in the accompanying table of the gold production of Australasia in ounces fine, for a series of years.

PRODUCTION OF GOLD IN AUSTRALASIA.
(In ounces fine.)

State.	1901	1902	1903	1904	1905	1906
Western Australia	1,669,072	1,819,308	2,064,801	1,983,230	1,955,316	1,794,547
Eastern States:						
Victoria	730,449	720,866	767,351	765,596	747,166	772,290
Queensland	598,382	640,463	668,546	639,151	592,620	544,636
New South Wales	216,888	254,435	254,260	269,817	274,267	253,987
Tasmania	69,491	70,996	59,891	65,821	73,541	60,203
South Australia	21,946	24,082	21,195	29,177	(a)20,330	(a)18,300
Total Commonwealth	3,306,228	3,530,150	3,836,044	3,752,792	3,663,240	3,443,963
New Zealand	412,875	458,933	479,715	467,898	492,954	534,616
Total Australasia	3,719,103	3,989,083	4,315,759	4,220,690	4,156,194	3,978,579

(a) Northern Territory.

Referring only to the eastern States, Victoria ranks as the chief contributor. The Bendigo field is the staple support of the industry, and has maintained an output averaging 19,800 oz. per month. The principal dividend-paying mines were the New Argus, South New Moon, and New Moon. The disclosure of a gold-bearing formation in the Victoria quartz mine, at a depth of 4154 ft., has given unmistakable proof of the permanence of the reefs on this field. The output from the Ballarat field continues to show a falling off, but the quartz mines at the other centers have yielded well. In connection with alluvial mining, the dredges at Castlemaine, Fryer's creek, Buckland and Bright contributed largely toward the total yield, as did also the mines working the deep levels at Chiltern and Rutherglen. Toward the close of the year a rich find of alluvial gold was made in the vicinity of Tarnagulla and nuggets weighing 960 oz., 675 oz., 502 oz., 387 oz., 373 oz., with many of less weight, were unearthed in shallow ground. The work of draining the Moolort deep lead was unremittently carried on, and it is anticipated that it will shortly be possible to rise into the gutter at the Loddon

Valley mine, at which mine the pumping operations have been concentrated for some time past.

The lessened output of Queensland may be accounted for in a great measure by the closer attention which has been given to the mining for the baser metals. The Charters Towers field is the chief center of gold-mining operations in this State, but the results compare unfavorably with those secured in previous years, and a further drop in the average grade of the quartz won is noticeable. Calls have been readily paid by investors, and thus a considerable amount of development work was carried out during 1906. Altogether the outlook seems to warrant the assumption that the output for 1907 is not likely to recede below the present level. The Mount Morgan mine, which has paid dividends aggregating £6,680,000 as a gold mine, entered on a new era during the last half-year as essentially a large copper producer.

New South Wales is unfortunate in having to record a decreased production, as the returns for the first 10 months of the year showed well ahead of those for the same period in 1905, but during the last two months the gold was not sent forward to the mint. The official figures accordingly disclose a decrease, but they do not accurately reflect the position of the industry, as it has been ascertained that the principal gold-buyer held gold to the value of £160,118 on Dec. 31, instead of depositing it with the mint. This gold has therefore to be carried over to 1907. The Mount Boppy stands out preeminently as the gold mine of this state, and the developments underground during 1906 were of such a satisfactory nature as to indicate that the output during 1907 will be still further augmented. On the Wyalong field rich sulphide ore is being obtained at a depth of 1000 ft., and the prospects are favorable to the maintenance of a large output. The yield from the Hillgrove field came well up to the average, although the operations of the New Hillgrove company were considerably hampered owing to explosions of rock carrying away the shaft in the lower levels. The various dredges throughout the State again won a considerable quantity of gold, and remunerative returns were secured by most of the plants, but the yields as a whole were not as satisfactory as in preceding years.

Tasmania has unhappily to record an output below that of 1905. This is due to the fact that just as the principal gold mine, the Tasmania, was on the eve of reaping the benefit of the expensive plant which had been installed, the underground workings were flooded by an inrush of storm waters, which caused the suspension of mining operations for some months. To the end of June, 1906, the quartz crushed from this mine totaled 613,924 tons and yielded 655,408 oz. gold.

The gold yield of South Australia is an inconsiderable item, the only

producing mine of any importance being the Tarcoola Blocks, which won 4922 oz. during the year ended June 30, 1906.

The export of gold from Australia during 1906 was very heavy, and was £5,591,000 in excess of 1905. The accompanying statement shows the value of the gold shipments from the respective ports.

GOLD EXPORTS FROM AUSTRALIA.

	1905	1906
Melbourne	£1,018,805	£3,187,457
Sydney	2,144,237	5,469,149
Brisbane	4,800	....
Adelaide	16,250	180,000
Fremantle	6,232,884	6,071,551
Total	£9,416,976	£14,908,157

The dominant feature of the silver mining industry was the exceptional interest evinced in the Broken Hill field, and at the close of the year the market value of the shares of this group of mines had reached over £11,-500,000. The confidence shown by investors appears to be fully justified, as it has been estimated that, with the complete plants now installed at the principal mines, operations can be remuneratively carried on with lead at £11 10s. a ton, and it is at once apparent that, with the sustained price of that metal, the way has been laid for the reaping of enormous profits. In the first half of 1906 production was curtailed by several causes, notably the fires in the Proprietary and Junction mines, and creeps at the Central and Block 10 mines. The total output is on this account below that of the previous year, but the value is considerably greater. In the last quarter of 1906 all the disabilities mentioned had been overcome, and the field was the scene of the greatest activity. Work underground has further added to the ore reserves, and, as bearing on this point, it may be mentioned that the lode in the Broken Hill South mine was opened up at the 975-ft. level during the year, and was proved to be over 390 ft. wide, with an average assay over the whole width of 13.1 per cent. lead, 16.1 per cent. zinc and 10.6 oz. silver per ton.

At Howell, N. S. W., the company which took over the Conrad and King Conrad mines, in 1905, obtained gratifying returns. The Yerranderie field, N. S. W., contributed large supplies of silver-lead ores, as did also the C. S. A. mine Cobar.

The total production of silver by the silver-lead mines of New South Wales was 5,575,410 oz. in 1906, against 6,804,934 in 1905; 7,751,667 in 1904; and 6,489,689 in 1903. In addition thereto, the ore exported in 1906 contained 3,111,013 oz., against 3,480,561 in 1905; 2,945,058 in 1904; and 1,736,512 in 1904.

The value of the production of silver-lead in Tasmania for the year 1906 is £462,443 which is an increase of £215,555 on the value of the output for 1905. The mines in the Zeehan and Dundas districts, on the West Coast, are in a thriving condition, and the indications are favorable to a sustained output.

In Queensland the bulk of the output was contributed by the mines at Mungana. The permanence and extent of the sulphide lodes are regarded as so assuring that a complete Huntington-Heberlein plant, crushing appliances, roasters, etc., were installed during the year, and have given gratifying results. The production of silver-lead from the mines of this State is valued at £151,577, and shows an increase of £49,-189 over that for the preceding year.

Western Australia.—The decrease in gold output in 1906 was largely due to the lower tenor of ores worked by the large companies in the Kalgoorlie district, which has been apparent for two or three years past. In that and other districts there were no discoveries of importance, and no new mines opened of sufficiently large production to make up for the decline. Nevertheless, the gold output remains large, and signs of a gradual recovery are not wanting. This State furnished 45 per cent. of the gold won in Australasia during the year, and so maintained its pre-eminence as a producer. Some further progress was made in working out the problems which are presented in the treatment of the ores of West Australian mines, and the industry, on the whole, closed the year in good condition.

Europe.

Russia.—The only European production of importance is from Russia, which includes the Asiatic provinces of Siberia. Those provinces, together with the Ural, furnish the great bulk of the gold output, and keep the country high in the list of gold producers.

The return of the Russian Imperial Mint for 1906, as furnished by our St. Petersburg correspondent, shows a total of 18,359 kg. of gold actually received and refined, with 12,379 kg. additional in transit, or at branches. This makes a total for the year of 30,738 kg., or 988,227 oz., having a value of $20,426,938. Theoretically, all of the gold mined in the country is turned over by the producers to the Imperial Mint; practically, it has been usual to make an allowance of 10 per cent. for gold not so transferred and reported, but disposed of in some other way. Some engineers hold that this allowance is not sufficient and that 15, or even 20 per cent. would be nearer the truth. Accepting the 10 per cent. as conservative, however, and making that addition to the official returns, gives a total production for 1906 of $22,469,632. This is a slight increase over 1905, which had hardly been expected. It must be remembered, however, that the districts in which gold mining is chiefly carried on are remote

and have been affected but little by the political and social troubles which disturbed the country during the year.

The silver production of Russia, as reported by the Imperial Mint for 1906, was 3740 kg., or 120,241 oz.; a reduction of about 30 per cent. from the previous year.

North America.

Canada.—The total production of gold in 1906 is reported by the Dominion Geological Survey at $12,023,932, of which $5,000,000 was from the Yukon, and $6,423,932 from other Provinces. The silver production for 1906 was 8,568,685 oz., an increase of 43 per cent. over 1905, due to the large production from the Cobalt area, which has put Ontario in the first place as a silver producer.

GOLD AND SILVER PRODUCTION OF BRITISH COLUMBIA.

	1903		1904		1905		1906	
	Oz.	Value. (a)	Oz.	Value. (a)	Oz.	Value. (a)	Oz.	Value.
Gold, placer....	53,021	$1,060,420	55,765	$1,115,300	48,465	$969,300	46,000	$920,000
Gold, lode......	232,831	4,812,616	222,042	4,589,608	238,660	4,933,102	250,000	5,167,500
Total gold....	285,852	$5,873,036	277,807	$5,704,908	287,125	$5,902,402	296,000	$6,087,500
Silver..........	2,996,204	1,521,472	3,222,481	1,719,516	3,439,417	1,971,818	3,470,000	2,201,675

(a) Placer gold is valued at $20 per oz.; lode gold at $20.67 per oz.; silver at average market quotations.

British Columbia. (By E. Jacobs.)—The further falling off in placer gold—1905 production having previously been the lowest since 1898—is due to causes which are matters for congratulation rather than the reverse. In both the Cariboo and Atlin districts—the two chief placer-gold sections of the Province—financially strong companies are engaged in preparations for extensive operations next year or the following. The Guggenheim enterprises in both districts are on such a scale as should add largely to the production. In the Atlin individual miners are rapidly giving place to companies, and steam shovels are superseding smaller appliances where large quantities of gravel are to be handled. The increase of nearly a quarter of a million tons of ore treated in the Boundary added to the lode-gold output, there being a little gold associated with the copper ore. Rossland's production has well kept up. The Ymir mine has again been a disappointment. The Nickel Plate, in Lower Similkameen, was not worked to full capacity of its mill, pending the outcome of negotiations for sale of the property, which was not made.

The St. Eugene increased its output of lead-silver ore and consequently its silver production, and the Sullivan mine, also in East Kootenay,

added its quota of silver from its low-grade ore. Boundary and Rossland helped a little, but the Slocan output was again much below expectations. The closing of the silver-lead smelters for several weeks, during the strike at the Crow's Nest coal mines was in part responsible for the decrease in this metal.

The St. Eugene again maintained its very considerable advantage in lead production, and the Sullivan was also an important producer. On the other hand, the Slocan fell short. The Blue Bell mine in Ainsworth division will likely produce a considerable quantity of lead when arrangements for treating its big body of lead-zinc ore shall have been completed.

Ontario.—The production of silver in this province is treated under the head "Nickel and Cobalt" elsewhere in this volume, to which reference should be made.

Quebec.—Actual production of precious metals continued small; but explorations in the Abittibi and Chibogamoo districts, adjoining the Ontario line, promise well for the future, and arrangements were made to exploit those regions.

Yukon Territory. (By J. P. Hutchins.)—This region is still in a state of transition, and the gold production in 1906 again showed a decrease. This is not surprising, for much ground which would have been worked under normal conditions has been acquired and held for future working when a large corporation and its subsidiary companies shall have completed purchases and installations of machinery for replacing present methods. The year was one of comparative idleness on the part of many claim owners who expected to be bought out.

Although the snowfall was heavy the thaw was so rapid that the water could not be used so as to produce the best results. There were few winter lays or leases given out, the region being in a buying and selling ferment. A heavy freshet caused considerable damage; one dredge was partly submerged, and a number of cabins and flumes were washed away. May and parts of August and September had rains, while June and July were very dry. The season was in the main unfavorable for working.

There was a shortage of labor during the summer when the preparations for future working on a large scale were at their hight. There are no unions in the Klondike. Labor is efficient, seemingly as a result of the remarkably stimulating climate. Wages are still high, 40c. per hour and board in summer; 30c. per hour and board in winter.

The gold production of the Canadian Yukon territory, which includes the Klondike and several other districts, is given in the accompanying table. This is all placer gold, for there are no gold vein mines in Yukon territory.

GOLD PRODUCTION OF THE CANADIAN YUKON.

Year.	Amount.	Year.	Amount.	Year.	Amount.	Year.	Amount.
1896......	$300,000	1899.......	$16,000,000	1902.......	$14,500,000	1905.......	$7,000,000
1897......	2,500,000	1900.......	22,250,000	1903.......	12,250,000	1906.......	5,000,000
1898......	10,000,000	1901.......	13,000,000	1904.......	10,350,000		

No discoveries of new deposits or of rich peneplains on the rims of the channels already worked were made in 1906. There is the likelihood of the occurrence of such deposits as secondary benches or terraces on the "back rims" of the "white channel." This white channel is what is left of the ancient creek beds; it has a course approximately parallel to, and an elevation of 150 to 300 ft. above the present creeks. Where it has not been eroded it appears as a bench deposit with one rim completely removed. There is one notable exception to this occurrence at Lovett gulch, off Bonanza creek, near its confluence with the Klondike river. There are remains of the White channel or of contemporaneous gravels, on Adams gulch, off Bonanza creek, and on Last Chance off Hunker creek; these deposits contain rich gravel, and have yielded well where cheap water is available for hydraulic mining. There is a promising future for these areas.

Although there were no discoveries of importance, there were many stampedes to known areas which were suddenly considered suitable for exploitation by the dredging method. This was due principally to the success of a 7½-cu.ft. bucket dredge, which operated about 60 days in 1905, and in the open season of 1906 with most gratifying results. The conditions encountered were unusually favorable for Klondike, for there was but little frozen ground to excavate and an easy bedrock and free gravel combined to make the enterprise a successful one. It was due to the success of this one installation that the dredging frenzy began; there have been other successes since. The circumstance that the dredge was installed in an area possessing exceptionally fortunate characteristics, not found generally in Klondike, was not kept in mind. There was much excitement and many abandoned claims were restaked for dredging. There is talk of installing about 30 dredges in 1907, but only about three are certainties.

The Klondike has always been a district where volatility, if not emotionalism, has prevailed to a considerable degree. A result of this is seen in the large number and variety of machines and methods applied to areas of similar characteristics. These experiments have generally been costly. To be sure some excellent devices like the steam thawer, steam scraper, inclined cableway and self-dumper have been evolved. It can be predicted with certainty that if floating dredges are installed

on some of the creeks, failure must occur even though a gold tenure of more than 50c. per cu. yd. shall be found. Frozen muck, a mixture of semi-decomposed vegetable matter, silt, sand, and water in varying proportions, usually more than 75 per cent., as an overburden, and blocky schist bedrock are the most difficult physical conditions, and a short season, high wages, costly fuel and supplies are adverse economic conditions.

No doubt there are other areas like the one proved by drilling during 1906, where a minimum of about 40c. and a maximum of over $2 per cu. yd. is said to have been found, which may be worked with success; but the promiscuous placing of dredges in Klondike will mean numerous failures, even though a gold content very much greater than that encountered at Oroville shall be proved. This prospecting has shown that there are wide pay streaks. No discoveries showing other than great vertical concentration have been made.

Mining methods did not change materially during 1906. The year was, however, one of transition. When the reservoirs to conserve local water, and the long ditches, are completed, the water will be used in generating power for operating floating dredges and electric machinery, lifts, pumps, etc., in conjunction with modified hydraulic mining machinery and methods not only to excavate and hoist material to the sluices but also to dispose of tailings. These operations, covering large areas of creek-bottom claims, make a merger of considerable magnitude and will depart in some cases rather radically from existing methods and the established ideas of the most effective means of working.

There are many reasons why these creeks are unusually amenable to working as one-man propositions. The total gravel section is usually shallow and solidly frozen and the overburden must be stripped in successive layers as it thaws by exposure. This may best be done by ground sluicing or with steam scrapers, as the material is then removed as it thaws and no attempt is made to handle frozen ground. Ground sluicing is the cheaper when ample water is available, but water will always be comparatively scarce and costly in Klondike.

Now, after most of the creek bottoms have been partially or thoroughly worked, in many cases more than once, and the original superposition of muck has been disturbed, the opportunities for ground sluicing are not so favorable. Steam scraping has been the favorite means of stripping and only in exceptional cases has any other method for removing overburden been used.

The means of excavating and transporting pay material to the sluices, after the overburden has been removed, are varied. The following are those employed: Shoveling to platforms, then to sluice; shoveling to wheelbarrows, wheeling to bucket, raising on inclined cableway

to sluice; shoveling into cars, hauling on inclined track to sluice; steam-shoveling into cars, hauling on inclined track to sluice; steam-shoveling directly into sluice; shoveling into skips, skidding and hoisting by derrick into sluice; hydraulicking to a sump, hoisting with steel bucket conveyer to sluice; hydraulicking into a sump, hoisting by centrifugal dredging pump into sluice; steam-shoveling into skip, hauling on cableway to sluice; and several other combinations. In a few cases a steam elevator was used to raise the material to the sluice. The means most generally used is shoveling into wheelbarrows, wheeling to bucket, hoisting on inclined cableway to sluice. It is well adapted to the one-man enterprise.

The methods and machines in contemplation will use many of the old features, but will include better means for tailing disposal and will have other advantages. Machinery will replace hand labor and larger capacities will be attained. Areas which have been worked on the one-man plan will be worked on a large scale. In the early days many of the creek claims were grouped and exploited under corporate ownership and management. There were failures in nearly every case, although in several noteworthy instances very rich ground was mined. However, bad management contributed largely to this result. Klondike operations will be watched with more than usual interest after the contemplated changes have been in force for a time.

Two new dredges began operations in 1906 and there were three in course of construction; much of the ground to be mined is frozen, and will need steam thawing, at a cost of 25 to 50c. per cu. yd. One dredge operating in ground but little frozen has excavated material yielding 50c. to $1.00 per cu. yd. at an operating cost of about 12c. per cu. yd.; a daily average capacity of over 3000 cu. yd. was attained. It is interesting to note that the daily average capacity of a dredge at Oroville, Cal., the counterpart of this one, for a year, was about 2900 cu. yd., at a cost of less than 5c. per cu. yd.

Dredging in the Klondike river valley, near the mouth of Bonanza creek, is said to have been profitable. From operations conducted during 1906 it is thought that, in normal seasons, dredging can be begun about May 1 and continue until about Oct. 20, although operations in the early and late parts of the season must necessarily be at higher cost and with greater difficulty. The design and construction of the dredges must be suited to operations in cold weather. A well designed dredge should be able to operate 30 to 45 days longer in a season than possible in hydraulic mining, and 45 to 60 days longer than open-cut mining may be carried on.

By hydraulic mining between 2,500,000 and 3,000,000 cu. yd. was washed, about twice as much as in 1905, in spite of a poor water

season. This was due to more available sources of supply and to the construction of storage reservoirs. There were no essential modifications in method and for cost, including amortization, about 20c. per cu. yd. is proper for well managed operation. A dam, 54 ft. high, on Adams creek, and forming a reservoir containing 58,000,000 gal., was completed in 1906. Another, to be 60 ft. high, and to conserve 2,000,000,000 gal., was begun on Bonanza creek.

Two ditches of considerable size, with capacities of 10,000 miners' inches and 500 miners' inches respectively, were partly built. About 5 cu.yd. per miners' inch was hydraulicked per day in mines operating on a scale employing 5 to 7 cu. ft. per sec. Owing to conflicting grants of water rights, there was considerable difficulty, much of which was aired in the courts. There were fewer applications for water rights, hydraulicking, as a means of profitable working, having been superseded in popular esteem by dredging. Hydraulicking with pumped water was carried on in but one notable instance. This method is extremely costly and seldom profitable.

The physiography of the Klondike is extremely unfavorable to getting ample supplies of water without long ditches, pipes flumes and inverted siphons. Topographic isolation is the cause. Climatic conditions are unfavorable to the construction and maintenance of water ways. In addition to the high cost of labor, materials and transportations, good ditching ground is seldom encountered and maintenance expense is large.

Attempts have been made to use steam shovels as ditching machines in gravel, loose rock and muck. Where the material was thawed, good progress was made, but as this was the exception rather than the rule, much difficulty was experienced. Wooden-stove pipe is being used instead of open flumes for the first time in Klondike. Horse scraping has been used rather generally of late in ditch construction. A cost of 30 to 50c. per cu. yd. was noted for such work.

The fuel problem was still a difficult one. Wood hauled on runners cost $10 to $16 per cord.; for that delivered on wheels, $14 to $20 per cord was paid for 16-ft. lengths. Wood tends to become more costly each year as it is necessary to haul greater distances. Coal from Coal creek, Tantalus and Five Fingers, was used in a limited way, but not with encouraging results. It possesses low calorific value, and is only used where its cost per ton is less than that of wood per cord.

The transportation problem is still a difficult one. Although the Canadian Government is pursuing its excellent policy of building and maintaining roads, cost of transportation is high. New roads were built in Stewart river district and elsewhere. A narrow-gage steam railway is being built out of Dawson to Stewart river. Freight rates

from the outside are still high; about $60 per ton is the charge. Passenger rates for railway, river steamer and stage travel are about as formerly: 20c., 11c., and 25c., per mile being the respective average charges.

Considerably more drill prospecting was done in 1906 than in former years. The frozen condition of the gravel makes no essential differences in the manipulation. Prospecting value and dredging value, as determined by drilling with percussion drills and dredging with buckets, respectively, check well.

A new mining code has been adopted and numerous betterments will result. The regulation, passed in 1904, to permit the sale of water, has had no effect for there has been no surplus water to sell. No new concessions were granted, and there is little likelihood of any being given in the future.

Mexico. (By C. A. Bohn.)—Mexico is still the leading nation in the production of silver and easily stands fourth among the world's gold producers. Much of the recent progress has been due to the application of cyanide process to the copper-free silicious ores, very few of which are not amenable to this treatment. In the introduction of this process great credit is due the Mexican Gold and Silver Recovery Company, which for 12 years has held the rights in Mexico to the McArthur-Forrest cyanide patents. This company's experts have demonstrated the process in Sonora, Sinaloa, Chihuahua, Durango, Zacatecas, Guanajuato, Michoacan, and all over the Republic. What promised to be a long and expensive litigation over the right to hold a patent on this process was instituted early in February by the manager of the Recovery Company, against the Guanajuato Reduction and Mines Company, but the controversy was amicably adjusted by the payment of a nominal sum for the perpetual right to use the process.

Guanajuato.—Of the silver-producing districts, Guanajuato is in the lead. Only a few properties were working in a small way at the time the Guanajuato Consolidated Mining and Milling Company entered the district and took hold of the Sirena mine seven years ago. After heavy expenditures in development and equipment, Jan. 1, 1905, saw this company with 90 stamps dropping. Including the treatment done in the patios, 300 tons were being treated daily in Guanajuato, which would have given a total valuation of $2,000,000 for the year's product. The opening of 1906 saw this output doubled; the pay rolls had increased from $80,000 to $200,000 per month, 200 stamps were dropping, treating 600 tons daily and giving a production at the rate of $4,000,000 Mexican per year. The cyaniding of low-grade ores and the old dumps is no longer an experiment; the profits on $30 ore have been raised to about $22, where formerly they were less than $3, and it has needed only cap-

ital to make Guanajuato one of the leading silver-producing camps of the world. This has been done principally with American capital.

The Guanajuato Reduction and Mines Company operated all the old producing mines of the mother vein of Guanajuato, except the Sirena. Work was started upon an 80-stamp mill and cyanide plant in March, 1905, and was completed in March, 1906, since which time the whole mill has run something over 98 per cent. of its possible stamp hours, upon ores about one-half of which have been old waste and one-half mine ore. In the meanwhile, development work and the unwatering of old workings have been carried on in the mines. The working of the dumps has been so much more profitable than the original estimates that contracts have been let for a larger mill, to be in operation by August, 1907. The stamp mill and concentrating units are located at the mines, the tailings being conveyed more than a mile through an 8-in. cast-iron pipe, laid on a 2¼ per cent. grade, to the Flores hacienda, in Guanajuato, where they are cyanided. In addition to the mine development satisfactory progress has been made in opening up the immense stope fillings in the old mines, which exist in widths up to 50 ft., the average grade of which is about $10 (U. S. currency) per ton.

With the other companies—Guanajuato Consolidated, Peregrinas, Guanajuato Amalgamated, Central, Nayal, El Cubo, Carmen, and Pinguico—practically all is new work, but the metallurgical problem is the same, and all are cyaniding, so that combined there are 650 stamps now dropping in Guanajuato, treating 2000 tons a day, or at the rate of 14,000,000 *pesos* a year. The camp is to be greatly aided by the extension of the Mexican Central Railroad from Marfil into Guanajuato, and the building of a line to Salamanca to connect with the Mexican National Railroad.

Pachuca.—Next to Guanajuato, of the old camps, comes Pachuca; but unlike Guanajuato it had not particularly declined, one of its old plants alone, the Hacienda de Guadalupe, of the Santa Gertrudis & Guadalupe Mining Company, treating 150 tons a day by the old process of Bartolomé de Medina. In February, 1906, the United States Smelting, Refining and Mining Company, of New York, purchased through the MacDonald Brothers, of Guanajuato, the famous old Real del Monte mines, and since that time the property has been practically closed down while a full electrical equipment is being installed, and a 250-ton cyanide plant is being erected. It is understood also that the Blanca y Anexas, another of Pachuca's largest properties, has been sold.

Zacatecas.—This famous old silver camp also seems to stand a fair chance of being revived. The old San Rafael group was taken up by El Grande Mining Company, of A. E. Stillwell and associates, and active mining operations were resumed about the middle of 1906. The property

is being slowly unwatered, and results so far, especially from the work in virgin ground, have been most encouraging. El Bote, has been under examination by an American company. The old Tajos de Panuco is being unwatered by New York capital, as is also the San Roberto, adjoining the Mala Noche, while it is probable the Veta Grande will also be unwatered to admit of examination.

San Luis Potosi.—The success of the cyanide process has been proved for some years in the treatment of the old dumps in the San Pedro district near San Luis Potosi, though the greater part of the ores still go to the smelter of the Mexican Metallurgical Company, at the latter point. This company opened up during 1906 large bodies of silver, lead and iron ores in the properties it holds under lease from the Victoria company in the San Pedro district.

Chihuahua.—In Ocampo, or Jesus Maria, the seat of the Rayon district in western Chihuahua, beyond Miñaca, the success of the Concheño Mining and Milling Company, with its 60 stamps and cyanide plant, has convinced the skeptical, and 1906 saw the extension of cyanide practice to the other companies. At the Dolores mine, May, 1906, saw the successful institution of cyanide after amalgamation. In Parral, the initiative has been taken by the Veta Colorado in the starting of a 250-ton cyanide plant, which will be the first in the district.

Jalisco.—In the Etzatlan district of this state great activity was shown during 1906. The Amparo Mining Company, of Philadelphia, owning the old Santo Domingo mine among others south of Etzatlan, has gradually brought up its production to 100 tons a day, and with a 20-stamp experimental mill has proved the practicability of the cyanide process for its ores, so that orders have been given for a 200-ton plant for the treatment of the low-grade ore. In the old properties of Mololoa and Tamara in Hostotipaquilla, a district north of Etzatlan, rich bodies of ore have been disclosed, and the last months of 1906 saw a large combination effected between these properties and those of Amajac. As yet the patio is in use in this district for the ores that are not shipped. At El Favor mines, controlled by Makeever Bros., of Boston, what is believed to be the Mololoa vein was encountered in October, and a large body of ore is being blocked out.

El Oro.—The mining camp of El Oro extends from the town of El Oro, in the State of Mexico, over into the State of Michoacan, and includes Tlalpujahua. Into this district electric power was introduced by the Mexico Light and Power Company, from its power plant at Necaxa over a distance of 200 miles, and with El Oro Mining and Railway Company the replacement of steam by electricity was begun in February and applied throughout the property. The operation of the electric plants has resulted in a reduction in the cost of power from \$150 or \$200 per horse-

power-year down to about \$50, and a lowering of the milling cost from about \$7 to \$5. This company reported in June, 1906, for the preceding 12 months' work, a total production of \$234,079 tons, from which were realized \$2,225,000 gold at a total cost of \$1,257,000. The Dos Estrellas Company has kept up its regular monthly dividend of \$40, Mexican, per share, or a total of \$120,000 per month; this should be materially increased after the completion of the new mill, with 120 stamps and three 24-ft. tube-mills, capable of treating 600 tons daily. The Esperanzas Mining Company has continued paying its dividends of \$400,000, gold, per month. This company has resorted to the novel expedient of further leaching the old dumps that have been acted on by weather.

The combined production of the three El Oro properties, amounted to a little over \$12,000,000, Mexican, for the year, of which 80 per cent. was gold. The reduced cost of extraction, owing to the introduction of electricity, and the development of the smaller properties will greatly increase this output in 1907, present year. At the properties of the Mexican Mines of El Oro, Ltd., several thousand tons have been opened up, and a plant of 20 stamps, with tube-mills and cyanide tanks for 100 tons daily capacity, is under erection. The Chihuahua, Aldebaran, Victoria and a number of smaller properties are actively engaged in development.

Other Districts.—In Sonora rich silver-lead ore has been opened by the Cubana Consolidated Copper Company, at Arizpe, 40 miles south of Cananea; at the Pichaco mines, southeast of Cananea, bought from the Phelps-Dodge Company by Clancy Bros., of Detroit, bodies of silver and gold ores are being developed and are treated at the 200-ton mill just completed on the Oro Maximo Mining Company property, under the same ownership. Chas. Butters has started the cyaniding of his low-grade ore at the Grand Central, while the Boss process, followed by cyanide, is used at the Creston Colorado, both of which are at La Colorado, Sonora. At the Llanos de Oro placers, Epes Randolph is putting in a 100-stamp mill, while the Greene Gold Company is installing dredges of 2000 tons daily capacity for operation on La Brisca placers. In Chihuahua the Lluvia de Oro is putting in a 100-ton plant with cyanide after amalgamation, while the Palmarejo Mexican Gold Fields, Ltd., has been put on a profitable basis by increasing to 100 tons a day and cyaniding after concentration.

Miscellaneous.—In connection with gold and silver production the most important new work of the year was the putting in operation of the electrolytic refinery of the National Metal Company, in Mexico City, from which the first shipment of fine silver and fine gold was made in January, 1906. The melting department of this plant consists of three doré furnaces, one reverberatory furnace and three tipping-retort furnaces; the parting and refining departments are made up of 10 sections

of four cells each for the electrolytic refining of silver by the Moebins process, and one section of six cells for the electrolytic refining of gold. The refinery is receiving base bullion and cyanide and sulphide precipitates from almost every State in the Republic, and among these receipts is practically the entire output of the Guanajuato, El Oro and Pachuca districts. The business has grown from a small beginning up to where now the monthly production amounts to 500,000 oz. of fine silver and 25,000 oz. of fine gold, which is about two-thirds of the actual capacity of the plant, and as the output is continually increasing there is no doubt that additions will have to be made to the plant during 1907. Of the product, which, in both silver and gold, is 999.8 fine, all the gold is sold to the Monetary and Exchange Commission of the Federal government, while the silver, with the exception of a small proportion disposed of locally, goes to London.

Railroad building was carried on at a pace as never before since the completion of the Mexican Central and National systems. The Cananea, Yaqui River & Pacific is running its extension to Guaymas, Sonora; the Kansas City, Mexico & Orient is driving across Chihuahua and Sinaloa to Topolobampo, the Mexican Central is completing its branch from Guadalajara to the Pacific coast port of Mazanillo; and the Southern Pacific is stretching out almost across the continent, from Guaymas, Sonora, through that State, and Sinaloa, Tepic and Jalisco, to Guadalajara, as fast as possible. But most important is the control obtained by the Federal government over the Mexican Central, which, taken with the Mexican National, of which the Government secured control in 1903, will form a combination of 10,000 miles, and for the operation of which a new $225,000,000 (gold) company will be formed, the Government holding $113,000,000 of the stock.

South America. (By J. P. Hutchins.)—The record of South America for 1906 is not one of either general success or hopeful promise for better results. It is not difficult to account for the circumstance that, with its physiographic advantages, South America still possesses great areas which have never been investigated even in the most cursory manner. There are three particular reasons, namely: a trying climate, poor transportation facilities and unstable governments. At low elevations in the tropics these difficulties are at their worst. Good transportation and stable government are encountered in the tropics only in countries whose flagging energies are constantly stimulated by the arrival of newcomers from temperate zones.

The countries lying in the south temperate zone are, largely because of more favorable climate, ahead of their neighbors in the tropics in matters of transportation and government. There are many difficulties of transportation which are common to all extremely rough and mountainous

regions, and all schemes for the building of railways in any of the countries, including either the main Cordilleras or their many ramifications, must encounter engineering problems of great magnitude and corresponding high cost of installation.

It is interesting to note the statistics of the total gold productions of some of the countries of South America. Colombia is credited with $895,000,000; Brazil, $720,000,000; Bolivia, $199,000,000; Peru, $119,000,000; and Chile, $33,000,000. This makes a grand total of nearly $2,000,000,000, about one-fifth of the total world's production. These figures are significant and suggest possibilities for further development and lucrative exploitation with modern economic methods.

In South America, as in all partially developed regions, the more important mines are the placers, and the major part of its gold product is from alluvial deposits. There were no developments during 1906 of extensive, rich auriferous deposits, and no noteworthy new exploitation. There were a number of installations of hydraulic mining machinery, with varying success. Several dredges were built, and dismal failure is to be noted in most cases; not because the ground was poor, but by reason of faulty and unsuitable design and construction. In Tierra del Fuego successful dredging has been conducted.

Areas of platiniferous gravel, seemingly well suited for exploitation by the hydraulic and dredging methods, were developed in Colombia. The high price of platinum makes these districts particularly attractive at present. An area of diamantiferous and auriferous alluvium was investigated on the Jequitinhonha river in Brazil and a dredge has been shipped to exploit it.

The Commercial Movement of Gold and Silver.

The total gold production of the world in 1905 was $379,635,413, in 1906 it was $410,000,000, an increase of over 10 per cent. That is, after making all deductions, the commercial gold reserve standing back of trade and credit was increased by some $250,000,000, possibly more. This great production of gold was one of the reasons of the activity of general trade, the rush of new enterprise and, generally, of what we call prosperity. In all the great commercial nations, business is conducted chiefly on credit of various kinds; but the basis of all credit, under our modern system, is found in the actual gold coin or bullion which is held, mainly as reserves in the banks and treasuries, and partly in general circulation. The greater this reserve is, the more credit is extended. The danger is that the enlargement of credit for legitimate, industrial and commercial enterprise is apt to be accompanied by a similar, or even greater, extension for purposes purely speculative, which may be carried far enough to endanger the solvency of the whole commercial system.

Twenty years ago Dr. Soetbeer estimated—and some other eminent authorities concurred with him—that not more than 25 per cent. of the world's gold production in any one year, was actually added to the available stock of money. The remainder was absorbed by use in the arts; by the amount actually destroyed in fire and wreck; by gold absorbed in private hoards, and in other minor ways; and finally by the amount needed to make up the actual destruction of values by war, fire and other public calamities. It is possible that even 20 years ago this estimate put the addition to the money stock too low. With the largely increased production of recent years, the improvements in transportation and communication and better methods in business, the proportion at the present day is certainly much higher, reaching very probably 60 per cent. The loss by public disasters is the most important in recent years. The financial derangements caused by the Boer war were hardly adjusted at the opening of 1906. The waste and loss of the Russo-Japanese war still weighed on the world's money markets, and the destruction by earthquake and fire at San Francisco and Valparaiso were serious factors during 1906.

The United States, which led in commercial, industrial and speculative activity, was in 1906 the chief absorber of gold. Adding the production of the country to the net imports there was, apparently, an increase of nearly $300,000,000 in the gold in the country; or of over $200,000,000, if we allow only 60 per cent. of the output to have passed into use. To a great extent, of course, this calculation is hypothetical; but it is evident that this country absorbed the greater part of the new gold supplies of the world. The large imports were provided in various ways. In part they were the result of favorable trade balances; in part sales of securities; in large part they represent actual borrowing of accumulated capital from Europe, while the payments of foreign insurance companies for losses incurred at San Francisco formed an item of some importance.

GOLD IMPORTS AND EXPORTS, UNITED STATES.

	1903	1904	1905	1906
Imports	$65,267,296	$84,803,234	$50,293,405	$155,579,380
Exports	44,346,834	121,138,415	46,794,467	46,709,158
Balance	I $20,920,462	E $36,335,181	I $3,498,938	I $108,870,222

The larger imports in 1906 were from Great Britain, $67,302,216; Canada, $27,926,768; Mexico, $13,312,722; Australia, $11,412,286. The imports from Canada were largely of Yukon gold, and a considerable part was received only in transit, the exports to Canada for the year being given at $14,586,029. That is, $13,340,739 of the gold from the Yukon and elsewhere in Canada remained in this country.

The United States Treasury estimate on Jan. 1, 1907, showed gold held in the Treasury as follows: Working balance, $250,005,397; reserve against legal tender notes, $150,000,000; deposits on which gold certificates had been issued, $652,570,869; total, $1,052,576,266. This is an increase of $136,801,097 over Jan. 1, 1906, and is the largest gold accumulation in the world. The gold in circulation, as estimated by the Treasury, was $695,539,841 on Jan. 1, 1907; an increase of $41,371,816 during 1906. The amount in circulation includes that held by banks in their vaults as reserves. It is probable that the estimate of gold in circulation is too high, for the reason that the Treasury takes account of the increment by import and production, but hardly makes sufficient allowance for losses, such as gold used in the arts, destroyed in various ways, carried out by travelers, and the like. This, however, is opinion only, not susceptible of absolute proof. The increase shown by the Treasury statement corresponds fairly with the known increment by production and import.

Great Britain.—Great Britain has been for many years a taker of gold from abroad, because of its large foreign trade and its enormous investments in foreign enterprises. In 1906 its gold balance was lower than usual, chiefly because of large shipments to the United States.

GOLD IMPORTS AND EXPORTS, GREAT BRITAIN.

	1903	1904	1905	1906
Imports	£28,657,393	£33,876,588	£38,567,895	£46,042,590
Exports	27,766,512	33,039,138	30,829,842	42,617,267
Excess Imports	£890,881	£837,450	£7,738,053	£3,425,323

The increase in imports came chiefly from South Africa and Australia. The exports to the United States rose from £1,817,000 in 1905, to £14,-188,394 in 1906.

France.—This country, like Great Britain, usually absorbs more gold than it gives out.

GOLD IMPORTS AND EXPORTS, FRANCE.

	1903	1904	1905	1906
	Francs	Francs	Francs	Francs
Imports	314,259,000	656,063,000	779,648,000	430,473,000
Exports	129,890,000	123,976,000	131,494,000	165,087,000
Excess Imports	184,369,000	532,087,000	648,154,000	265,386,000

European Bank Reserves.—The gold holdings of the leading European

banks reduced to dollars, in the closing weeks of the year, for four years, are shown in the accompanying table.

GOLD HOLDINGS OF THE LEADING EUROPEAN BANKS.

	1903	1904	1905	1906
Bank of England	$153,536,320	$149,980,465	$142,651,255	$145,322,390
Bank of France	502,500,480	469,307,905	575,671,135	541,150,235
Bank of Germany	145,434,000	154,370,000	166,300,000	137,940,000
Austro-Hungarian Bank	230,700,000	231,165,000	224,325,000	233,045,000
Bank of Russia	382,865,000	426,375,000	576,215,000	589,520,000
Bank of the Netherlands	23,493,000	21,036,000	33,019,500	27,680,000
Belgian National Bank	14,983,335	15,613,335	16,233,335	17,076,665
Bank of Italy	84,345,000	108,520,000	134,345,000	159,440,000
Bank of Spain	71,925,000	72,780,000	75,117,000	76,840,000
Bank of Sweden	....	....	18,900,000	19,780,000
Total	$1,609,778,135	$1,649,147,705	$1,962,775,225	$1,947,794,290

A large increase over the preceding year was shown in 1905, but in 1906 there was a small decline in the total. It was, however, still $300,000,000 above that of 1904. A remarkable feature in 1906 was the increase in the already large holding of the Bank of Russia; but that bank, it must be noted, holds practically the reserves of the Russian treasury as well as its own commercial balances. It has been charged also that the bank includes foreign gold bills in its balance; this, if true, would be to that extent a duplication of gold reported by other banks. The specie holdings of the New York banks, including both gold and silver, were $176,212,800 in the last week of 1905, and $179,323,000 at the end of 1906.

Price of Silver.—The commercial value of silver rose gradually through the year, reaching its highest point in November. From that there was a recession, but comparatively small in amount. The average monthly prices of silver in the New York and London markets for five years past are given in the accompanying tables.

AVERAGE PRICE OF SILVER.
New York, in cents per fine ounce.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1902	55.56	55.09	54.23	52.72	51.31	52.36	52.88	52.52	51.52	50.57	49.07	48.03	52.16
1903	47.57	47.89	48.72	50.56	54.11	52.86	53.92	55.36	58.00	60.36	58.11	55.375	53.57
1904	57.055	57.592	56.741	54.202	55.430	55.673	58.095	57.806	57.120	57.923	58.453	60.563	57.221
1905	60.690	61.023	58.046	56.600	57.832	58.428	58.915	60.259	61.695	62.034	63.849	64.850	60.352
1906	65.288	66.108	64.597	64.765	66.976	65.394	65.105	65.949	67.927	69.523	70.813	69.050	66.791

London, in pence per standard ounce, 0.925 fine.

1902	25.62	25.41	25.00	24.34	23.71	24.17	24.38	24.23	23.88	23.40	22.70	22.21	24.09
1903	21.98	22.11	22.49	23.38	24.89	24.29	24.86	25.63	26.75	27.89	27.01	25.73	24.75
1904	26.423	26.665	26.164	24.974	25.578	25.644	26.760	26.591	26.349	26.760	26.952	27.930	26.399
1905	27.930	28.047	26.794	26.108	26.664	26.910	27.163	27.822	28.528	28.637	29.493	29.977	27.839
1906	30.113	30.464	29.854	29.984	30.968	30.185	30.113	30.529	31.483	32.148	32.671	32.003	30.868

AVERAGE PRICE OF BAR SILVER IN LONDON, 1833–1902.
In pence per standard ounce, 0.925 fine.

Year.	Pence.	Year.	Pence	Year	Pence.	Year.	Pence	Year.	Pence.	Year.	Pence.	Year.	Pence.
1833	59.1875	1843	59.1875	1853	61.5000	1863	61.3750	1873	59.2500	1883	50.5625	1893	35.6250
1834	59.9375	1844	59.5000	1854	61.5000	1864	61.3750	1874	58.3125	1884	50.6250	1894	28.9375
1835	59.6875	1845	59.2500	1855	61.3125	1865	61.0625	1875	56.8750	1885	48.6250	1895	29.8750
1836	60.0000	1846	59.3125	1856	61.3125	1866	61.1250	1876	52.7500	1886	45.3750	1896	30.7500
1837	59.5625	1847	59.6875	1857	61.7500	1867	60.5625	1877	54.8125	1887	44.6250	1897	27.5625
1838	59.5000	1848	59.5000	1858	61.3125	1868	60.5000	1878	52.5625	1888	42.8750	1898	26.9375
1839	60.3750	1849	59.7500	1859	62.0625	1869	60.4375	1879	51.2500	1889	42.6875	1899	27.4375
1840	60.3750	1850	60.0625	1860	61.6875	1870	60.5625	1880	52.2500	1890	47.6875	1900	28.2500
1841	60.0625	1851	61.0000	1861	60.8125	1871	60.5000	1881	51.6875	1891	45.0625	1901	27.1875
1842	59.4375	1852	60.5000	1862	61.4375	1872	60.3125	1882	51.6250	1892	39.8125	1902	24.0900

The high price of silver in 1906 was due chiefly to two causes, the first being the heavy demand from India, the result of three successive years of prosperity in that country; the second, the large demand for silver for use in the arts, incident to a period of general prosperity. Another reason for the maintenance of prices was the purchase of a considerable quantity of the metal by the United States mint.

United States Mint Purchases.—Under authority of an act of Congress, the United States Treasury Department began in August, to buy silver in the market, to be used by the mint for the coinage of subsidiary coins. These purchases continued up to the early part of November, when they were temporarily discontinued, but were resumed about a month later. Before inviting tenders in August, the Department, anticipating that the invitation to sell might have an effect upon the market, bought about 2,500,000 oz. in London for future delivery. These purchases were afterward exchanged for metal in this country, which was delivered at the several mints, as shown below. The total purchases made, with the deliveries, are shown below in ounces, the London silver above noted being given separately:

Mint.	American.	London.	Total.
Philadelphia	875,000	1,630,307	2,505,307
New Orleans	850,000	400,119	1,250,119
Denver	1,950 000	403,299	2,353,299
N. Y. Assay Office	50,000	....	50,000
Total	3,725,000	2,433,725	6,158,725

The London purchases were made at a price ranging from 65.474 to 65.772c. per ounce, and averaging about 65.60c., which was close to the current market price. The lowest price paid in the American market was 65.17c., Philadelphia delivery, on Aug. 2; the highest was 71.952, New Orleans delivery, on Nov. 9. The prices paid in America included delivery, and they ranged almost uniformly about 0.4c. per ounce for Philadelphia and Denver, and about 0.5c. for New Orleans delivery above

the New York quotations of even dates. The total silver bought by the mint was about 10 per cent. of the United States production for the year.

UNITED STATES: EXPORTS AND IMPORTS OF SILVER.

	1903	1904	1905	1906
Exports	$40,610,342	$50,312,745	$57,513,102	$60,957,091
Imports	23,974,508	26,087,042	35,939,135	44,227,841
Excess, Exports	£16,635,834	$24,225,703	$21,573,967	$16,729,250

Silver Movement.—The approximate quantity of silver exported from the United States in 1906, at the average value of the year, was 91,265,-400 oz., an actual decrease from 1905, though the value showed an increase owing to the higher price of the metal. The exports directly to the East in 1906 were only $2,762,316 in value, while those in Great Britain were $53,004,996. The larger imports in 1906 were $36,108,487 from Mexico and $5,697,255 from Canada. The approximate quantity imported in 1906 was 66,218,300 oz.; so that the net exports for the year were 25,047,100 oz., a decrease of 10,700,000 oz. from the preceding year.

GREAT BRITAIN: EXPORTS AND IMPORTS OF SILVER.

	1903	1904	1905	1906
Exports	£11,466,726	£13,263,694	£14,561,677	£18,865,285
Imports	10,310,330	11,687,339	12,992,014	17,288,063
Excess, exports	£1,156,396	£1,576,355	£1,569,663	£1,577,222

At the average prices of the year, the value of the exports from Great Britain represented approximately 125,53,100 oz. silver in 1906, or about 73 per cent., of the silver production of the world. The main part of the imports in 1906 was credited to the United States, which furnished 86.1 per cent. of the total. This includes a large quantity of Mexican silver, parted and refined in this country. There were in 1906, considerable imports from the East, due to shipments from China, an unusual movement. In 1905 there were large exports of silver to Russia and Japan, which presumably found their way ultimately to China through Manchuria. There were only very small shipments to those countries in 1906.

EXPORTS OF SILVER FROM LONDON TO THE EAST (*a*)

	1903	1904	1905	1906
India	£7,423,330	£9,527,618	£7,230,421	£14,867,196
China	310,060	512,792	886,847	430,700
The Straits	821,879	79,268	38,299	1,750
Total	£8,555,269	£10,119,678	£8,155,567	£15,299,646

(*a*) As reported by Pixley & Abell.

The exports from London to the East in 1906 were the largest recorded in any one year for 75 years; with one exception—the year 1877—when they reached a total of £17,007,458, but on a much higher value of the metal. The cessation of the shipments to the Straits was due to the adoption of a new currency, based on a gold standard in that colony.

To sum up, the figures given show that London, through the control of the Eastern trade, is still the great silver center where about three-quarters of the world's output is marketed; and the East is still the great buyer, taking from 60 to 65 per cent. of the total yearly supply. Among Eastern countries India is the most prominent; it takes the largest quantity, and the silver sent there practically disappears from commerce. From China there is occasionally a reflex movement—as there was in 1906—when the trade balance turns against that country; but India never gives up the silver it has taken. Apart from the large amount required for its currency, the accumulations of its people, small and great, are made in silver, as they have been for centuries. These hoards are in coin and in the form of ornaments of different kinds, and they do not leave the country, even in the times of extremest stress.

Another cause which contributed to the higher prices of silver in 1905 and 1906 was a large demand for use in the arts; a condition which usually follows a period of business activity and prosperity.

France is the largest purchaser among European nations, using a large amount for coinage and holding also a leading place in the manufacture of ornamental and other wares in which silver is used.

Progress in Gold Milling in 1906.

By Robert H. Richards and Charles E. Locke.

Hoscur Ore Feeder.[1]—This gives satisfactory results in all the mines on the Kolar field. It is based on the principle of the Challenge feeder but eliminates all springs save one. The rotation of the feed plate is obtained through a specially designed pawl working in a V-shaped groove cut in the friction wheel. This pawl is kept in constant contact by means of a rubber buffer.

Muntz Metal Plates.[2]—The Bamberger-DeLamar Gold Mines Company stamping in a 0.15 per cent. cyanide solution, formerly used copper plates with 1 oz. of silver per square foot. These rapidly became rough and pitted and were replaced with silver-plated Muntz metal plates which gave satisfactory results.

Richards Rapid Economy Stamp Mill.[3]—This is a form of spring stamp which is given an up and down motion through a bell crank lever driven by an eccentric.

[1]*Mining Reporter*, 1906, LIV, p. 83; abstracted from London *Min. Journ.*
[2]A. R. Parsons, *Min. and Sci. Press*, 1906, XCII, p. 281.
[3]*Mining Reporter*, 1906, LIII, p. 247.

Battery Frame.[1]—A cantilever steel battery frame designed by M. P. Boss and installed at Tonopah and Goldfield has concrete foundations and a side clearance around the mortar. The stamp frame is supported by steel beams extending horizontally from the ore bin.

Heavier Stamps on the Rand.[2]—The new mills of the Cason and Angelo use 1350 ℔. stamps and at the Simmer & Jack mill, it has been decided to use 1415-℔. stamps. In 1895 only one mill had 1250-℔. stamps; now the majority of stamps are of this weight. In the same time the stamp capacity has been increased from four to five tons per stamp per day.

Mills of the Simmer & Jack Proprietary.[3]—An improvement in the Challenge feeder has been made by removing the rocking bar or feeder arm from the top of the mortar box and ore chute and placing it on the king post under the top guide block, the power to drive the feeder gear being transmitted through a manilla rope of ½ or ⅝ in. diameter. The center box, fulcrum bracket, connecting rod, springs, etc., are no longer required. This feeder can be adjusted and operated entirely from the cam platform. For measuring the drop of the stamps a rod sliding loosely in a tube is used. The tube has a longitudinal slot through which projects a pin attached to the rod. Two rings may be moved up and down on the tube. By placing this instrument vertically between the tappet and upper guide for a few seconds the rings may be set to mark the upper and lower ends of the pin movement and this distance can be readily measured when the instrument is removed. A solution of hydrochloric acid of 1.6 per cent. has been found preferable to sulphuric acid, nitric acid, sal ammoniac, and potassium cyanide for dressing the plates. The use of cyanide has been shown by experiment to dissolve gold from the plates.

Some Experiences in Stamp Milling.[4]—Algernon Del Mar makes the following suggestions: Have the mill well covered and keep it warm. Avoid too many stamps on one cam shaft. Keep the plates in good condition. Keep the mill in good repair; loose guides will cause a lot of trouble.

Percy Morgan advocates the separation of amalgamated plates from the stamp battery and the placing of them in a separate room. His arguments are as follows: (1.) Less noise and vibration. (2.) Better lighting. (3.) More space for both stamps and tables. (4.) Better arrangements for the tables, and for pulp distribution. (5.) No need of hurrying the work. (6.) Prevention of theft. (7.) Possibility of classifying the pulp.

J. R. Sears suggests that pulp flow sidewise over the plates instead of endwise.

[1]*Min. and Sci. Press*, 1906, XCII, p. 104.
[2]*Mining Reporter*, 1906, LIII, p. 287.
[3]G. O. Smart, *Journ. Chem., Met. and Min. Soc.* of South Africa, 1906, VII, p. 133.
[4]*Min. and Sci. Press*, 1906, XCIII, pp 138, 379, 534, 566.

C. G. Dennis reports that a plate at a distance from a mortar was not satisfactory. More water and greater slope were necessary, and the saving was not so good. The lack of the wave produced by the mortar splash appeared to be a disadvantage.

Advantages of Stamps.[1]—In a general article on stamp milling the author states that the advantages of simplicity and ease of making repairs have given stamps their prestige. The one-stamp batteries have the advantage of higher crushing capacity but they do not amalgamate so satisfactorily as the five-stamp batteries and they are more complicated.

Wood Stamp Mill in Colombia.[2]—The stem weighs 125 ℔. and the steel shoe is of the same weight. The stamps crush 400 to 600 ℔. per stamp per day through 11-mesh screen. In some mines the tailings are concentrated in sluices or on buddles and the sulphides are worked in arrastras. The cam shaft is a wooden log having wooden pins for cams.

Quartz Milling in Colombia.[3]—Wooden stamps with iron shoes and a rock mortar are used. Stamps are in multiples of 3, 4, 5, or 6 stamps per battery. An overshot wheel is generally the motive power. Screens are wooden boards punched with nail holes. The pulp passes over blankets or gunny sacks and the concentrates are collected and washed in a *batea* to remove free gold, ground in an arrastra, reconcentrated on rough planks and again washed in a *batea*. The final tailings may be allowed to rust and then be ground and washed again. About $300 will erect a 10-stamp mill with a capacity of one to three tons in 24 hours.

Stamp Mill Practice.[4]—Some general ideas are given from the author's own experience. Hight of discharge and amount of feed water should be depended upon to regulate the size of the product rather than the size of the screen aperture. The pulp should be frequently tested with testing screens. Russia iron screens with clear slots diagonally arranged are to be preferred. They should have a high percentage of opening. Tyler rolled wire screen is good but it is weak and short lived.

The author's experience with concrete mortar blocks has been unsatisfactory owing to rapid wear. Rubber cushions under the mortar accelerate the wear. Anvil blocks increase the difficulty of rigidly anchoring the mortar and have a tendency to increase the breakage of stamp stems. In one case the breakage with wooden mortar blocks was only one-third of that with concrete blocks. A wooden mortar block with a sheet of pure rubber ¼ in. thick on top is best. Next to this is a concrete block on top of which is placed a ¼ in. wrought iron sheet 4 in. wider than the mortar base and 2 in. longer. The rubber cushion is placed on top of this.

[1]Algernon Del Mar, *Min. and Sci. Press*, 1906, XCIII, p. 597.
[2]H. G. Granger, *Eng. and Min. Journ.*, 1906, LXXXII, p. 194.
[3]F. F. Sharpless, *Eng. and Min. Journ.*. 1906, LXXXII, p. 485.
[4]Courtenay DeKalb, *Journ. Can. Min. Inst.*, March, 1906; abstracted in *Min. Reporter*, 1996, LIV, p. 11; *Eng. and Min. Journ.*, 1906, LXXXII, p. 245. *Can. Min. Rev.*, 1906, XXVI, p. 153. *Mines and Minerals*, 1906, XXVII, p. 135.

Stamp duty should not be increased at the expense of the efficiency of amalgamation. Mortar liners are objectionable since they soon wear loose. A mortar with a thick base and sides is preferable. For shoes he recommends open hearth steel with 0.60 per cent. carbon, forged at a bright cherry red and necks annealed; for dies use 0.80 per cent. carbon steel forged at a dark yellow with bases annealed subsequently. These wear to extreme thinness. Inside plates should be used and the splash should be so adjusted by the drop of the stamps as to prevent scouring these plates. Bedding of ores is suggested as a means of obtaining constant ore mixtures.

Outside plates are best cleaned by borax soap. Potassium cyanide and caustic alkalies should never be used. Discolorations may be remedied by a hard brisk rubbing of the plate with whisk brooms. Discolorations from copper sulphide in the ore, which forms copper sulphate, is remedied by feeding salt to the battery with the ore. A dark-crusted deposit indicates antimony and for this there is no effective remedy. Abundant area of outside plates (not less than 48 sq.ft.) is necessary. There is no advantage in coating the plates with more than 1 oz. of silver per square foot. A first dressing on bare copper with dilute silver amalgam is more sensitive as a catcher of amalgam than a silver-plated copper plate.

Masonry Foundations for Stamp Mortars.[1]—Mr. Foote differs with Mr. DeKalb who considers wooden mortar blocks to be better than masonry or concrete. If the foundation is well constructed and if the mortar has a perfect fit on its block, either with or without the use of a rubber cushion, then there will be no difficulty. He gives details of mortar blocks in three mills, two of which were of masonry and one was concrete (See THE MINERAL INDUSTRY, Vol. XIII, p. 200) which have given satisfactory results. He agrees with Mr. DeKalb in condemning the use of anvil blocks but does not believe that solid foundations cause an increased breakage of stamp stems.

North Star Mill.[2]—The special features of the new 40-stamp North Star Mill at Grass Valley, Cal., are rough cement masonry foundations, steel-channel stamp frames, thin punched steel screens with 1/40-in. holes and space of unperforated metal left around each square inch for strength, special iron guides, cast iron tables for the amalgamated plates, concrete panning sinks and Dodd buddles for concentrating. The mill is of the regular California pattern and crushes 3.2 to 3.25 tons per stamp per 24 hours. Tailings are sent to the cyanide plant.

Stamp Mill Notes.[3]—Experiments with 1050-lb. stamps gave results shown in the following table:

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[1]A. D. Foote, *Eng. and Min. Journ.*, 1906, LXXXII, p. 886.
[2]A. D. Foote, *Min. and Sci. Press*, 1906, XCII, p. 240; G. E. Alexander, *Mining Reporter*, 1906. LIV p. 56
[3]W. Beaver, *Journ. Chem., Met. and Min. Soc*. of South Africa, 1906. VI, p. 215.

Size of feed.	Capac. per Stamp per Day.	Size of Feed.	Capac. per Stamp per Day.	Size of Feed.	Capac. per Stamp per Day.
Inches.	Tons.	Inches.	Tons.	Inches.	Tons
2.5	4.04	1.5	4.60	0 5	4.35
2.0	4.10	1.0	4.74	0.25	4.10

In another experiment it was found that a stamp duty of 4.6 tons per day was maintained irrespective of whether material below 200-mesh was sifted out before feeding to the battery. The ore in this case contained 71.53 per cent. below 1.5 in., 53.19 per cent. below 1 in., 35.14 per cent. below 0.5 in., 24.12 per cent. below 0.25 in., 16.19 per cent. below $\frac{1}{8}$ in., 11.54 per cent. below 200-mesh (per square inch) and 7.84 per cent. below 700-mesh (per square inch). The author concludes that 1.5 in. is the correct size for his stamps and that it is advantageous to screen out all below 0.25 in. and send it to a tube mill.

A higher stamp capacity was obtained when the ore was all fed under the middle stamp than when it was distributed under all the stamps. A launder which may be placed in front of the mortar to deliver the pulp to an auxiliary plate allows the regular plates to be cleaned without stopping the stamps and adds considerable to the daily capacity of the mill. In order that the pulp may run in the gentle sloping launder the latter is supplied with a glass bottom.

Sodium amalgam assists amalgamation slightly but too much of it has a tendency to soften the amalgam on the plates until they finally become bare.

Discussion on this paper brought out the opinion that to crush fine material the stamps should be set with a lower drop and run faster. This aids discharge and increases the capacity. For 2.5-in. ore a drop of 9 in. and 96 drops per minute will give the highest efficiency; for 0.5-in. feed use 6-in. drop and 108 drops, and for 0.25 in. use 5-in. drop and 112 drops. The hight of discharge should be twice the largest lump in the feed, i.e., for 2.5-in. feed use 5-in. discharge and so on. The ratio of the area of the shoe to the weight of the stamp is believed to be an important point in obtaining the maximum efficiency with varying sizes of feed.

Stamp Milling in Northern California.[1]—At the Southern Cross mill of the Edel Mining Company in Butte county, Cal., the ore is pure quartz, very friable, and containing 1 per cent. of sulphides of iron, copper and lead. There are two batteries. Each battery has two stamps in a quadruple discharge mortar. The stamps weigh 950 ℔. and drop 5.5 in. 115 drops per minute. Each battery has inside plate, two splash plates, a lip

[1]Algernon Del Mar, *Min. and Sci. Press*, 1906, XCII, p. 143.

plate and an apron plate in five sections, each 4 ft. square and overlapping. One Frue vanner for each battery completes the equipment. Several tests were made with the following results:

TESTS AT SOUTHERN CROSS MILL.

Test No.	Screen.	Size of Opening Inches.	Hight of Discharge. Inches.	Per Cent. through 60-mesh Brass Wire.	Per Cent. through 40-mesh Brass Wire.	Assay of Tailings.	Tonnage per Stamp per Pound-hour.
1	No. 6 wire	0.027	6	64	88	$0.90	300
2	No. 4 blued steel	0.035	6	70	88	1.20	270
3	No 6 wire	0.027	5½	58	81	Trace	320
4	No. 0 tin	0.024	5½	91	94	0.90	131
5	No. 6 blued steel	0.035	5½	64	88	Trace	280
6	No. 6 wire	0.027	4½	45	86	1.45	340
7	No. 0 tin	0.024	4½	70	85	1.85	146
8	No. 6 wire	0.027	3½	33	79	1.45	420
9	No. 0 tin	0.024	3½	50	80	0.94	360

A test of the tailings showed:—material through 60 mesh assayed, $4.36; material through 60-mesh after washing off slime and fine particles assayed, $0.30; slime washed off assayed, $6.18; coarse material assayed, trace.

Gold Milling in Western Siberia.[1]—Crushing is done by Krupp stamps or the Urals edge runners and these are followed by amalgamated plates. Only one mine uses concentration and it has Stein-Bilharz tables. The largest mine in the Mariinsk district has four Chile mills. The weight of each roller is four tons. Running at 12 r.p.m. one mill with two rollers crushes 12 to 14 tons of moderately hard quartz and beresite through a 20-mesh screen. A few details of stamps are given.

Ore Treatment at the Combination Mine, Goldfield, Nevada.[2]—There are two classes of ore, viz., the oxide and the sulphide. The oxide ore is crushed in stamps and undergoes inside and outside amalgamation. The coarser material from the plates is reground in Bryan mills and joins the fine to be treated on Frue and Triumph vanners. The tailings of these vanners are cyanided. Full details of the apparatus are given in the original article.

The sulphide ore is silicious and contains fine auriferous pyrite which is not susceptible to raw cyanide treatment. It undergoes the following treatment:

(1) Sturtevant breaker, 10x16 in. crushing to 0.5 in., By belt conveyor to stamp bins and thence to (2).
(2) Ten 1350-lb. stamps with 12-mesh screens. To (3).
(3) Two amalgamating tables. Amalgam; pulp to (4).
(4) Two classifiers. Spigots to (5); overflow to (6).
(5) One Wilfley table. Concentrates; tailings elevated to (7).
(6) One Bryan mill with 40-mesh screen. To (7).
(7) Three 6-ft. Frue vanners. Concentrates; tailings by sand pump to (8).
(8) One classifier. Spigot to (9); overflow to (10).
(9) One 4x12-ft. Abbé tube mill. To (10).
(10) Three Wilfley slimers. Concentrates; tailings to cyanide plant.

[1]L. Tovey, *Eng. and Min. Journ.*, 1906, LXXXII, p. 577
[2]F. L. Bosqui, *Min. and Sci. Press*, 1906, XCIII, pp. 413, 451; Mark R. Lamb, *Eng. and Min. Journ.*, 1906, LXXXI, p. 1236.

Over 80 per cent. of the gold is saved by this concentration. Some idea of the efficiency of the various steps may be gained from the following results obtained from a little different arrangement of machines: Material containing 3.01 oz. gold per ton is crushed by stamps to 30 mesh and concentrated on a Wilfley table yielding concentrates (6 per cent. of the total) assaying 27.4 oz. The tailings are reground through 60 mesh and reconcentrated yielding a concentrate (2.11 per cent.) assaying 19.8 oz. and tailings which are further crushed to 100 mesh and concentrated. This step yields 0.4 per cent. of concentrates assaying 20.4 oz. and tailings which are ground to 200 mesh and passed over a canvas table. The concentrates of this table are 10 per cent. of the ore and assay 3.97 oz. while the tailings assay 0.52 oz. and are cyanided. The extraction by concentration is 80 per cent. The excessive concentration indicated is necessary to recover the gold in the pyrite before the cyanide treatment. Detailed figures as to the cost of milling and cyaniding are given.

Amalgamation of Gold Ore.[1]—The following is a summary of a large amount of historical and experimental data on the subject of amalgams and amalgamation.

(1) Gold absorbs mercury, forming a solid solution which may contain as much as 13 atomic per cent. of mercury. Beyond this, an inter-metallic compound containing gold or mercury in solution (or a second solid solution) is formed, which contains 17.5 atomic per cent. of mercury. Ordinary amalgam, which is not in a state of equilibrium, consists of one or both of the foregoing, usually the former, mixed with an excess of mercury which coats the particles and causes them to cohere.

(2) Amalgamation is a physical process, the chemical actions involved being chiefly inimical (excepting those purposely induced). The gold grains are wetted by the mercury and sink beneath the surface of the mercury film on the plates; this is facilitated by feeding mercury to the stamp, so that the grains may be thoroughly wetted before coming in contact with the plates. The disadvantages of this procedure are discussed in the original article. The surface-tension of the mercury draws the gold beneath the surface, and holds it against the plate. By diffusion into the metal of the plates the amalgam often becomes strongly adherent. Silver-plating is useful, because it prevents the solution of the copper in the mercury, and therefore the harmful chemical reactions that result therefrom. Muntz metal plates exhibit the same effect, and in addition diffusion of amalgam into them is very slight, so that it is readily removed. Silvered plates will hold a thicker film of mercury than plain copper, and plates coated with gold amalgam a thicker film than either. This assists the "catching" of the gold.

[1] T. T. Read, *Bulletin* A. I. M. E., 1906, p. 467; abstracted in *Mining Reporter*, 1906 LIII, p 612; *Min. and Sci. Press*, 1906, XCIII, p. 15.

(3) Variations in temperature make themselves felt in slight changes of a number of factors rather than large changes in any one. According to the relative importance of these factors in each case the total effect may vary. The most important undesirable effects of raising the temperature are the increased solubility of harmful salts, and a corresponding increase of the precipitation of base metals into the mercury; this both hinders its proper action and leads to its loss. Rise of temperature also diminishes the surface-tension and viscosity of the mercury, which allows it to be more readily "floured." The force with which the gold is drawn beneath the mercury and held against the plate is also decreased. On the other hand, by an increase in temperature the wetting of the gold by the mercury and the "catching" of it by the plates is facilitated, as is the coalescing of the globules of mercury.

(4) Increase of temperature causes increased absorption of mercury by the gold and by the plates. Changes in temperature cause changes in all the foregoing factors. The retaining of a constant temperature is therefore most favorable to successful working. A comparatively low temperature is better where the influence of soluble salts in the ore has to be considered (which is usually the case); but when this may be neglected, as high a temperature as can economically be maintained, without variation, is most favorable to successful amalgamation.

Fine Grinding by Tube Mills at El Oro, Mexico.[1]—Ore is crushed in stamps with 0.74 mm. screens. After leaving the plates the pulp is classified and the coarse pulp is reground in tube mills. In comparing results of work the "Sand index" of a product is used. That is, a sizing test of a product is made and the percentage of each size is multiplied by the average percentage extraction on that size of sand. The sum of the products of these multiplications is the "Sand index" or calculated extraction of the special product under consideration.

Concrete foundations are used for the tube mills. The driving pinion should be so revolved as to lift the gearing of the tube mill and press down on its own bearing. Danish pebbles and steel liners are used. Home made liners last as long as Krupp liners and cost one half as much. The consumption of liners per ton of sand is 1.2 lb. and of pebbles 5 lb. The average cost of regrinding is \$0.371 per ton by steam power or \$0.274 by electric power.

Comparison of "Sand indexes" of numerous products indicates; (1) that the efficiency of the tube mill increases proportionally to the amount of pebbles contained in the mill; (2) that the efficiency increases with the coarseness of sand fed to the mill; (3) the efficiency decreases proportionally to the rate of feed.

Tube Mills.—These are considered somewhat in the article on "Progress

[1]G. Caetani and E. Burt, *Bulletin* A. I. M. E., 1906, p. 83; abstracted in *Mines and Minerals*, 1906, XXVI, p. 511.

in Ore Dressing," and in other articles under "Gold and Silver" elsewhere in this volume.

Wet Silver Mill.[1]—The combination wet silver mill in Granite county, Montana, treats a quartz ore containing argentiferous tetrahedrite and malachite. The latter is free milling; the former requires roasting. The ore runs 25 oz. silver per ton. Some waste is sorted out under ground. The ore is hoisted in cars and dumped on (1).

(1) Grizzly. Oversize spalled and hand-picked into waste and good ore to (2); undersize to (2).
(2) Bin. By wagon to (3).
(3) Storage bin. By car to (4).
(4) Grizzly with bars 1x3 in. and 10 ft. long, with 2-in. spaces. Oversize to (5); undersize to (6).
(5) Blake breaker, 10x20 in., making 200 r.p.m. To (6).
(6) Bin. By four Challenge feeders to (7).
(7) Twenty stamps. By two bucket elevators (only one used) to (8).
(8) Distributing box. To (9).
(9) Twelve 4-ft. Frue vanners. Concentrates (tetrahedrite) to (10); tailings (malachite and gangue) to (11).
(10) Two Brückner roasters and cooling floor. By elevator to roasted ore bins and thence to (12).
(11) Eighteen tanks, 9¼x6½ ft. by 2¾ ft. deep. Settlings to (12); overflow water.
(12) Fourteen amalgamating pans. The charge in each is 3500 to 4000 ℔. of which one-fourth is roasted ore and the balance tank settlings. These pans are followed by seven settlers and the necessary equipments for pan amalgamation, such as mercury elevator, clean up barrel, retort, etc.

Details of the stamps are as follows: weight 1100 ℔.; drop, 6 to 9 in.; 100 drops per minute; double discharge mortar with the back discharge closed; brass screens 30-mesh (No. 30 wire) or 24 mesh (No. 25 wire); screen frame 4x2¼ ft. divided into three panels; mortar block 2½x5x8 ft. deep imbedded in concrete; dies and shoes 9 in. diameter; cylindrical dies imbedded in iron filings; tappet with three keys and a cast iron split collar above it; stems 3½ in. by 20 ft.; Weir guides consisting of wooden bushings in cast iron boxes; Blanton cams; order of drop, 1-3-5-2-4.

THE WEARING PARTS OF COMBINATION MILL:

Part.	Material.	Weight Pounds.	Life Months.	Total consumption of metal	
				Per month, Pounds.	Per ton of ore, Pounds.
Stamp shoe	White cast iron	125	0.5	5,000	2.21
" die	" " "	125	0.5	5,000	2.21
" boss	" " "	220	8.0	550	.24
" tappet	" " "	130	24.0	108	.05
" cam	Cast steel	220	Indefinite		
" cam shaft	" "	925	18.0		
" stem	" "	650	18.0		
" screen	Brass wire	7	0.06⅔	420	.19
Pan die	White cast iron	800	1.5	747	.33
" muller	" " "	600	1.0	8,400	3.73
" spider	" " "	225	12.0	263	.12
Settler die	" " "	440	1.0	3,080	1.37
" muller	" " "	1,400	6.0	1,633	.72
" spider	" " "	425	3.0	992	.44
Retort	Gray cast iron	2,200	8.0	1,100	.49
Vanner hanger rod	Cast iron		0.3		
" " bearing	" "		0.3		
" worm gear	" "		0.75		
" 12-in. end roller	Galvanized steel		4.0		
" 2¼-in. belt roller	" "		3.0		
" belt	Rubber		10.0		

[1]R. B. Brinsmade, *Mines and Minerals*, 1906, XXVI, p. 492.

The vanners have smooth belts; slope 3 to 6 in. in 12 ft.; travel 6 ft. per minute; 240 strokes per minute; concentrates removed by scraping conveyor from the head box into a "crawl box" near the foot of the vanner. This box is hoisted up and trammed to the roaster room when full.

Full details of the roasting and amalgamating appliances are given in the original article. The mill treats 75 tons per day and concentrates four tons into one. The mill is run by a 130-h.p. simple non-condensing Corliss engine.

The labor required is one day foreman, one night foreman, one breaker man, two battery men, two vanner men, three vanner helpers, six tank men, two pan men, two pan helpers, one settler man, one retort man, two roaster men, two cooling floor men, three engineers, one millwright, one blacksmith, one blacksmith's helper and one laborer.

Concentration of Black Sands.[1]—This report embodies the results of extensive assays and concentration tests upon the black sands of the United States. For the concentration tests, the following apparatus was used:—Screens with 8-mm., 2-mm. and 0.5-mm. holes, hydraulic classifier, hand jig, Wilfley, Christensen, Woodbury and Pinder tables, Wetherill magnet, greased plate and amalgamated plate. Before using the Wetherill magnet, standardizing tests were made to determine the strength of current necessary to lift the following minerals: platinum, chromite, chrome iron ore, almandine garnet, monazite, zircon, wolframite, columbite and josephinite. The results in the case of platinum are given in the accompanying table.

STRENGTH OF CURRENT NECESSARY TO LIFT PLATINUM.

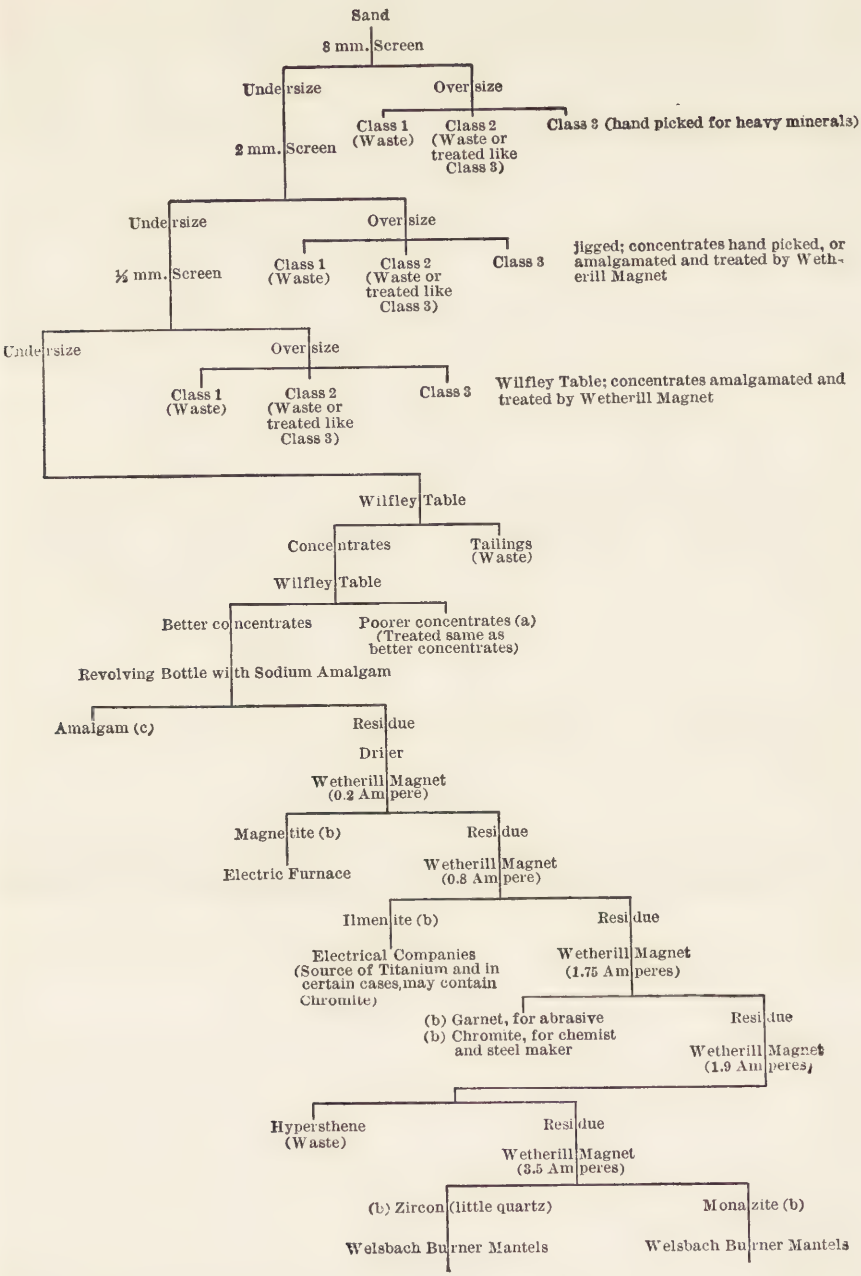
Current Amperes.	Weight Grams.	Per Cent. Extracted	Current Amperes.	Weight Grams.	Per Cent. Extracted.	Current Amperes.	Weight Grams.	Per Cent. Extracted.
0.15	0.606	4.07	1.50	.258	1.73	3.00	.200	1.34
0.20	.269	1.81	1.75	.156	1.05	3.25	.217	1.46
0.50	.260	1.75	2.00	.973	6.54	3.50	.044	.30
0.75	.082	.55	2.25	.272	1.83	3.75	.048	.32
1.00	.210	1.41	2.50	.452	3.04	4.00	.052	.35
1.25	.218	1.47	2.75	.254	1.71	4.00 (a)	10.314	69.26

(a) Not lifted at 4 amperes.

This shows that although most of the platinum was non-magnetic, still a little was lifted at each strength of the magnet. The same condition was found to exist to a greater or lesser degree with all the minerals tested.

The accompanying tree is recommended for the commercial separation of three classes of material, viz., (1) River or beach sands. (2) Hydraulic or dredger tailings. (3) Hydraulic or dredger clean-up sands.

[1] D. T Day and R. H. Richards, U. S. Geological Survey, advance chapter from "Mineral Resources" for 1905.



(*a*) Should the gold be encased in some other mineral and therefore can not be amalgamated, these two products should be ground finer to free the gold, and the amalgamation repeated.

(*b*) Each of these six minerals may need a little cleaning on the Wilfley table before being shipped

(*c*) Contains gold and platinum. On small scale dissolve in nitric acid leaving gold and platinum metals. On large scale, retort.

Hydraulicking and Gold Dredging.—The reader is referred to the article by Mr. J. P. Hutchins in this volume for a record of the progress in these branches during 1906.

Progress in Gold-ore Treatment During 1906.

By Alfred James.

Fine sliming and the treatment of slime may be taken as the main directions of improvement in this branch of metallurgy during 1906. The use of tube mills has progressed by leaps and bounds, and is now rapidly becoming well-nigh universal, but with the adoption of fine sliming the question of slime treatment has forced itself into still greater prominence, and, as stated in my article last year, some of our keenest metallurgists have been attacking this problem. Their endeavors have met with a considerable measure of success.

As will be indicated later, progress has also been made in the treatment of difficult ores—including silver-gold ores, antimony-gold concentrates and cupriferous tailings—and in crushing, roasting, and conveying we are ahead of last year's practice. Fine sliming and prolonged cyanide treatment are in many cases ousting chloridizing roasting and the old *patio* process for the treatment of silver ores.

It is difficult to put one's finger on any one part of the world as having made the most striking progress of the year. Australia probably still has the pre-eminence. Quick-witted inventions, which may have originated in other countries, appear to be more quickly put into practice with useful results there than elsewhere; and just as Australia has taken the lead and been pre-eminent in fine sliming and in circulating cyanide solutions through the mill, so now it seems to lead in slime treatment. Probably the emulation arising from the keen up-to-date methods of the leading firm of mining engineers on the one hand, and independent managers on the other hand, has tended to place Australia in the front in the matter of up-to-date methods, high extractions and low costs. Certainly such men as Hamilton, Moss, Nicholson, and Klug, and the gentlemen associated with Bewick, Moreing & Co., have cause for congratulation on the results they have achieved, and are achieving, and the adoption of their methods in other fields shows that the industry generally, and not merely locally, is under an obligation to them as well as to the alert and progressive West Australian Chamber of Mines, which has published an excellent series of articles by Mr. Allen on ore treatment as carried out at the various local installations.

American practice has chiefly busied itself with tube mills and slimes treatment. Argall, Bosqui, Butters and Merrill have all been contributing to our advancement by devising improved methods. In Africa the publication of Messrs. Denny's results at the Meyer & Charlton and New

Goch has caused much interest, and generally extraction and output have been increased and cost lowered, mainly by the use of tube mills; indeed, once more record gold outputs are being produced from this territory, which has more than recovered the set-back occasioned by the war.

Crushing.—The tendency to stage-crushing is in still greater evidence. Coarse crushers, fine crushers, heavy stamps, pans, tube mills, are not an uncommon sequence. With reference to crushing (or breaking), however, it has not yet been proved that crushing for stamps to finer than 2½-in. cube is advantageous. Experiments seem to indicate that finer crushing is more expensive than the gain arising from any increase in stamp duty resulting from such finer crushing.

As for crushers, the Gates type still mainly holds the field for big work, but the Bigelow crusher (of the Blake type, but with the pitman working in compression and having its weight assisting the crushing stroke) appears to be replacing the other types at Kalgoorlie, and to show a lower running cost than the gyratory or older reciprocating patterns.

Stamps of 1550 lb. weight are now in use on the Rand, and it looks as if the weight limit has not yet been reached. One has heard but little this year of anvil blocks for mortars, but at the Waihi Grand Junction they seem to have made for a considerable increase in output beyond that originally expected. Judging by output per horse-power, ball mills would appear to be the most efficient machine, the best types giving an output of over two tons per horse-power day, crushing all the ore through screening of 700 apertures per sq.in., but the cost of dry crushing (milling) at the South Kalgurli (August) appears to be about 3s. per ton as against roughly 1s. 9d. per ton (wet crushing) at the Oroya Brownhill, Lake View and Ivanhoe. On the other hand, the latter crush through very much coarser screening (100-, 300- and 225-apertures per sq.in. respectively), reducing the over-size product by their pans and tube mills.

Fine Grinding.—The tendency has been to use stamps for coarser crushing only—i.e., to substitute coarser screens for those formerly in use—and from the pulp to separate the coarse product and to grind it in tube mills or pans. There have been no notable contributions to our knowledge this year as to which is the best appliance for this purpose. Tube mills seem to be generally employed, except at Kalgoorlie, where the roasted ore can be efficiently ground to the required fineness in pans. Not that pans are not effective on raw ore. On the contrary, the Ivanhoe results show very good work, indeed, and it is to be much regretted that the comparative tests of pans and tube mills made at that mine were carried out in such a way as to cause the results to be worse than useless. But it is a matter for serious thought that the Ivanhoe costs for regrinding sands

should be only 8½d. per ton treated, when those of the Oroya-Brownhill, Lake View and other companies are double that figure. It cannot well be a matter of pans only, or one would have heard of similar low results from the other companies exclusively employing pans (at the Oroya and Lake View pans are used in conjunction with tube mills). Is it that the Ivanhoe ore is softer, or that the final product is not slimed so fine? The management has recently been very keen on reducing residues—and has succeeded very materially in doing so—and one would therefore expect that it would not overlook the gain from crushing as fine as its neighbors. One therefore assumes that the crushing is as fine; but anyhow the fact remains that the Ivanhoe sands-grinding cost is less than half that of neighboring wet-crushing companies of the same group. We still need that careful comparative test of pans and tube mills which has so often been asked and which we have hoped would be undertaken by Messrs. Klug or Denny (Johannesburg). Indeed, I have understood that the latter was making such tests, but nothing seems to have come of them.

Meanwhile, I am informed that such a test has been made at Broken Hill with a pan, a wet-grinding ball mill, a disk grinder, and a tube mill, and that the tube mill has proved itself the best of the lot, but in the absence of details I can only take this statement as the expression of individual opinion. Even if we assume tube mills to be the best slimers, there is still the question of stage-grinding, and it is by no means certain that the preliminary crushing of the coarser particles could be effected more economically in pans.

The best pan figures have been given above. The following tube-mill figures will probably prove of interest: African costs have been reduced from 8½d. per ton ground, down to 5½d., of which 1.89d. is for power, 0.7d. for pebbles and 0.93d. for liners. Mr. Leupold states that silex linings cost only half as much and last two and a half times as long as iron liners. The Consolidated Gold Fields made tests on manganese-steel liners as against silex, with the result that the former was shown to be more expensive and the grinding nothing like as good, owing to the slipping of the pebbles caused by the polished surface of the manganese-steel. At El Oro a 20-ft. mill with 7½ tons of pebbles grinds 125 tons of sands per day, using 6 h.p., to such a fineness that 90 per cent. will pass a 100-mesh and 50 per cent. a 200-mesh screen. At Waihi an 18-ft. mill grinds 77 tons of 20-mesh sand a day so that 93 per cent. will pass 150-mesh, using 37½ h.p. The Waihi is notorious for the hardness of its material, which ball mills fail to grind satisfactorily. Its tube mills do not appear to be run to their utmost capacity, but the figures are interesting, as an example in practice of direct tube-mill work—i.e., without returns. The above figures of over two tons slimed per horse-power at first sight show a less duty than that mentioned in my article last year, viz., four tons per horse-power

in Africa, but it must be remembered that the African crushing was much coarser, they having at that time adopted a standard of under 2 per cent. retained on 60-mesh.

But apart from the question of comparative cost of work done, tube mills have very fully shown their economical advantage as well as their capacity as regrinders. Reference was made last year to the huge additional net profit of £65,000 a year, accruing to the El Oro company from this source. At Waihi three 18-ft. tube mills and 90 stamps have increased the output from 2.89 tons per stamp per day to four tons, and have reduced the assay of the residues by 50 per cent. They have improved the amalgamation returns by from 5 to 7 per cent., and have effected a 75 per cent. saving in screening. It is found also that since the adoption of tube mills the slimes are more permeable and the filter-press charges take less time for treatment, so that the same presses can now deal with 30 per cent. more slime than before. The new Barry tube-mill linings made on the spot seem to be most effective, and to result in a great saving of cost over the imported silex. One set of linings is stated to last six months; a new set is always kept ready at hand and can be replaced in two days. Mr. Barry makes a great point of having his linings as rough as possible, so as to give a grip to the pebbles, and has modified his doors also to effect this.

At Johannesburg, Mr. Dowling emphasizes the necessity for the complete separation of the sands to be re-ground from the slimes in the mill pulp, and for keeping the proportion of water to sand at as low a figure as possible. He shows at the Robinson Deep a 0.48 dwt. lower residue (formerly 1.26 dwt., now after tube-mill treatment 0.78 dwt.) and an increased tonnage of 2300 tons per month. The Eckstein mines show in addition to the increased output resulting from the tube mills an additional profit of from 1s. to 1s. 6d. per ton (1s. at the Robinson, 1s. 4d. at the Ferriera) resulting from the increased extractions, which have reduced the assay of the residues by about 0.5 dwt. There are now 58 tube mills at work on the Rand—and more on order—at 24 different mines, and these 58 tube mills, working in the 24 mines only, have increased the amalgamation returns of the whole of the Rand by 3 per cent. At present the Knight's Deep appears to have the highest monthly output per stamp —for complete battery—this being 7.68 tons per stamp, since reduced by the substitution of different screening to 6.68 tons. In West Australia the Oroya-Brownhill outputs 7¼ tons per stamp through 10-mesh screening, and the Great Fingall (1150-lb. stamps) has an average duty of seven tons per 24 hours through 12-mesh screening, and the Sons of Gwalia at Leonora (1000-lb. stamps) crushes 6.68 tons per head per day through 20-mesh screening.

It is estimated that in addition to the above gains resulting in Africa

from the use of tube mills there is a saving of 30 per cent. in equipment, arising from the adoption of tube mills in place of obtaining the same output by the addition of the increased number of stamps necessary (see *South African Mines*, Oct. 27, 1906).

Amalgamation.—The increased amalgamation returns resulting from the use of tube mills have beeen referred to above. At Waihi amalgamation is carried out on plates set in a large building right away from the mill, sands and slimes being treated separately. There is much to be said for this arrangement. Companies circulating cyanide solutions through the mill find a scouring of the copper plates by the cyanide solutions. Denny, at the Meyer & Charlton, proposed to minimize this action by adopting shorter plates, but probably the better method would be to exclude copper plates from the amalgamation arrangements. At Lake View (Kalgoorlie), for instance, plates are discarded and amalgamation takes place in pans. Fifty-one per cent. of the value of the concentrates is there recovered by amalgamation and 46 per cent. by subsequent cyanidation.

Roasting.—Edwards has increased the capacity of his recent furnaces by a parallel system of rabbles, which is stated to promote inter-rabbling. Klug has adopted this for the Holthoff-Wethey furnaces at the Perseverance, and claims high results. Merton has also modified his furnace by making it a 5-hearth and also of greater hearth area; but, except by the Great Boulder Proprietary Company with its roasting cost of 2s. 3d. per ton (with its Merton furnaces) for roasting ore going 4 to 5 per cent. sulphur down to 0.07 per cent. sulphur as sulphide, the record of the South Kalgurli type of Merton furnaces does not yet appear to have been improved on, either for cost (2s. 6d. per ton roasted), output (32 tons per diem for small sized furnace, requiring 1½ h.p.,) or efficiency (3.1 per cent. of sulphur down to 0.01 per cent. of sulphur as sulphide; one ton of green wood roasting 11 tons of ore). With regard to the tip, referred to in my article of a year ago, of adding say 2 lb. of lead acetate per agitator charge (40 to 60 tons) of roasted ore, Mr. Hamilton of the Great Boulder finds that the use of lead salts appears to result in an increased consumption of zinc in the precipitation boxes, as against which he benefits by a saving of roasting fuel, an increased tonnage through the mill for the same labor, a reduction in the assay of the residues, and a general feeling of confidence in ability quickly to set foul roasts all right in spite of fluctuations in natural draft.

Concentration.—There appears to have been no great advance made in 1906 in the concentration of gold ores. We are still faced with the difficulty that an attempt to concentrate out refractory particles from an ore usually leaves tailings practically as refractory as the original ore.

It was hoped that fine sliming and the Wilfley slimer might overcome this

difficulty, but the latter machine is by no means perfect and maintenance is a most formidable item. So far the old canvas tables or frames appear to give the best results, but the cost of washing down is very heavy indeed —amounting in certain cases to 30 or 40s. per ton of concentrates produced. A continuous rough-rubber or canvas-belt table, somewhat on the Luhrig or Buss system, might solve the difficulty, though the capacity per unit would necessarily be small; or the old treatment table could have a traveling system of washing pipes worked by a water balance, and thus avoid the expense of so much boy-labor.

Treatment of Difficult Ores.—Cupriferous tailings, with copper not exceeding 0.5 per cent., are mainly successfully treated by leaching out the copper with dilute sulphuric acid and precipitating on scrap iron as described by W. S. Brown in his paper in the *Transactions* of the Institution of Mining and Metallurgy. With ores containing more copper—but not rich enough to smelt—a preliminary roasting of the ore has been found to improve matters. The addition of ammonia to the cyanide solution has been previously suggested and carried out in practice.

Auriferous antimonial concentrates, containing say 20 per cent. of antimony and arsenic, have been successfully treated by an ordinary careful roast of the ore previously mixed with from 2 to 5 per cent. of charcoal or coal, followed by a hot acid wash of dilute hydrochloric acid (obtainable by exposing old chlorine solutions to direct sunlight), and then lixiviating with cyanide solution or chlorine water.

A variation successfully employed is to roast as above, add salt at the end of the roast and chlorinate. Both methods have yielded 90 per cent. extractions, but have the drawback of requiring very careful roasting.

Slimes Treatment.—The Messrs. Denny have made a considerable sensation at Johannesburg by the running of their new plants at that center. In these they adopted the West Australian method of circulating dilute cyanide solutions through the mill, and filter-pressing the slimes in Dehne hydraulic-closed filter-presses. They claim a recovery of over 94 per cent. at a treatment cost, including cyanide, filter-pressing, and disposal of residues (5d.) of 1s. 10d. per ton. As this carrying into practice of their proposed treatment scheme was not effected without many prophecies of failure, and as the cost of the installation necessary appears to be less than half that of the method of treatment locally in vogue, it is natural that their work should have received much attention; and as their figures do not seem to be seriously contested, the Messrs. Denny certainly look like scoring heavily again as the result of their enterprise and foresight—just as they did previously by the introduction of tube mills on the Rand.

But perhaps the two greatest successes in slimes treatment—as being a real advance on established filter-press practice—are the respective methods evolved by Ridgway at the Boulder and by Barry at Waihi. Any-

one reading R. Gilman Brown's article in *Min. and Sci. Press* of Sept. 8, 1906, on the Moore filter must have noticed the very sloppy, badly arranged frames shown in the illustration. One can understand, perhaps, from these illustrations one of the reasons for the small success attending the Moore filter in America. At Waihi, Barry has a frame (see London *Min. Journ.*, Sept. 30, 1905) which is much more effective than that shown in the illustration referred to above, and he has successfully treated many thousands of tons by his method of open-framed atmospheric filtration; but the drawback to this—as to the Moore basket filter—is the necessity of having men constantly in attendance to clean the frames.

Butters seems to have been alive to the defects of the Moore frame and has got out one which he appears to be working with considerable success at Virginia City, as does also Bosqui, at Tonopah. Butters has apparently, however, allied himself to the Cassel inclosed type of press, which looks similar to that referred to in a previous paper as being experimented with at Johannesburg, apparently unsuccessfully.

The Ridgway machine has now been in use for a year at the Great Boulder, and appears to have treated some thousands of tons of slime quite automatically and without any continuous supervision, and Mr. Hamilton, who has been testing it alongside his filter-press installation, is so satisfied with the results that he is laying down a plant to treat 500 tons a day, the largest slimes plant in the southern hemisphere. The principle of the machine is a number of horizontal plates revolving around a central vertical post, which is really a tube, and to which pipes communicating with each plate are attached. Each plate has an under filtering surface, and dips through a portion of its revolution into the slimes pulp. A cake is formed by the application of a vacuum, and the plate in the course of its journey around its axis finds itself next in a water bath, where it remains sufficiently long for thorough washing. The cake is then automatically dislodged into the residues deposit system. This machine is certainly cheaper than filter-presses, both in first cost and in operation, and a plant to do the duty of an African decantation plant should not only yield higher recoveries at a lower working cost, but also should be installed for less than half of the expense of the existing system.

Of the other methods referred to in last year's article but little has been heard. Merrill appears to be still at work on his hydraulic emptying filter-press, but details of successful work have not yet been communicated.

Argall has a fixed frame and movable tank, which probably he will install in his new plant at Colorado City for the treatment of Cripple Creek ore.

In West Australia one of the groups had a method of upward percolation of the solutions through the slime pulp during agitation, but this method does not appear to have been attended with any great success. Generally

speaking it looks as if some direct method of automatic atmospheric filtration would displace both decantation and Dehne filter-pressing, although at the outset it will probably require a higher degree of intelligence for its effective and successful working.

Tailings Re-treatment.—We have heard but little of the Stark process during 1906. It seems to have been most profitably applied at the Crown Reef dump. Is it that there is any special characteristic feature of this dump which makes it exceptionally suitable for the Stark process?

Mercuric Cyanide.—A reference to a test by Butters, on El Oro ore, of mercury salt added to cyanide solution recalls the investigation made by the Cassel company (the owner of the cyanide patents) in 1895 into the use of this salt. It obtained on some ores 3½ per cent. greater extraction of the gold and 3 per cent. greater extraction of the silver, with a lower consumption of cyanide, but the tests were not continued as the improved extractions seemed insufficiently encouraging in view of the additional expense of the added salt, but Butters' tests have renewed interest in the subject, and we hope for fuller data.

West Australian Costs.—Tube milling, fine grinding, milling and roasting costs have been given above. The following statement of West Australian costs will show the progress still being made on this field by comparison with those in preceding papers, as well as with those obtaining elsewhere.

Rock-Breaking—Lake View, 1.71d.; Ivanhoe, 1.87d.; South Kalgurli (dry), 3.39d.

Milling—Lake View, 1s. 9d.; Ivanhoe, 1s. 9d.; South Kalgurli (dry), 2s. 11d.

Concentrating—Lake View, 6s. 7d. per ton concentrated, 9d. per ton milled; Ivanhoe, 8s. per ton concentrated, 10d. per ton milled.

Roasting—Lake View, 3s. 10d. per ton roasted (concentrates only); Ivanhoe, 5s. 5d. per ton roasted (concentrates only); South Kalgurli, 2s. 6d. per ton roasted (all the ore); Great Boulder Proprietary, 2s. 4d. per ton (roasted all the ore).

Fine-Grinding Sands—Lake View, (pans and tube mills) 1s. 10d. per ton ground; Ivanhoe (pans only) 8d. per ton ground; South Kalgurli, (pans only) 1s. 3d. per ton ground.

Cyaniding by Agitation—Lake View, 3s. 1d. (includes 1s. 4d. for KCy and 1s. 2d. for BrCy); Ivanhoe, 4s. 1d. (includes 11d. KCy, 2s. 6d. BrCy, and royalty 2d.); South Kalgurli, 1s. 4d. (includes 7d. KCy.)

From the above it appears that the agitation treatment, less cyanide and bromo-cyanide, but including power, lime, labor and supplies, costs roughly 7d. per ton. (Ivanhoe 6d.).

Cyaniding by Percolation—Ivanhoe, 2s. 2d. (including 9d. for KCy and lime); Great Fingall, 11¼d. (including 6d. for KCy and lime).

Filter-Pressing—Lake View, 1s. 7d.; Ivanhoe, 1s. 6d.; South Kalgurli, 1s. 6d. per ton filter-pressed. (The cost of filter-cloths at the Lake View consols was under ½d. per ton pressed for every month in 1906, save one, when it was under ¾d.)

Total Treatment Costs—Ivanhoe, 9s. 0d.; South Kalgurli, 11s. 3d; Great Boulder Proprietary, 11s. 6d.; Great Fingall, 6s. 11d.; Sons of Gwalia, 5s. 2d. per ton treated.

General.—The battle of the processes in Kalgoorlie is now over. It is admitted that the all-roasting process gives the most profitable extractions, but the good fight made by the wet-crushing bromo-cyanide section, and especially by the Ivanhoe, has been of the greatest service to the industry, and the Ivanhoe's costs are such as to reflect very great credit on the management and the staff; but the fine showing made by the South Kalgurli, the Great Boulder, the Kalgurli, and other companies, appears now to have convinced even the former bromo-cyanide advocates.

Cyanidation in the United States and Mexico.

Charles H. Fulton.

The great advance in cyanidation in 1906 was in slime treatment, which, in view of the increased tendency to fine crushing everywhere manifested, year by year, assumes greater importance. The efforts of 1906 were attended with much success and slime treatment now is no longer the bugbear of the mill-man. The development of the Butters-Cassell filter during the year; certain improvements in the Moore process; and the installation on a large scale of the Merrill press are the noteworthy developments in this department. The decantation process while still extensively used, especially in Mexico, is unquestionably passing out of use. For fine crushing, tube mills have found extended use in this country during the year, and they are becoming recognized as the machine pre-eminent for this purpose. The questions of tube mill lining; pebbles for grinding; thickness of pulp fed, etc., are being freely discussed by American metallurgists. As in other countries using the tube mill, wooden blocks, cast iron and wrought iron linings, native pebbles, etc., are being and have been used, but the general tendency is to come down finally to imported Greenland or Danish pebbles, no matter what the local cost, and imported silex lining. The Abbé and the Allis-Chalmers mills seem to be the favorites in the United States, while Krupp mills are largely used in Mexico. Certain plants like El Oro, Mexico, crush all of their product to finer than 150-mesh. In this country the Standard Consolidated at Bodie slimes practically all of its product and most other new mills are designed with regrinding machinery. Bryan mills are used to some extent for this purpose, but only for medium regrinding, and do not compete with tube mills for sliming.

Wherever the nature of the ore permits it, crushing in cyanide solution

is finding extensive application, as at the Standard Consolidated in California; the Liberty Bell, Colorado; the Dolores mill, Mexico, and in the Black Hills. Certain metallurgists do not favor this as it tends to interfere with perfect classification later on, but its advantages in cases where it can be applied are so great, that this disadvantage can be put up with. Weak cyanide solutions in the battery do not interfere seriously with amalgamation. Wherever sands and slimes are made the tendency is to effect as close a separation of these two products as possible, for coarse material and even fine sand cause more or less trouble in the filtering processes. Aside from the usual type of cone classifier, the Dorr mechanical scraping classifier is finding extensive application. Zinc dust precipitation is employed at some of the larger plants, e.g. the Homestake, for which it has undoubted advantages. It is not generally in favor and many failures seem to be recorded, but these are due to incomplete knowledge of the requirements of the process rather than to the process itself. It is much more difficult to carry out than ordinary zinc shaving precipitation, and in general does not seem so suitable for solutions weak in cyanide, of which many mills have large quantities to precipitate, owing to slime treatment. The difficulties, however, are being overcome, and this method of precipitation will see increased application during the next few years. No developments took place in electrical precipitation during 1906. Many metallurgists are at work on the question of the cyanidation of concentrates and are meeting with considerable success. A very long time of leaching with reasonably strong solutions, followed at certain stages by regrinding and releaching is the general method of treatment indicated. The process has been extended quite generally to the treatment of silver ores, especially in Mexico and more particularly at Guanajuato.

Slime Treatment.

The Butters Filter.[1]—In general principle this is similar to the Cassell and Moore filters. It consists of three essential parts,—viz., a series of filter frames; a containing box; and a system of pulp, solution, vacuum and water pumps, together with the necessary vats, pipes and valves. A single filter frame is usually 10 ft. long by 5 ft. deep, the upper member of which is a wooden bar, long enough for its edges to rest on the sides of the containing box, thus supporting the frame. The remaining three sides of the frame are made of iron pipe, the bottom one of which is perforated. The filtering leaf consists of a piece of cocoa matting exactly filling the frame, and covered on each side by sheets of canvas large enough to overlap the frame. Separating strips, which also act as stiffeners, are fastened vertically on each side of the frame. The frames are 4.5 in. apart from center to center. The containing box is a rectangular

[1]Pamphlet published by Chas. Butters' & Co., L't'd.

wooden or steel tank with a hopper bottom to facilitate the discharge of the pulp; and at the lower end of each hopper is a large gate valve for its quick discharge. A vacuum pump serves to remove cyanide solution and wash water from within the frames, and a centrifugal pump is used to transfer pulp, solution, water, etc., from the filter box and tanks. The operation of the process is as follows:

The slimes from the agitation vats are transferred to storage vats and from there to the filter boxes, which are filled to a point above the filtering surface. The vacuum pump is then set in operation and solution is withdrawn through the filters. Slimes pulp is pumped into the box at intervals to keep the slime above the filter surface. The cyanide solution, issuing from the frames, goes to clearing storage tanks and from there to precipitation. When a cake of slimes varying from 0.75 to 1 in. in thickness has accumulated on the filters, the remaining slimes in the box are withdrawn and pumped back to the slimes storage vat. A wash solution is then admitted to the filter box and drawn through the slimes cake to displace values remaining in it. Surplus wash solution is then withdrawn from the tank and pumped back to its storage. Clear wash water is applied in the same way. While transfers of slime and solution are being made and the slimes cake is exposed to the air, the vacuum in the filters is reduced to 5 in., sufficient to hold the slimes cake in place.

The valueless slimes cake is removed by shutting off the vacuum and permitting either air or water at slight pressure to enter the interior of the frames through the vacuum connection. This displaces the cakes which fall to the bottom of the box, from where they are washed out with water. The filter is now ready for a new charge. Under certain conditions of location, the filling and emptying of the box can be done partially by gravity. The time for the cycle of an operation varies very much with the character of the slime, and ranges approximately from two to four hours. With ordinary material and a cake averaging 0.75 in. thick, each frame will carry approximately 360 lb. of dry slime, giving a capacity per frame of from 1.25 to two tons in 24 hours. The slimes cake before detaching from the frame contains approximately 30 per cent. moisture. The thickness of the slimes cake is not uniform, it being thicker at the bottom than at the top, as the formation of the cake is automatic, more or less, the coarser (near the bottom) forming a thicker layer than the finer, thus maintaining an equal permeability of the leaf, essential for perfect washing. When the permeability of the leaves decreases, owing to the formation of carbonate of lime in the fibers of the canvas, arrangement is made for removing the frames from the boxes and immersing them in a 2 per cent. solution of hydrochloric acid. The frequency of this operation depends on the quantity of lime used in slime treatment and is necessary every two to nine months. The filter leaves

possess great durability and last two to three years. An operator's platform overlooks the whole installation, and all switches and valve stems to control the process are operated from that point.

The 200-ton plant at Virginia City, Nev., operates at 10c. per ton of slime.[1] At the Combination mill, Goldfield, Nev., 40 tons of slime per day are treated at a cost of 45c. per ton. This could be reduced to 31c. if the plant were operated at its full capacity. At this mill 15 h.p. are consumed in slime treatment for the following purposes: (1) Driving a 4-in. Butters centrifugal pump. (2) Operating a 10x12 in. Gould vacuum pump. (3) Operating a 2-in. centrifugal pump for raising filtered solution to the clarifying press. (4) Operating a 2-in. centrifugal pump for returning the slimes overflow from the leaching vats to the slimes settlers. (5) Operating stirring mechanism in two slime reservoirs, 14 ft. in diameter.

The power may be distributed as follows: (1) For actual operation of filter (56 tons per day), 9 h.p.; (2) agitating slime pulp, 3 h.p.; (3) uses not connected with filter, 3 h.p. Power costs $11.25 per h.p. per month.

The cost of the slime treatment proper is distributed as follows: three men at $4 per day, 30c.; 12 h.p. at $11.25 per h.p. per month, 11c.; Lubricants and incidentals, 4c.; total 45c. per ton.

The time of a cycle of operation is 3 h. 20 min. There are 28 frames, 5x10 ft., set 4.5 in. apart in a box 10 ft. square with a pointed bottom inclined at an angle of 50 deg.[2] The slimes pulp is introduced at the point of the box and a vacuum of about 22 in. of mercury maintained for about 20 min., during which time a cake builds up on each side of the frame from 0.75 to 1 in. thick. The surplus pulp is then withdrawn to the slime reservoir and the wash introduced, consisting of a weak solution of cyanide. When the cakes are washed, the weak solution is withdrawn and water is introduced, until the frames are completely immersed. After the solution has been displaced, water is pumped into the interior of the frames under a low head, causing the cakes to drop off clean into the bottom of the box, from whence they are finally removed by sluicing. The whole cycle of treatment requires 3.5 hours. The plant is operated by one man per shift. At the Butters cyanide plant, Virginia City, Nev., the slimes are collected in vats, thickened and the solution decanted.[3] From the settling vats the pulp goes to agitation vats equipped with paddle agitators as well as a centrifugal pump for agitation and aeration. Cyanide is added to bring up the solution strength to about 3 lb. KCN. The agitation is continued for from 48 to 60 hours, the solution then decanted and the sludge drawn to storage tanks whence it goes to the Butters filters. The two

[1] "Recent Improvements in Cyanide Process." F. L. Bosqui, *Min. and Sci. Press*, Dec. 15, 1906.
[2] *Eng. and Min. Journ.* Feb. 9, 1907.
[3] *Eng. and Min. Journ.* June 30, 1906.

filter boxes used are old electric precipitation boxes, 29 ft. long, 10 ft. wide and 4 ft. deep, with V-shaped bottoms. In each tank there are ninety-two 10x4 ft. frames (approximate) spaced 3.875 in. between centers. A vacuum of 22 to 23 in. mercury is used to build up the cakes to a thickness of 0.75 in. The cakes are discharged from the filters under water by admitting water inside the filter at low pressure. Other details of operation are similar to those already described for the Butters process. The capacity of the filter tanks is 150 tons per day. To date, about 1650 Butters frames have been installed chiefly in Mexico, Central America and Nevada.

The ore is crushed by 20 stamps in 2-lb. KCN solution, and the pulp is passed over amalgamated plates, with a grade of 2.5 in. per ft. This grade with the hardening effect of the cyanide solution decreased extraction by amalgamation from 60 to 50 per cent. Lime at the rate of 10 lb. per ton of ore is added at the crushers. After passing over vanners, the pulp is raised by Frenier pumps and conveyed to the slime plant 1800 ft. distant, where it is separated into sands and slimes by cone classifiers. The sands are reground, with the addition of some old tailings, in a 22x5 ft. Allis-Chalmers tube mill. The product of the mill is returned to the cones.

The slimes overflow from the cones goes to nine flat bottomed wooden settling vats of an aggregate capacity of 21,000 cu.ft. During filling the pulp is agitated, then allowed to settle and solution decanted to the upper surface of the slimes. Agitation is accomplished by drags actuated by a vertical shaft, driven by overhead gearing. The drags are short lengths of 30-lb. T-rail, attached to an 8x8 inch cross arm. The speed is 7 to 8 r.p.m. After decantation the pulp is made homogenous by stirring with the drags and then drawn off by centrifugal pumps to storage vats or the Moore filter tanks. The Moore filters are of medium weight canvas 5 ft. wide and 16 ft. long, the canvas being double sewed around three edges, while the fourth or long edge is bolted between strips of Oregon fir, 1.5x6 in. in cross section. In the bottom edge of each filter, or plate, is a 0.75 in. channel iron, which serves as a launder for the infiltered solution. The filters are stitched through both sides vertically, at 4 in. distances and in the compartments thus formed 1x0.25 in. strips are inserted to permit circulation. In each filter a 1-in. vertical suction pipe, flattened at the end, dips into the channel launder.[1]

The outer end of the suction pipe is connected by short lengths of hose to a 3-in. manifold to which the suction is employed. A "basket" of filters consists of 49 filter plates hung from a framework of steel I-beams. The distance between centers of frames is 4 in., so that the "basket" is approximately 16 ft. square, containing 7840 sq.ft. of filtering area. On top of the I-beam frame rests a 12x12 in. suction pump, driven by a 10 h.p.

[1]Mr. Nutter (*loc cit.*) states that this construction, i. e. the long filter and letting it hang, practically without the framework usually employed, gave rise to considerable trouble from twinning or coalescing of the slime cakes and that the filters continually broke loose from the spacing bars. The length of the individual frame is probably too great.

direct current motor. With a 20-in. vacuum, 3 h.p. are consumed, but in the early part of the cycle when filtering is fast, greater demands on power are made, and to equalize the load, the pump has heavy fly-wheels. The basket is hung by four endless, 1-in. pitch-chains to the overhead crane and is raised or lowered by a 10 h.p. motor through the medium of balanced worm gearing and differential chain wheels. The total maximum load of the full basket on the crane is 35 tons, and this is raised about 7 ft. in five minutes. This represents about 3 h.p., but as the motor consumes 8.4 h.p., the loss in this type (worm gearing, although balanced) is heavy. In general the worm gearing and differential chain-wheel system of lifting the basket is not so satisfactory as the method of hydraulic cranes, more usually employed.[1] The traversing device for the crane, to move the basket longitudinally from one treatment vat to the other, is a 0.625 in. plow steel cable run over three and four grooved sheaves after the manner of the Koepe hoist, the main sheave being driven by a worm gear from the main line shaft.

The filtering vats are round, flat bottomed vats (old sand tanks) 24 ft. in diameter and 7 ft. deep. They are provided with four-armed agitators revolving close to the bottom at 7 r.p.m. The shaft to which they are attached is driven by crown gearing placed below the vats. These agitators serve to keep the heavy particles of the pulp from settling. The slimes coat averages 0.75 in. in thickness. The general method of operation of the Moore filter has been described in former volumes of The Mineral Industry. After solution treatment about 0.7 tons of wash water are used to displace cyanide solution, and the cakes ready for the discharge tanks carry about 40 per cent. moisture. After the basket has been transferred to the discharge hopper, suction is continued to remove excess moisture and then the slimes cakes are caused to drop off by turning on air at a pressure of 35 lb. in successive blasts of a few seconds each. This causes about 85 per cent. of the weight of the cakes to drop off. Every alternate day the filters are cleaned by substituting water at 20 lb. pressure for air. The discharged sludge averages 68 per cent. solids and is sluiced from the hoppers into the waste flume with about one ton of water per ton of slime.

The time of a treatment cycle depends upon the nature of the slime and the thickness of the pulp fed. Pulp containing 20 per cent. solids will scarcely build up a 0.5 in. cake in the filters in 10 hours, while 40 per cent. solids in the pulp will give an inch coating in two or three hours. In general the pulp should be as thick as it can be made so that the period of the building of the cake will be as short as possible, and the maximum capacity will be obtained from the plant. The average thickness of pulp is usually 0.75 in. and the period of accretion 3.5 hours; washing and

[1] E. H. Nutter, "Cyanide Practice with the Moore Filter," *Min. and Sci. Press*, Dec. 15, 1906.

discharging also takes 3.5 hours, and allowing for delays, there are usually three complete cycles in 24 hours. This long cycle is due to the high percentage of clay in this ore. In one filter the average load handled is close to 18 tons of dry slime per cycle, so that the two units of the plant average 108 tons per day, or approximately 3000 tons per month. Six months is a fair average life of the filters, entailing a cost of 5c. per ton treated. From tailings (pulp from the regrinding, going to the Moore system) averaging \$8 per ton, the recovery has been 83 per cent. of the gold and silver, and of the gold alone close to 90 per cent. The installation of the Moore system with regrinding by tube mills, for the treatment of the stamp mill tailings, replacing percolation in vats, with a wasting, or rather storage of the unworkable slimes, showed a *net* gain in recovery on \$20 ore of \$1.90 per ton. The cost of cyaniding tailings with the Moore system was \$2.35 per ton, of which 31.4c. is chargeable to the Moore system proper and 57.2c. to regrinding; the remainder to chemicals and general expenses.

The Liberty Bell Mill.—W. E. Tracy[1] described the Moore process installation at the Liberty Bell mine in Colorado. The ore is crushed by eighty 850-lb. stamps through a 14-mesh, No. 22 wire screen in a solution containing 1 lb. KCN per ton, and passed over amalgamated plates. The battery pulp is classified by two Dorr scraping sizers, the sand product being fed to three Abbé tube mills. The slimes from the classifiers and the tube mill discharge are passed over a second set of amalgamated plates. The pulp from these plates is reclassified, the underflow being returned to the tube mills. The slimes finally pass to five settlers, slightly coned at the bottom, are settled for 14 hours and about one-half of the solution is decanted. The thickened pulp, consisting of two parts of solution to one of slime, is transferred by bottom discharge valves to six agitators. These are 17 ft. in diameter and 11 ft. deep to the beginning of the cone bottom, and hold 35 tons of dry slimes per charge. Cyanide is added to raise the solution strength to 1.5 lb. KCN per ton. The pulp is agitated for 12 hours and then is discharged by a centrifugal pump to a settling tank from which it is drawn to the loading tank of the Moore system, which has two loading and four displacing tanks.

There are four "baskets" each containing 666x8 ft. plates, giving 6336 ft. of filtering surface per basket. Twenty-one ounce duck is used on the frames. The baskets are raised and lowered by two 30-ton hydraulic cranes. A motor drive transfers the baskets among the tanks. The pulp in the loading tanks is kept from settling by air lift agitation. The baskets remain in the loading tank one hour, during which time a 0.875 in. cake is built up with a vacuum of from 17 to 19 in. In the displacing tank, a weak solution is first used, followed by water. The

[1] "Cyanide Practice at the Liberty Bell Mine, Telluride Colo." *Eng. and Min. Journ.* July 28, 1906.

average time of displacing valuable solution is about 1 h. 20 min. Displacing is continued until the discharge from the vacuum pump titrates 0.35 lb. KCN per ton. This leaves from 5 to 7c. dissolved gold and silver in the cake per ton of dry slimes and about 3c. per ton of KCN. When the solution issuing from the vacuum pumps shows dilution in KCN from the strength of ordinary weak solution, it is turned to the weak solution zinc boxes and then to waste. The cakes are discharged from the filters under water by 10 lb. hydraulic pressure. For clarifying strong solution from the Moore filters and decanted solution, an inverted Moore basket of 20 frames is placed in the gold solution tank and removes all traces of slime.

Relative Merits of Butters and Moore Filters.—F. L. Bosqui[1] and E. H. Nutter[2] discussed the relative merits of the Butters and Moore filters. Mr. Nutter states that when a large capacity (i.e., a rapid building up of cake and short cycle, generally dependent on the nature of the slimes) can be obtained with a small plant, the Butters system has certain advantages over the Moore system, but where the filtering plant has to be large the reverse is true. He claims that the Moore process for a given capacity is less costly to install and that the cost for operating power is less, and to prove the contention calculates the cost of the Butters system for the Liberty Bell plant, now using the Moore system. Mr. Bosqui states that the vital points at issue are not the less cost of installation or smaller consumption of power for the Moore process, but that operating costs and items of repair and maintenance will finally decide the question and then states the great advantage of the Butters process over the Moore process as follows:

A stationary filter, constituting one of the chief points of merit of the Butters filter. For in order to insure good work (i.e., a uniform deposition of cake and a clear filtrate) the frame must be straight and true, and the canvas free from weak places, ripped seams, etc. In the complete deposition and displacing the cake there is a period of suspension during which the vacuum is reduced just enough to hold the cake to the canvas, yet not enough to dry and crack the cake. The maintenance of the cake in the best condition for displacement by water is a delicate operation. Obviously, to give the best results the filter must not be stretched or strained; there must be a uniform and quiescent condition of the filtering area throughout. In the Moore system the frames are mechanically lifted from one tank to the other, and it is practically impossible to prevent disturbing the integrity of the cake, owing to the straining of the frame and filters. Minute cracks develop in the cake and portions drop off. The seams of the canvas, subjected to alternate tension and relaxation, weaken and finally

[1] "The Moore and Butters Filters," *Min. and Sci. Press*, Feb. 2, 1907.
[2] Loc. cit.

give way, causing leaks, interfering with filtration and leading to annoying interruptions and endless repairs. The Butters system at the Combination Mill at Goldfield has operated from the day of its installation with uninterrupted smoothness, and with no repairs except an occasional overhauling of the centrifugal slimes pump. The filters have not been taken out for repairs for nearly one year. There is no sloppiness in the operation, and no unsightly filter frames to move about. The system shows a mechanical orderliness and cleanliness which, whatever its other merits may be, appeal strongly to both operator and observer. The sum of the advantages seems to be on the side of the Butters-Cassell process, whichever point of view is taken.

It is to be noted that in all present filtering installations, whether of the Moore or Butters type, ample provision is made for the mechanical agitation of the slimes pulp, in order to insure the solution of gold and silver, before the pulp goes to the filter.

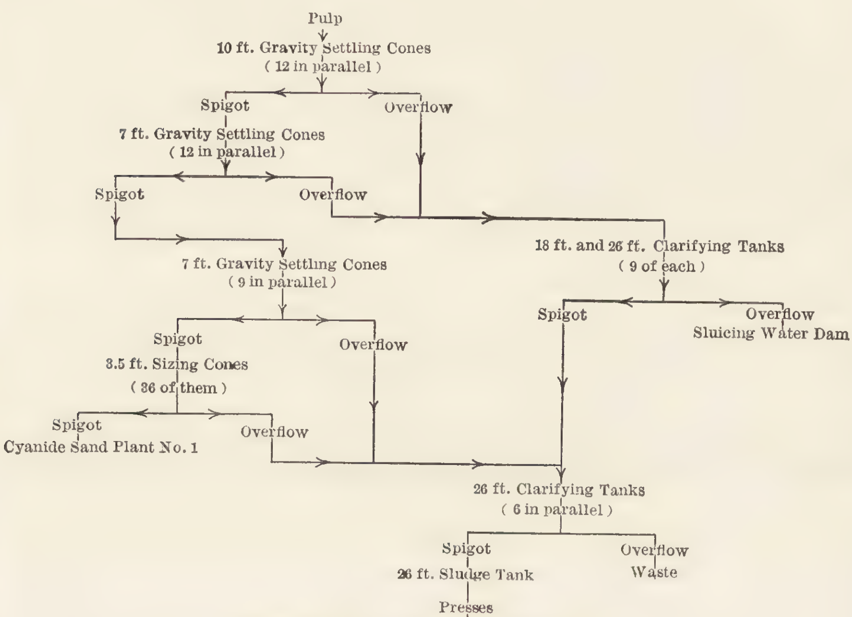
There is a close classification of sands and slimes, or if no sands are treated by percolation, a thorough regrinding of the sands to pass approximately 150-mesh. If considerable coarser material finds its way to the filters, trouble is almost certain to follow.

The Merrill Filter Press.[1]— Part of the new slime plant employing the Merrill automatic discharging filter press of the Homestake Mining Company at Deadwood, So. Dak., was put in operation late in 1906. This press was described in a general way in Vol. XIV of THE MINERAL INDUSTRY. Additional data are as follows: The plant contains 24 presses, each of 92 frames, 4x6 ft. in size (approximate), making a cake 4 in. in thickness. The length of each press is 45 ft. and its weight 65 tons. It holds when filled 25 to 26 tons of dry slimes. The cost of plant was approximately $400,000 and cost of pipe line to convey pulp from Lead and from Central, $50,000. The sluice pipe, with its nozzles operating in the continuous channel in the lower part of the press, is actuated by a special differential cam so as to vary its speed during portions of its partial revolution, slowing up when diagonally opposite the corners of the cake, thus throwing more water to these parts and insuring their complete sluicing out. About four tons of water are required, at 50 lb. pressure, to sluice one ton of slimes. The pulp from the sludge tanks, charged to the presses at 30 lb. pressure, by gravity consists of one part solid to three parts water. Lime in the form of an emulsion is added as the pulp leaves the sludge. tanks. All of the pulp fed passes a 200-mesh screen. It takes one hour to fill a press.

After filling, the treatment with cyanide solution, forcing air through cakes, washing and drying take 5.5 to six hours. This is divided into 5.2

[1]For complete authorized description of this installation, see article by Mark Ehle, Jr., in *Mines and Minerals*, March, 1907.

hours solution contact, 2.5 hours air, and 0.5 hour wash water. The exact time and succession of solutions, strong and weak, and air are matters of local interest as applying to Homestake conditions only. When the cakes are ready for sluicing they contain about 20 per cent. moisture. The solution pressure used is 15 lb. and air pressure 20 lb.; the washing pressure employed is 25 lb. The amount of solution handled per ton of slime is 0.6 ton, of which only one-half needs to be precipitated. It takes 50 minutes to discharge a press. The complete cycle of operations takes 7.5 to eight hours, or three charges (totaling 75 tons) per press per day. The plant has a total capacity of 1800 tons per 24 hours and takes the slimes from all the Homestake mills. One man takes care of four presses, and the labor cost per ton of slimes is very small.



SCHEME OF PULP CLASSIFICATION AT LEAD.

All solutions coming from the presses are handled by two 8x6 triplex electric pumps, and the wash water by one 11x9 pump of the same type. The slimes going to the plant have a value of from 80c. to $1 per ton. On the basis of tests involving 6000 tons, the average extraction is placed at 81 per cent. The calculated cost of treatment is 25c. per ton. In the Merrill press all of the leaching and extraction by solution takes place in the press itself. Precipitation of solution is carried on by means of zinc dust in two presses of special design. The frames of these presses are triangular in shape (equilateral). The zinc dust emulsion enters at a

channel in an upper corner, then descends in a duct in the interior of the frame to the bottom or apex of the triangle, from which it rises through the solution. The press also has other special features. Before the slimes sludge reaches the presses it undergoes a very extensive and close classification. This is essential in order to have the presses operate uniformly and successfully.

The scheme of classification of the pulp from the 640 stamps at Lead City is shown in the accompanying diagram. The classification for the Central stamps is similar.

The relative merits of the Merrill press and the vacuum or basket filter depend upon the nature of the slimes and local conditions. There is no question that certain slimes cannot be filter-pressed, but can be treated successfully by the basket or vacuum filter. On those slimes amenable to both processes, local conditions will probably become the decisive factors.

Cyanidation of Silver Ores.

The progress made in the treatment of dry ores, containing a large part of their value in silver, was rapid during 1906, especially in Mexico and more particularly at Guanajuato. The year at that camp saw the introduction of cyanidation for its ores, practically displacing the *patio* process and the smelting of the ores, except the high-grade sulphide concentrates. There are now in operation[1] or will be shortly, nine mills, dropping between 500 and 560 stamps. In general the method followed is to crush the ore by stamps through 30-mesh screens or finer, concentrate the pulp on Wilfley tables, separate sands from slimes, treat the sands by leaching and the slimes by agitation and decantation. The sands are treated from 10 to 15 days with a solution containing 6 to 8 lb. of KCN per ton, while the slimes are agitated with much weaker solution, approximately 1 lb. KCN.[2]

The Peregrina Mill under construction at Guanajuato was described by F. J. Hobson.[3] The ore is crushed by two Gates breakers and then distributed by belt conveyors to bins from which it is fed by 20 suspended challenge feeders to 100-1050-lb. stamps, which crush through a 40-mesh screen. The pulp goes over amalgamated plates to Wilfley tables. The tailings from the tables go to cone separators for separation into slimes and coarse sands which are reground in a 5x26 ft. tube mill. The reground product joins the overflow from the classifiers and the whole goes to a second series of cone classifiers, consisting of a large cone 8x10 ft., placed above four smaller cones, 5x5 ft. The overflow from the four small cones joins the overflow from the sand-collecting tanks,

[1] *Eng. and Min. Journ.* Oct. 13, 1906.
[2] F. Hobson, *Min. and Sci. Press*, Jan. 6, 1906.
[3] *Eng. and Min. Journ.* May 19, 1906.

provided with Butters distributors placed below the cones, and goes to the slimes plant. The steel sand tanks, 18 in number, 6x28 ft., are charged from the collecting vats by Blaisdell tank excavating and distributing machinery. The 16 steel slimes vats for decantation are 36x12 ft. in size, have peripheral overflow launders and are equipped with aero-mechanical agitators.

Precipitation is carried on by zinc shavings. All power in the mill is furnished by electric motors. The ore averages about $12 U. S. currency, of which 60 per cent. is gold. Of this gold, 40 per cent. is amalgamated on the plates, 10 per cent. is saved in concentrates as well as about 30 per cent. of the silver. The total extraction by the mill is about 94 per cent. of the value of the ore. It is essential for successful work that all sands pass a 100-mesh screen. The ore makes approximately 55 per cent. slimes and 45 per cent. sands. The power of the mill is as follows: rock breakers, two 30 h.p. motors; stamps, five 50 h.p.; tables, one 50 h.p.; tube mill, one 75 h.p.; agitating machinery, one 15 h.p.; Blaisdell machinery, 30 h.p.; pumps, one 75 h.p.; air compressor, 30 h.p.; total, 585 h.p.

New Milling Practice.

El Oro Mill.—T. A. Rickard[1] described the practice at the new 100-stamp mill of the El Oro company. The ore is crushed in water by 100-1180-℔. stamps dropping 6 in., 102 times per minute, with a 2.5 in. discharge and a capacity of 4.7 tons per stamp, and is passed over amalgamated plates. Woven brass wire, 35-mesh screens are used. The pulp goes to the first system of cone classifiers and spitzkästen, which separate coarse sands, to be reground in a 5x26 ft. Krupp tube mill. The product from this tube mill unites with the overflow from the classifiers and passes to a second system of spitzkästen, the overflow from which goes to the slimes agitators, while the underflow or spigot product goes to an elevator wheel, which raises the pulp to a cone classifier. The overflow of this classifier goes to the sand receiving tanks, while the underflow passes to two Krupp mills, 20x4 ft., and 24x5 ft. respectively. The product from these mills passes again to the second set of spitzkästen for reclassification. The aim of the crushing and classification is to make sand as near 150-mesh as possible, none to be coarser, and a product consisting of sand and slime, all finer than 200-mesh. The overflow from the sand receiving tanks goes to a third series of spitzkästen, the underflow from which goes to the elevator wheel, while the overflow joins the slimes.

In respect to classification the El Oro mill has no peer. The sands constituting about 25 per cent. of the mill product are leached in 12

[1] "Cyanide Practice at El Oro," *Min. and Sci. Press*, Sept. 29, 1906 also "Cyaniding at El Oro," Caetani and Burt, *Trans.* A. I. M. E., Feb. 1906.

40x6 ft. steel tanks after being brought from nine sand receiving tanks. These receiving tanks are emptied by Blaisdell excavating machinery, the pulp being conveyed by belts to the treatment tanks to which it is distributed by mechanical distributors. The sand is treated by 10 washes of solution alternating in strength between 2 and 4 ℔. KCN per ton, introduced six hours apart. These are followed by 30 weak solutions of approximately 0.6 to 1 ℔. KCN per ton in strength, 13 tons of solution being used at intervals of four to six hours. This lengthy treatment is especially designed to extract silver. The tanks are emptied by Blaisdell excavators and the tailings taken by belt conveyors to the dump. The slime, consisting of 75 per cent. of the mill product, goes to a collecting vat from which the thickened pulp is drawn off to 12 steel slime tanks 34x12 ft. in size. These tanks are provided with mechanical revolving arm agitators. Cyanide solution containing 1 ℔. per ton KCN is added simultaneously with the charging of the pulp, as well as a certain amount of lead acetate solution. This salt when added in small amounts of cyanide solutions forms probably an alkaline plumbite (K_2PbO_2) by reaction with caustic alkali present in the solution. The most useful effect of this soluble lead salt is to precipitate any soluble sulphides present in solution, which are particularly apt to be formed during treatment of silver ores, and whose presence seriously retards the solution of silver and gold, and may even precipitate silver already dissolved. Lead compounds in cyanide solution also aid precipitation according to well known principles. The slime (a charge consists of 60 tons) is mixed with 2.5 tons of solution (1 ℔. KCN to the ton) per ton of slime and agitated for six hours. Settling and decantation last for eight hours. The settling of the pulp is aided by lime, added in the tube mills. After decantation, the vat is filled with KCN solution containing 0.6 ℔. KCN per ton and agitated for 1.5 hours. This is repeated three times. The vat is then filled with solution for the fifth time and the pulp transferred by a centrifugal pump to one of six deep settling vats, holding 450 tons. These take successive charges from the agitating vats until full. The clarified solution from these vats is too low in gold and silver for individual precipitation, but is used again directly as first wash in the treatment vats. The slimes go to waste. For the course of solutions, dewatering of sands and slimes, and question of solution accumulation, reference must be made to the original papers.

Precipitation is carried on in zinc boxes, by zinc shavings, strong, weak and medium solution boxes being employed. A cyanide drip of 2.5 per cent. cyanide solution is added at the head of the weak boxes to aid precipitation. In the clean-up the zinc from the boxes is sent through launders and carefully screened while it is being washed with water. It is then pumped to two filter presses. The cakes in the press are washed and then dried by steam. When discharged from the press they are mixed

with the requisite fluxes and briqueted and the briquets are dried and melted in crucibles. Acid treatment is not used. Other details of precipitate treatment are lacking. The sands treated in the mill have an assay approximately of $8.28 in gold and $1.59 in silver; the slimes $7.68 in gold and $1.64 in silver. The extraction on sands is gold 83.94 per cent., silver 65.41 per cent; on slimes, gold 92.45 per cent., silver, 78.05 per cent. The average total extraction is: gold, 90.28 per cent.; silver, 75.08 per cent. Sodium cyanide is used in the mill but all figures used are expressed in the equivalent of KCN.

Combination Mill, Goldfield, Nev.—Francis L. Bosqui[1] described the plant of the Combination mine at Goldfield, Nev. The present plant to treat this difficult ore was decided on after elaborate experiment, a summary of which is given in the first part of the paper. The mill has 20 1350-℔. stamps (100, drops 6 in. per min.) crushing through 12-mesh wire screens. Both inside (chuck block) and outside amalgamation are used. There are three outside plates arranged in step form, 53 in. wide and totaling 12.5 sq.ft. These plates are on a movable platform to facilitate cleaning up the batteries. The stamp duty, owing to the extreme hardness of the ore is only 3.5 tons per stamp. The mill to this point treats both oxidized and sulphide ores alike, 10 stamps being devoted to each. Beyond this point the treatment differs for the two classes of ores.

Treatment of the Oxidized Ores.—The pulp from the plates goes to small cone classifiers which separate it into coarse sand and fine sand and slime. The former (20 tons per day) goes to a 5-ft. Bryan mill running at half speed, which crushes the sand through a No. 9 slotted screen, equivalent to a 40-mesh. The product from the Bryan mill unites with the fine sand and slime, all going to two 6-ft. Frue vanners and two 6-ft. Triumph vanners. The tailings from these concentrators are raised by two 54-in. Frenier sand pumps to two sets of cone classifiers, modifications of the Merrill type at the Homestake mills. The top cone serves to equalize the intermittent discharge from the Frenier pumps and feeds the smaller lower cones. The underflow from these cones goes to the sand settling vats and the overflow to a larger cone where a closer separation of fine sand and slime is made. The sands from this cone also go to the sand settling vats, while the slimes go to the slimes settlers. The sand settling vats are filled by pipe distributors, the overflow being withdrawn through slimes gates and sent to the slimes settlers for clarification. When the sand settler is filled the surplus moisture is removed by a Gould vacuum pump and the charge is shovelled into treatment vats below. It is there given an eight day treatment with 2-℔. and 4-℔. KCN solutions. The slimes are delivered to two conical-bottom settlers, each of which is alter-

[1]*Min. and Sci. Press*, Oct. 6 and 13 1906; also M. R. Lamb *Eng. and Min. Journ.* June 30, 1906.

nately allowed to fill and overflow for 12 hours and then allowed 12 hours for settling. The requisite lime, 5 to 10 lb. per ton, is added at the Sturtevant roll-jaw crusher, crushing ahead of the stamps. After settling and decanting off water, KCN is added to the pulp to give a solution strength of 3 to 4 lb. KCN per ton, also giving a final pulp consistency of three parts of solution to one of slime. The pulp is then transfered to a steep-cone steel agitation vat, by a centrifugal pump. The pulp is there agitated by means of the pump and a mechanical stirrer. The agitation lasts from 12 to 18 hours, after which the pulp is withdrawn to the slimes reservoir from which the Butters-Cassell filters are charged.

Treatment of the Sulphide Ores.—The pulp from the plates goes to cone classifiers and the underflow, consisting of coarse sands and sulphides, goes to a Wilfley concentrator. The tailings from the Wilfley go to a 5-ft. Bryan mill which crushes through a No. 9 slotted screen. The product of the mill joins the overflow from the classifiers and passes to four 6-ft. Frue vanners, which remove a large quantity of fine sulphides. The tailings from the vanners go (by a sand pump) to a classifier, which acts both as a classifier and dewatering device. The coarse sand passes to a 4x12 ft. Abbé tube mill, while the slimes overflow joins the tube mill product and goes to three Wilfley slime tables of the latest pattern. The tailings from these go to the system of cone classifiers (already mentioned) treating the oxidized pulp, and pass to the cyanide plant. Of the final product of the slime table 87 per cent. passes a 200-mesh screen and the three stages of reduction show a saving by concentration of over 80 per cent. of the contained gold.

Precipitation.—This is carried on by zinc shavings. The solutions to be precipitated are unusually rich, reaching as high as 1 oz. gold per ton, and owing to the absence of silver require very large zinc surface for precipitation. The precipitates are refined by sulphuric acid. The average consumption of chemicals per ton of ore treated for five months in 1906 was KCN, 1.27 lb.; zinc, 0.6 lb; lime, 11 lb. In May, 1842 tons of ore were treated at a cost of $5.83 per ton—a very good record when Tonopah conditions are considered. The total recovery in the mill varies between 90 and 93 per cent.

The Butters Cyanide Plant at Virginia City, Nev.[1]—This plant is designed to treat accumulated tailings from the old Comstock pan amalgamation mills, averaging $1.40 in gold, 4 oz. in silver and 2 to 3 lb. of copper as sulphate, per ton. The plant is now chiefly devoted to the treatment of silicious Tonopah ores, and as it is not provided with crushing machinery, these ores are crushed in two Virginia City mills and the tailings from tables in those mills are sent in a flume to the Butters mill for cyanidation.

[1]Claude T. Rice, *Eng. and Min. Journ.*, Feb. 9, 1907.

The Virginia City plants use Kinkead mills for crushing. The ore is crushed through 30-mesh screens, and sent to classifiers. The sands are concentrated on vanners and the tailings from those are recrushed through 40-mesh screens in Kinkead mills. The product from these mills is reclassified and the sands concentrated on vanners. The overflows from all classifiers join the tailings from the last vanners and go by flume to the Butters plant. There the pulp is fed to collecting tanks by Butters distributors, the slimes overflow being caught in slimes tanks, after being classified in spitzkästen. The sands are discharged from the collecting tanks by Blaisdell machinery and charged into the leaching tanks, the transfer of 120 tons requiring two to three hours. The sands undergo double treatment in two sets of leaching vats. The length of treatment is 12 to 14 days with a solution containing 6 lb. KCN per ton, long treatment being requisite on account of the high silver contents of the ores. The slimes from the collecting vats go to the agitation vats, after settling and decantation. Agitation is accomplished by means of paddle agitators and centrifugal pump. The slimes are agitated with a solution containing 3 lb. KCN per ton for 48 to 60 hours and then pass to the Butters filter already described. Precipitation is carried on by the Butters electrolytic method. The precipitation boxes are 29 ft. long, 10 ft. wide and 4 ft. deep, with V-shaped bottoms. Commercial tin plate is used for the cathode and peroxidized lead for the anode. There are 216 plates, 25x48 in size, spaced 6 in. apart in each box, arranged in 12 compartments. A current of 250 amperes at 25 volts is used, but details as to current density, etc., are lacking. (See THE MINERAL INDUSTRY, Vol. XIII.) The mill treats at present 90 to 100 tons of ore per day, although its capacity is 200 to 220 tons. All power in the mill is electric, 265 h.p. being used. A total of but eight men is required in the mill in 24 hours, due to the fact that most of the work is done by machinery.

GOLD DREDGING IN 1906.

BY J. P. HUTCHINS

Last year (1906) was unusually favorable to most metallurgical operations, but those having for their object the recovery of gold were in some ways seriously handicapped. While the producer of gold received no more for his product, he was compelled to pay more for machinery, labor, and for all supplies needed in the installation and operation of machinery. He had to wait tedious periods for machinery or renewals, all manufacturers being swamped with orders; he paid higher prices for his necessities than ever before. That gold dredging should flourish in such circumstances vouches for its prosperous condition. That operating cost was reduced when there were many features conducive to increased expenditure

bears testimony to the improvements in design, construction and operation of gold-dredging machinery.

Search for New Areas.—The great success of dredging, especially in California, stimulated the search for workable areas. Engineers have been scouring remote parts of the world and the dredging horizon was widened somewhat. The results, however, were hardly commensurate with the energy and capital expended, and there was considerable disappointment. Many countries credited with large total productions of placer gold, practically all of which has been won from dry deposits, have been thought to contain rich under-water alluvion and have been investigated, rather casually to be sure, with disappointing results by operators whose experience has been confined to California, where conditions are remarkably favorable; their unfamiliarity with Tropic and Arctic environments has resulted in areas being condemned hastily and probably unjustly. The characteristics of the Oroville field are said to be ideal. It must be remembered, however, that there has been some remarkably lucrative placer mining elsewhere in the world. The California dredgeman is prone to look for ideal conditions and to condemn areas not possessing similar features. It is extremely probable that, like the success achieved in the Klondike dredging fields, which were turned down by the California operators, similar success will follow in the Tropics. Dredging cost is extremely sensitive; it is affected particularly by environment, and any unfavorable conditions raise cost in a surprising manner.

Misdirected Exploitation.—As a result of the prosperity of dredging and of the recent strong general tendency toward mining investments, numerous dredging corporations have been found to exploit areas of dubious value. Some of these are entirely dishonest but have succeeded in disposing of their shares. Other companies have been promoted by ignorant but enthusiastic persons who have supposed that certain areas could be profitably worked. While they are honest in their belief, they are reprehensible in not ascertaining accurately the gold content and the characteristics of their holdings.

Following the great success of one dredging operation in Klondike, there is a condition approximating a boom in that territory and its influence is likely to boom other northern districts. Large purchases of mining ground in the Klondike by powerful interests have made for this same result, opening great opportunities for illegitimate promotion.

During 1906 there were several failures, all due to lack of sufficient preliminary investigation. Thus, attempts were made either to work undredgeable ground or to work dredgeable ground with unsuitable machinery. The choice of dredging machinery of design and construction proper for the area it is to work is a matter of prime importance. The press of business is so great that some of the dredge manufacturers are

prone to urge the purchase of "standard" dredges. While these dredges have been successful in circumstances similar to those in which they were evolved, there have been failures where they have been worked under different conditions. It is easy for the manufacturer to build machinery like others that he has made, since there is no need of changing drawings, patterns, etc. This feature has been to a degree responsible for the installation of unsuitable machinery.

Testing Dredging Ground.—The year served to emphasize the advantages of test pits of large diameter over holes of small diameter. When shafts are sunk, it permits not only a more accurate determination of the gold contents, but it allows, as well, a thorough inspection of the gravel section and an accurate investigation of all the characteristics which have such a potent influence upon the working of dredging machinery. Drilling machines, in areas whose characteristics are already known (by exploitation with dredging machinery), are entirely satisfactory. When an investigation is conducted in a new district, however, the characteristics of which are not known, it is much better to sink enough shafts to determine them, even though drilling machines are used in conjunction.

Prospecting cost did not vary materially during 1906. Shaft sinking cost, in general, from \$1 to \$8 per ft. Drilling cost about \$1 to \$2.50 per ft., though the more general use of larger and more powerful drilling machines resulted in a generally lower average cost. It is interesting to note that, under exceptionally favorable conditions, considerable drilling was done in California for less than \$1 per ft.; some of it for less than \$0.50 per ft.

Expansion of Dredging.—Although new dredges were constructed in 1906, the total number of dredges operating in the world remains about the same, namely, 500. This was due principally to the working out of holdings in New Zealand. Investigation proved the existence of new areas, and enlarged the boundaries of old districts. A considerable area was drilled in the Klondike and considerable ground, said to contain from \$0.40 to over \$2 per cu. yd., was proved. Dredges in the Klondike achieved the largest extraction per cubic yard; more than \$1 per cu. yd. was saved, out of considerable volumes. The lowering of cost widened the dredging horizon, and ground formerly thought unprofitable is now included.

South America is attracting much attention and this will have beneficial results. It is almost certain that such countries as Colombia, Brazil and Bolivia contain dredging ground. Russia and Siberia are known to contain large areas of auriferous gravel and some of it is now being dredged profitably, although in one instance it is costing about \$0.16 per cu. yd. British Guiana has an inefficient dredge which is said to have produced 20,000 oz. of gold in about nine months.

Northern South America, already credited with about 14 dredging fail-

ures, added another during 1906. A dredge installed on an affluent of the Magdalena river seems to be a complete failure. It is not certain whether mechanical features were entirely responsible, but a continuous series of breakdowns, such as would have meant disaster even in California, occurred when operation began. Thus Colombia unjustly got more bad repute, even though there are probably areas of rich gravel suitable for dredging on the Magdalena, Atrato and other rivers.

A dredge is in course of construction for working auriferous and diamondiferous material in the bed of the Jequitinhonha river in Brazil. It is expected to be operating in 1907, when its unusual features will be watched with great interest. Its essential difference from the ordinary dredge is a large dredging pump to take the material of a diameter between $\frac{1}{16}$ and 1 in., after its passage over gold-saving devices, through a pipe-line (sustained as in hydraulic harbor dredging), to a diamond-saving plant on shore, where jigs and other devices will save the diamonds.

Investigation of ground in Ecuador progresses actively and a dredge having a producer-gas engine was installed. No data as to its working are available.

The island of San Domingo is known to contain auriferous gravel and it is being investigated with the idea of installing gold dredges. Some of the Central American countries are being examined similarly. Some Mexican placers near Chihuahua, which were unsuccessfully worked with a steam shovel some years ago, are now being considered as possibly suitable for exploitation with floating dredges.

There is a class of deposits in the United States which have been worked unsuccessfully with floating dredges of old and inefficient types. They have been investigated during 1906 and are now considered as well suited for profitable exploitation with modern dredges of large capacity. There are such areas near Breckenridge, Col., and Idaho City, Idaho.

Dredging Procedure.—The variations in practice and design in 1906 were almost as wide as before. This was noticeable, particularly, in the gold-saving phase of dredging, and the remarkable difference between the practice of New Zealand and that of California remains as great as ever. It is difficult to account for this fact when it is remembered that the characteristics of the gold in the two countries are practically identical. Part of the variation in dredging practice arises from the circumstance, not generally understood, that it is a difficult matter *not* to save a large percentage of the gold in most placers—almost impossible to lose it, in other words—while, on the other hand, it is almost impossible to save the gold in some deposits, like those of Snake river, Idaho, when operation is conducted on a large scale. So some dredge operators strive for large capacity, working their machines at top speed, and save only that gold which is easily saved.

Fortunately this is a very large percentage—often over 90 per cent. of the total—though this is not generally acknowledged.

The conflict of practice in gold saving is due largely to ignorance of the gold content of dredge tailing. Any investigation in this connection is sure to have valuable results; as valuable as is the sampling of tailings in lode mining.

Recent Dredge Design.—There were no remarkable novelties in design and construction during 1906; there were some general improvements and dredging machinery is now heavier and more powerful than ever. The 5-cu. ft. bucket of 1900, weighed about 600 lb.; this bucket now weighs about 1500 lb. Comparisons of other parts indicate similar increase. Hull construction received more attention; the rotting and distortion of some of the older dredge hulls made it seem more likely that steel hulls will be used to prevent these difficulties. Steel hulls have not been used in the United States for gold dredges, and for no good reason, although more structural steel now enters into a gold dredge than ever. The tendency is to strengthen hulls in this way, for which there is a crying need, for some hulls have been greatly distorted. One of them, formerly with a 6-in. crown to the deck, now has a 6-in. cup, and the well is over 12 in. wider at the bow than originally built.

The large dredge was well tried in 1906. A 13-cu. ft., continuous-bucket dredge was at work near Folsom, Cal., in shallow free alluvion; this material is said to be excavated at the rate of 200,000 cu. yd. per month at a cost of 2.25c. per cu. yd. Other large dredges working in coarse, indurated gravel, demonstrated, better than before, that the concomitant weight of the large dredge is a great advantage in digging. There was a distinct retrogression at Oroville several years ago when a number of 3-cu. ft. dredges were installed; at present there is one 3-ft. dredge at work near a 5-ft. dredge in indurated material. Comparison of the operation of the two machines indicates most strongly the advantage of the larger and heavier dredge.

A dredge with 12½-cu.ft., open-connected buckets was installed in Montana in 1906. This dredge is electrically operated and is owned by a company which also works a 7-cu. ft. dredge and one with 10-cu. ft. buckets, both of which are steam driven. All three dredges work in gravel of great tenacity. The results indicate that the largest dredge is the cheapest to run per cubic yard, and that electricity is more satisfactory than steam, particularly for driving dredge buckets.

The centrifugal tailing elevator has held its own in the antipodes, but no dredges have used it in the United States. Considerable experimenting has been conducted by one of the large dredge factories with a similar device and it is possible that it will be used here in the future.

Clay-Bearing Gravel.—The problem of handling alluvion containing clay,

or lying on a clay bottom, received greater attention in 1906. It has been generally held that the proper way to treat such material is to macerate and disintegrate it as far as possible in revolving screens having spirals or projections to effect this result. Considerable effort has been expended in this direction with dubious success toward disintegrating really tenacious clay. Investigation will prove that really it is exceptional to find gold either in clay strata or on a clay bottom. It may occur directly in contact with them but seldom in them, since the conditions that made for the deposition of clay and tufaceous material are not those that favor the contemporaneous deposition of gold. However, gold is often found in sticky gravel, and it is fortunate that such gravel usually yields to treatment in a revolving screen with water jets. It is also fortunate that the really tough clays, almost without exception, contain no gold.

Much effort is being wasted in attempting to disintegrate tough clay, and, in fact, harm is often done in treating this material in such a way as to bring it in contact with gold, which may then be lost by adhesion to clay balls. Thus the revolving screen, where tough, barren clay occurs, is not so suitable as the shaking screen which does not roll the auriferous material around with the adhesive clay.

Some loss is occasioned through the picking up of gold by clay balls when clean-ups are so infrequent as to permit gold and amalgam to accumulate near the top of the riffles.

Winter Dredging.—During the past winter, operations at Ruby, Mont., demonstrated that work can be carried on even when temperatures less than —25 deg. F. are experienced for a time. This means that dredging can be carried on all the year in any part of the United States, by using modified design and manipulation as at Ruby or in the Klondike, where housing the belt stackers and having a boiler for heating the dredge permitted work after all other out-door mining had been stopped.

Steam-Shovel Dredging.—Within the last year a steam-shovel plant was installed at Atlin, British Columbia, for exploiting placer ground, and preparations were made for similar work in Klondike in the future. The Atlin plant has not been in operation long enough to form a positive conclusion. The outcome of such installations will be watched with considerable interest, for such plants can be successful only under exceptional conditions. Lack of mobility and the difficulties of tailing disposal are the more important of a number of obstacles.

Dredging Cost.—The cost of dredging did not vary materially during 1906. The cheapest work was done in California (said to be 2.25c. per cu. yd.); the cost in Montana was more than 10c. per cu. yd. In Klondike, in unfrozen ground, the cost was about 15c. per cu.yd. with a large modern dredge, and about 75c. per cu. yd. with a small, obsolete dredge in partially frozen ground, which required considerable thawing.

GRAPHITE.

By John T. Glidden.

The domestic production of crystalline graphite in 1906 showed but little change from the previous year. The production of artificial graphite, which in 1905, for the first time surpassed that of the natural, crystalline mineral, showed a further increase in 1906. The statistics of the industry are given in the subjoined table.

STATISTICS OF GRAPHITE IN THE UNITED STATES.

Year.	Refined Crystalline Graphite.						Amorphous Graphite. Production.		Artificial Graphite. Production.	
	Production.		Imports.		Consumption (c)					
	Pounds.	Value (b)	Pounds.	Value.	Pounds.	Value.	Tons 2000 lb.	Value.	Pounds.	Value.
1897	993,138	$44,691	19,113,920	$ 270,952	20,107,058	$ 315,643	1,200	$11,400	162,382	$10,149
1898	1,647,679	82,385	30,199,680	743,820	31,847,359	826,205	1,200	11,400	185,647	11,603
1899	3,632,608	145,304	41,586,000	1,990,649	45,218,608	2,135,953	1,030	8,240	405,870	32,475
1900	4,103,052	164,122	32,298,560	1,389,117	36,401,612	1,553,239	1,045	8,640	860,750	68,860
1901	3,967,612	135,914	32,029,760	895,010	36,997,372	1,067,921	809	31,800	2,500,000	119,000
1902	4,176,824	153,147	40,857,600	1,168,554	45,034,424	1,322,401	4,739	55,964	2,358,828	110,700
1903	4,525,700	164,247	32,012,000	1,207,700	36,537,700	1,371,947	16,591	71,384	2,620,000	178,670
1904	4,357,927	162,332	25,350,000	905,581	29,707,927	1,067,913	19,115	102,925	3,248,000	217,790
1905	4,260,656	170,426	34,914,611	983,034	39,175,267	1,153,460	21,953	80,639	4,595,500	313,979
1906	4,894,483	170,866	50,974,336	1,554,212	55,868,819	1,725,098	....	....	4,868,000	312,764

The exports of graphite from the United States were valued at $834 in 1901, $365 in 1902, $4,220 in 1903, $8,455 in 1904, and $91 in 1905. (b) Nominal. (c) Neglecting the small re-exports of foreign produce.

Classification and Occurrence.

Crystalline.—The graphite produced in the United States is divided into three classes, viz., crystalline, amorphous, and artificial. The distinction between crystalline and amorphous graphite is purely one of quality, the crystalline variety being a very pure graphite which is particularly suited to lubrication, the manufacture of crucibles and pencils and uses for which a very high grade of graphite is essential. New York is the leading State in the production of crystalline graphite, and is followed by Pennsylvania, which two States are practically the sole producers of this variety. Virginia may, in the future, become a producer of crystalline graphite.

Amorphous.—This is an earthy, impure mineral, employed for paint, foundry facings, stove polishes, etc. Under amorphous graphite is included the graphite anthracite of Rhode Island, and the Baraga graphite of Michigan, which is essentially a graphitic schist. Rhode Island, Michigan, Wisconsin and Georgia are the chief producing States. Many other occurrences of amorphous graphite or graphitic schist in various places are known, but the difficulty of purifying these schists, which are impreg-

nated with quartz, intermingled with the graphite in a very intricate manner, is so great that many deposits cannot be exploited even though they contain high percentages of graphite.

Artificial.—This is produced only by the Acheson Graphite Company, of Niagara Falls, N. Y., which produces hard graphite in electrical furnaces to be used in the manufacture of electrodes and as a pigment. Heretofore the company has not encroached on the field occupied by the natural graphite producers, but recently Mr. Acheson announced that he has discovered a process for making artificial soft graphite. This new form can be used as a lubricant, polish, coating for gunpowder, and in electroplating. It is expected that this artificial soft graphite can be produced at a low cost, and it is said that the product from the experimental furnace has been very satisfactory as to uniformity. It is believed that artificial graphite will become a successful competitor of natural graphite and that its use will make the manufacture of graphite one of the largest industries at Niagara Falls.

Value.—The quality of graphite, in so far as that determines to what use it may be put is the all-deciding feature of the value of a graphite deposit. That deposit of amorphous graphite which might be entirely suitable for the manufacture of foundry facings, stove polishes, or such uses, could not be applied at all to those purposes for which crystalline graphite is essential. The only final way by which the value of a body of graphite can be arrived at is to submit samples of the material to the various concerns which are consumers of graphite, a list of whom may be found in THE MINERAL INDUSTRY, Vol. XIV.

Quotations on graphite vary all the way from 1 to 10c. per lb. in carload lots. The Ceylon brand brings from $2\frac{3}{4}$ @ $3\frac{1}{2}$c. per lb., while best pulverized Ceylon is worth from 4 to 8c. per lb. in large lots. German and Italian pulverized brands are worth from 1 to 2c. per lb.

GRAPHITE MINING IN THE UNITED STATES.

Colorado.—The Ethel Gold Mining and Milling Company sold its graphite mine to the Federal Graphite Company, of Colorado, in November, 1906. This property is reported to contain an exceptionally large and rich deposit, the vein being 5 to 6 ft. wide, about a mile long, and 100 ft. in depth, as shown by shafts sunk along its strike. The graphite shows over 90 per cent. carbon. The factory of this company is in Denver, and was put into operation in January, 1906. It is the only graphite factory west of Chicago, and supplies a large portion of the Western trade. The company contemplates the erection of another factory in the East, as its Denver plant is insufficient for both the Western and the Eastern markets.

New York.—The Joseph Dixon Crucible Company of Jersey City, N. J., is the largest operator in the Adirondack region, which borders on Lake

George and the southern part of Lake Champlain. This company also controls the American Graphite Company, which operates a mine at Graphite, near Hague, that is noteworthy for its size and the superiority of its product. The Adirondack Mining and Milling Company controls a graphite deposit at Whitehall. A mill is erected but was operated only for two weeks during 1906 and made but a nominal production. It is in readiness, however, to make a considerable output. The graphite works of this company are at South Bay, in the town of Dresden, and all the graphite is shipped by water four miles to the canal at Whitehall.

The Silver Leaf Graphite Company, of Whitehall, owns property at South Bay, and has a crusher and some other mining machinery, but has never erected a mill. The Champlain Graphite Company, also of Whitehall, reports that as a result of improvements in its equipment it is now in a position to produce one to two tons of graphite per day.

(By D. H. Newland.) The production of crystalline graphite in the Adirondack region showed an unexpected falling off during 1906. It amounted to a total of 2,811,582 ℔., valued at $96,084, as compared with 3,897,616 ℔. valued at $142,948 in 1905 and 3,132,927 ℔. valued at $119,-509 in 1904. The output for 1905 was probably the largest that has been made in any one year since the establishment of the industry. The average value of the graphite per pound was 3.4 c. in 1906, 3.7 c. in 1905 and 3.8 c. in 1904, showing that there has been a decrease also in the market prices received for the product. The relatively small output in 1906 can not be ascribed to any general conditions that are likely to influence adversely the future progress of the industry, but is doubtless a temporary feature incidental to the limited scope of the operations as carried on at present.

The Adirondack graphite deposits which are being exploited are of the bedded type, the graphite occurring as disseminated flakes in a rock matrix, which may be limestone, quartzite or a silicious schist. The rocks are ancient sediments that have been thoroughly metamorphosed, and it is believed that the graphite originated from included organic matter that was distilled by the heat to which they were subjected, with a resultant loss of the volatile constituent. The rocks, which have a thoroughly crystalline character, occur in force on the borders of the Adirondacks, but are less abundant in the interior. Their content in graphite is variable. As a maximum it may reach 8 or 10 per cent., which is only found in the deposits owned by the American Graphite Company, near Hague. The average for the other deposits that have been worked probably does not exceed 3 per cent.

The companies which reported a production for 1906 are: American Graphite Company, with mines at Graphite, near Hague, Warren county; Crown Point Graphite Company, operating near Penfield pond, Essex county; Pettinos Bros., with mines at Rock Pond, Essex county; Adiron-

dack Mining and Milling Company and Champlain Graphite Company, with mines at South Bay, near Whitehall, Washington county. The plant of the Ticonderoga Graphite Company, at Rock pond, was leased to Pettinos Bros., of Bethlehem, Penn., who operated it only for about two weeks. A fault cut off the supply of graphitic rock. The Adirondack Mining and Milling Company was active only for a period of three weeks, and will not resume operations during 1907.

The Glens Falls Graphite Company has been incorporated for the purpose of working a graphite deposit situated near Conklingville, eight miles west of Hadley, Saratoga county. The deposit is described as a quartzose schist with a thickness of 50 ft. where opened. The erection of a mill is in progress, which it is expected will be completed early in 1907. The Empire Graphite Company is engaged in developing a property near Greenfield, Saratoga county.

Some prospecting has been done in the vicinity of Graphite with a view to locating a possible extension of the bed worked by the American Graphite Company. Drill tests have indicated that the quartzite underlies a considerable area to the west of that company's property. In three holes which were put down the graphitic bed was found to be from 19 to 22 ft. thick and was encountered at 50 to 75 ft. depth.

Pennsylvania.—Operations in this State were greatly curtailed in 1906 on account of many changes in the management of some of the producers and in other instances because the companies temporarily abandoned production to undertake development work or to increase milling facilities. The New Philadelphia Graphite Company of Philadelphia, which operates at Chester Springs, made but a small production during 1906, as it was developing its plant. The Federal Graphite Company, of Chester Springs, closed its plant in July, 1905, and in December sold its property. Operations will be resumed in 1907 under new management. The National Graphite Company, at Anselma, changed hands and the mine was worked by the new management during the last two months of 1906. The United States Graphite Company, of Byers, was a producer. The Continental Graphite Company of Byers, and the Sterling company and the Parker Graphite Company, both of Chester Springs, are all reported to be constructing mills. The Columbia Graphite Company, of Philadelphia, which must not be confounded with the company of the same name at Ticonderoga, N. Y., is reported to have re-opened its mines in November, 1906, after having been idle for 18 months.

Virginia.—The expected production from this State did not materialize during 1906. The only graphite concern here is the Naylor-Bruce company of Free Union, Va., having deposits in Albemarle and Orange counties. The location, extent and mode of occurrence of these deposits were briefly described in THE MINERAL INDUSTRY, Vol. XIV. Up to the end

of 1906, the company had put none of its product on the market but was working along the outcrops of three veins and was doing a little boring. The erection of a refining plant in the spring of 1907 is contemplated. The graphite mined by this company is of the flaky, crystalline variety, although the outcrop mineral comes near being amorphous.

Other States.—The Detroit Manufacturing Company operates on the upper peninsula of Michigan, and has factories at L'Anse in the same State. The product of this company is nearly all ground and made into paint. The Hathaway mine of the L'Anse district is no longer operated. The Crystal Graphite Company of Dillon, Mont., produced a small amount of crystalline graphite, but did not dispose of its product, holding it at the mines. The Wisconsin Graphite Company, operating at Stevens Point, Wis., maintained a steady production throughout 1906. In Illinois the Chicago Graphite Manufacturing Company retired from business in July, 1906. A new concern, the Cherokee Chemical Company, of Atlanta, Ga., is reported as having been organized late in 1906 for graphite exploitations in that State. In North Carolina the Southern Engineering Company is to develop graphite deposits near Newton, Catawba county, and it is stated that the intention is to build a manufacturing plant.

Graphite Mining in Foreign Countries.

Canada.—The appreciation of the high grade of Canadian graphite and of its wide distribution is resulting in the steady development of an industry which is likely to become of considerable importance to the Dominion. In Quebec, the Diamond Graphite Company, of which J. J. Tonkin, of New York, is president, has acquired several hundred acres of land five miles north of Buckingham. These graphite properties have been known for more than 25 years. The graphite is of the flaky variety and occurs in a rusty, laminated gneiss. According to Fritz Cirkel (*Eng. and Min. Journ.*, Feb. 9, 1907) a number of prospecting ditches, open cuts and drill holes have proved the deposit to be of large extent. A 100-ton mill for the treatment of graphite rock is almost completed. It is calculated that the mill will turn out eight tons of finished product per day. Another 100-ton mill for the same company is in course of construction on another graphite property near by.

At Calumet station on the Canadian Pacific railway, the Calumet Graphite Mining and Milling Company, composed of American capitalists, is operating a property containing over 300 acres. A large mill is in process of construction. A new departure in this mill will be pebble-tube mills for the cleaning and polishing of graphite flakes.

The graphite deposits in the Laurentian district of northwestern Quebec are to be developed on a large scale by American capital. A. L. Dubois, vice-president of the Dry Concentrator Company, of New York, states

that his company, which has secured 20 properties situated between Buckingham and Lavelle, will commence active mining operations shortly.

As to the Canadian practice in milling graphite, much secrecy is observed, and all the methods described are somewhat uncertain.[1] In earlier times, crushing and concentration were carried out by means of stamps and buddles, and the concentrates, after being dried, were put through a system of buhrstones and revolving screens until more or less finished, and were then graded and marketed. Recent practice, as exemplified in the mill of the Anglo-Canadian Syndicate is to use jaw crushers and rolls, sizing screens to prepare the crushed product for pneumatic jigging, dryers, and air jigs. The concentrates are saved, and the jig tailings in this mill go to a Brumell concentrator, a wet separator which relies on the floating power of graphite when dry, upon the surface of a moving body of water. The concentrates from this wet separator are dried in a revolving steam dryer and all concentrates go to buhrstones, screens and graders which prepare the material for market.

Ceylon.—Graphite from Ceylon is the most expensive grade that is found on the market and owes its value to its fibrous character which makes this mineral particularly suitable for the manufacture of crucibles. In fact, no other deposit of graphite is known which is a satisfactory substitute for the Ceylon variety. The fibrous nature of this graphite gives a grain to the crucibles and thereby develops a wood-like texture in the crucible and makes it tough. If it were not for this fact, a crucible could not withstand the heavy weight of the metal which it does and it would break like clay. The Mineralogical Survey of Ceylon reports an increase in production for the last three years, but a considerable falling off in value which in some cases leaves barely sufficient margin to cover expenses. The chief mines are in the Ratnapura and Kurunegala districts and are mostly managed by native capitalists. At Dumbra in the Ratnapura district, a European syndicate is carrying on operations and the prospects are said to be favorable.

Queensland.—Graphite occurs in many localities in Queensland, but is produced on a commercial scale only at Mount Bopple, which is situated between Gympie and Maryborough on the line of the North Coast railway. The graphite bearing belt occurs on the spurs of the gap in the range that connects Mount Bopple and Beacon Peak. The deposits are being proved by four different workings. The seams total 11 ft. in thickness and average 28 per cent. graphite carbon, and 70 per cent. of ash, which is mainly silicate of alumina. The graphite itself lies under a graphitic shale and caps a bed of graphitic and anthracite shale which in turn overlies a stratum of anthracite interbedded with blue shale. Other deposits in Queensland occur in the Mackay district on Scrubby creek, from which

[1] H. P. H. Brumell, *Journ. Can. Min. Inst.*, Vol. IX.

place 20 tons of graphite were bagged for experimental purposes, but the results of the trial have not been ascertained. Still other deposits are known at Hornbush Hill and at Cloncurry.

New South Wales.—A deposit of graphite bearing shale is exposed near Underwood Station, about 12 miles from Wilsons Downfall, near the Queensland border. A trial was made of this graphite by the Springfield Facing Company, of Massachusetts, which reported that it was suitable for foundry facing and was especially good in foundries where the facing is used wet. Transportation prices, however, will limit the markets which can profitably be supplied by this deposit. Occurrences of graphite are also known at Grafton, Mudgee, Fairfield, Hillgrove, Tobin's creek, Cowell creek, near Port Macquarie, Dundee, Tenterfield, Pambula, Cordeaux river, Plumbago creek in the county of Drake, and at the head of the Abercrombie river.

WORLD'S PRODUCTION OF GRAPHITE.
(In metric tons.)

Year	Austria	Canada	Ceylon (*d*)	Germany	India	Italy	Japan	Mexico	Sweden	United States (*b*)	Totals
1896	35,972	126	10,463	5,248	(*a*)	3,148	215	620	14	184	55,356
1897	38,504	396	19,275	3,861	61	5,650	204	907	99	450	69,311
1898	33,062	1,107	78,509	4,593	61	346	346	1,857	50	824	125,006
1899	31,819	1,105	29,037	5,196	1,548	9,990	55	2,305	35	1,648	80,962
1900	33,663	1,743	19,168	9,248	1,858	9,720	942	561	84	1,799	79,938
1901	29,992	2,005	22,707	4,435	2,530	10,313	88	1,473	(*e*)56	1,800	75,399
1902	29,527	978	25,593	5,023	4,648	9,210	97	580	63	1,895	77,517
1903	29,590	670	24,492	3,720	3,448	7,920	114	1,952	25	2,053	73,984
1904	28,620	411	26,478	3,784	3,800	9,765	114	1,952	55	2,045	77,024
1905	34,416	491	(*c*)	4,921	2,361	10,572	(*c*)	(*c*)	(*c*)	1,933	....

(*a*) Not reported in the government statistics. (*b*) Crystalline graphite. (*c*) Statistics not yet published. (*d*) The figures for 1897 and 1899 are exports; the enormous production in 1898 as reported in official government publications is not reflected in the exports for that year, which amounted to 24,349 metric tons. (*e*) The production of crude graphite in 1901 was 1727 tons.

Chemical Analysis.

The complete and accurate analysis of graphite is not easily accomplished. Moisture is determined by drying a portion of the sample at 150 deg. C. This temperature will not drive off chemically combined water which may be in the clay or other foreign substances, and in order to eliminate this, it is necessary to heat the graphite at low redness in a current of dry air and collect the expelled water in a calcium chloride tube.

Carbon is determined most accurately by oxidizing to carbon dioxide through the agency of chromic and sulphuric acids and collecting the evolved CO_2 in weighed caustic potash tubes. Another method for carbon is to heat the sample in a current of oxygen in an ordinary combustion tube. However, as all graphites are not amenable to this last process, a preliminary test should be made to show whether the graphite can be thus oxidized quantitatively.

Other substances present in a graphite are determined indirectly as a whole, by deducting the percentage of moisture, chemically combined water, and carbon from 100. But if it is desired to make a direct deter-

mination, a portion of the finely powdered substance is burned to an ash, which is subsequently treated with a concentrated solution of ammonium carbonate to replace any carbon dioxide which was driven off by ignition. After being moistened by ammonium carbonate, the sample is dried and gently ignited; the weight of residue will represent the total mineral constituency and it may be analyzed according to any well known method for inorganic substances.

Bibliography.

Graphite is dealt with in every volume of The Mineral Industry and each contribution contains a complete review of the progress of the industry for the year in question, with a review of the discoveries of graphite and the deposits which were worked. Uses of graphite and statistics of production are also given. Aside from these general features various special topics received more complete treatment and these are summed up as follows:

In Vol. II, extensive mention of the graphite deposits of Canada, England, Germany and Russia, together with a few notes on the occurrences of graphite in the United States that were not exploited during 1903. In Vol. VI, an account of tests made to determine the relative efficiency of iron oxide and graphite paint for covering and protecting steel structures, with notes on the best system to follow in applying graphite paint as recommended by Toltz. In Vol. IX, a description of methods of assaying graphite by blast and by fusion with a discussion of the relative importance of each method. In Vol. X, a few brief comments upon the manufacture of artificial graphite by the Acheson process at Niagara Falls. In Vol. XI, a very complete review of the progress of the graphite industry during 1902, a short article by W. E. H. Carter describing the graphite industry in Canada during 1902, and a review of the condition of the artificial graphite industry. In Vol. XIII, lists of crucible manufacturers and grinders, paint manufacturers, stove polish manufacturers, manufacturers of foundry facings, and producers of graphite lubricants are given.

Following are some of the more important of the recent publications and patents relating to graphite:

"Die Technische Gewinnung von Graphit und Amorphem Kohlenstoff." E. Donath. ((*Stahl u. Eisen*, Oct. 15, 1906; 6 pp.) Outlines the method of making artificial graphite in the electric furnace, with a discussion of the chemistry involved.

"Graphite in Maine." G. O. Smith. (*U. S. Geol. Surv., Bull. No.* 285, 1906; 3½ pp.) Notes the occurrence of graphite at two localities in western Maine.

"Graphite in Queensland, with Special Reference to the Mount Bopple Graphite Deposits." B. Dunstan. (*Geol. Surv. of Queensland, Pub. No.*

203; 20 pp., 5 plates.) Discusses the occurrence of graphite in Queensland, its mining and concentrating; some analyses of Mount Bopple graphite are given.

Graphite-Separator. John H. Davis, Glens Falls, N. Y. U. S. pat. 816, 303.

Production of Graphite. Edward G. Acheson, Stamford Township, Ontario, Canada. U. S. pat. 836,355.

Extraction of Graphite. Morgan Crucible Co., London, England. Improvements in processes for extracting graphite from gangue by greasing and flotation. Brit. pat. 10,475 of 1905.

GYPSUM.

BY A. VAN ZWALUWENBURG.

In the gypsum industry of the United States during 1906, the operations of the United States Gypsum Company overshadowed all others. This corporation controls 35 plants distributed through 12 States and during the year produced more than half the gypsum mined in the entire country. The deposits from which the company draws its supplies include beds in Iowa, Kansas, Michigan, Ohio, Oklahoma and New York. A number of new companies were organized during the year, mostly to work deposits for local demands, but few are producers, the reason being

STATISTICS OF GYPSUM IN THE UNITED STATES.
(In tons of 2240 lb.)

Year.	Production.		Imports.					
			Crude.		Ground or Calcined.		Plaster of Paris.	Total Value of Imports.
	Quantity. (*b*)	Value. (*c*)	Quantity.	Value.	Quantity.	Value.	Value.	
1896	195,553	$583,136	180,269	$193,544	3,292	$21,982	$11,722	$ 227,248
1897	268,187	889,177	163,201	178,686	2,664	17,028	16,715	212,429
1898	281,130	864,415	166,066	181,364	2,973	18,501	40,979	240,844
1899	376,840	1,155,581	196,579	220,603	3,265	19,250	58,073	297,926
1900	432,323	1,316,255	209,881	229,878	3,109	19,179	66,473	315,530
1901	588,981	1,577,493	235,204	238,440	3,106	19,627	68,603	326,670
1902	(*a*) 728,998	2,089,341	305,367	284,942	3,647	23,225	52,533	360,700
1903	(*a*) 930,093	3,792,943	265,958	301,379	3,526	22,784	54,434	378,597
1904	(*a*) 840,104	2,784,325	294,238	321,306	3,278	11,276	23,819	356,401
1905	(*a*) 931 475	(*d*) 821,967	356,457	402,378	3,471	20,883	22,959	446,220
1906	(*a*) 1,375,588	3,837,975	390,178	464,724	3,203	22,821	21,297	508,842

(*a*) Statistics of the U. S. Geological Survey. (*b*) Represents the amount of crude gypsum quarried. (*c*) Represents the value of the marketed gypsum, including its various finished forms. (*d*) Value of crude material.

the abundance and the relatively low price of the material. The price of the rock ranged from 46c. to $1.22 per ton according to quality and locality.

California.—Frank L. Hess, in a bulletin issued by the United States Geological Survey, reviewed the gypsum industry of California. There are four mills in the State, one at Amboy, one at Los Angeles and two at Palmdale. The product of these mills is consumed in the State, mostly for wall plaster, but a smaller amount is ground for use as a fertilizer. De-

posits suitable for the manufacture of plaster of paris occur in California, but they are far from railroads, and are at present unavailable. The gypsum mills of the State use oil for fuel in calcining.

Colorado.—In Jefferson county a large body of gypsum was encountered in the tunnel driven near Morrison for the American Cement Plaster Company of Lawrence, Kan., the bed showing a thickness of nearly 12 ft. The gypsum is said to be of good quality and the plans of the company include the erection of a mill of 200 tons daily capacity near Denver.

PRODUCTION OF CRUDE GYPSUM IN THE UNITED STATES.
(In tons of 2000 lb.).

States.	1903. (*a*)		1904. (*a*)		1905. (*a*).		1906. (*a*)	
	Sh. Tons.	Value. (*b*)	Sh. Tons.	Value. (*b*)	Sh. Tons.	Value. (*h*)	Sh. Tons.	Value. (*b*)
Cal, Ohio and Va	103,392	$467,113	101,809	$318,723	147,136	$188,974	(*f*)202,376	$536,940
Colo and Wyo...	33,549	133,347	35,778	135,045	26,880	26,930	(*g*)	....
Iowa,Kan and Tex	307,102	1,087,045	(*c*) 319,080	1,027,792	(*d*) 375,239	296,247	(*d*)639,885	1,546,188
Michigan.........	269,093	700,912	238,385	541,197	299,585	143,597	341,716	753,878
New York.......	137,886	462,383	158,892	432,358	153,367	151,272	288,631	749,896
Oklahoma.........	69,158	234,621	53,523	190,245	(*e*)	....	(*e*)	....
Other States.....	121,524	707,522	33,450	138,965	40,995	14,947	67,977	251,073
Total........	1,041,704	$3,792,943	940,917	$2,784,325	1,043,202	$821,967	1,540,585	$3,837,975

(*a*) Statistics of the U. S. Geological Survey. (*b*) Value includes that of prepared products. (*c*) Of which, Iowa, 145,359 tons—$475,432. (*d*) Includes Oklahoma. (*e*) Included with Iowa, Kansas and Texas. (*f*) Includes Nevada and Oregon. (*g*) Included in other States. (*h*) Value as mined.

PRODUCTION OF GYPSUM IN THE PRINCIPAL COUNTRIES. (*a*)
(In metric tons.)

Year.	Algeria. (*b*)	Canada.	France. (*b*)	Germany. (*c*)		Greece.	India.	United Kingdom.	United States.
				Baden.	Bavaria.				
1896..............	37,512	187,818	2,051,124	32,801	28,799	120	7,605	196,404	201,305
1897..............	36,750	217,392	2,004,339	40,702	26,153	51	8,187	184,287	272,493
1898..............	37,337	198,908	2,115,261	28,037	25,688	83	8,390	199,174	285,644
1899..............	39,950	221,862	1,807,454	29,419	29,727	81	6,546	215,974	382,891
1900..............	42,237	228,713	1,774,492	26,381	35,484	129	4,415	211,436	439,265
1901..............	44,025	266,605	2,385,633	28,183	3,581	671	(*d*)	204,045	598,529
1902..............	44,975	301,229	2,185,346	(*d*)	31,701	*Nil*	(*d*)	228,274	740,906
1903..............	41,550	285,380	1,998,804	(*d*)	30,894	94	(*d*)	223,426	945,285
1904..............	48,375	309,220	1,957,802	(*d*)	22,766	393	3,937	237,749	853,546
1905..............	34,743	395,453	1,376,145	(*d*)	46,247	(*d*)	4,877	220,416	982,626
1906..............	(*d*)	378,950	(*d*)	(*d*)	(*d*)	(*d*)	(*d*)	199,281	1,397,480

(*a*) From official reports of the respective countries, except the statistics for the United States. (*b*) A part of the product is reported as plaster of paris. In converting this to crude gypsum it has been assumed that the loss by calcination is 20 per cent. (*c*) Prussia is a large producer of gypsum, but there are no complete statistics available. (*d*) Statistics not yet available.

Uses of Gypsum.

H. Worth describes a series of experiments to determine the possibility of using gypsum instead of sulphuric acid in recovering ammonia from the liquors from gas works.[1] The gas water from the municipal gas plant of Birmingham was subjected to a preliminary test which showed that practically all the ammonia was uncombined with heavy acids which would permit its recovery by evaporation. By shaking the liquor with gypsum or plaster of paris all the ammonium carbonate was converted with ammonium sulphate, resulting in the recovery of 80 per cent.

[1] *Chem. News,* June 1, 1906.

of the ammonia without the use of sulphuric acid. By treating the liquor first with gypsum and the remaining clear solution with ferrous sulphate, 95 per cent. of the total ammonia was recovered.

The Perin Process.—Louis Perin[1] developed a method for calcining gypsum so as to obtain a plaster of a predetermined quality. The amount of dehydration desired and the character of the gypsum is first determined and the amount and duration of heating required calculated. The operation is performed in a special kiln having a central fire-box placed between two cylindrical calciners in which the dehydration takes place. The coke fire upon the grate is controlled by mechanical draft, the heated gases passing through one of the cylinders, and the air for combustion being drawn through the other. After a charge has been sufficiently dehydrated, the direction of the air and gases is reversed, the cold air cooling the burned charge and the heat from the fire dehydrating a fresh charge. The cylinders revolve at a rate of one r.p.m. and are provided with numerous partitions and passages for thoroughly stirring the charge. The degree of dehydration is controlled by adjustment of the temperature, indicated by a pyrometer, and of the duration of the operation.

Influence of Calcining Temperature Upon the Hardening of Plaster.—E. Leduc and Maurice Pellet[2] described a series of experiments to determine the effort of the temperature of calcining upon the "set" of plaster. Quantities of pure alabaster were heated to different temperatures between 120 and 1185 deg. C. and the resulting plaster mixed with water and allowed to harden. A thermometer in the mass recorded the rise in temperature due to the reaction of hardening and an instrument bearing a needle recorded the time and degree of solidification. Material dehydrated at 120 deg. C. began to harden in eight minutes, the "set" being complete in 16 minutes. Calcining at 250 deg. C. gave solidification between the fourth and sixth minute; 450 deg. C. gave an early beginning of the "set" which later proceeded more slowly. Material calcined at 500 and 600 deg. C. yielded a solid mass, but temperatures above 650 deg. C. destroyed all power of hardening. Plaster dehydrated at 250 deg. C. solidified with a marked rise of temperature, while that calcined at 450 deg. C. gave no evolution of heat.

Manufacture of Calcined Plaster of Paris.

By C. O. Bartlett.

Gypsum rock is found in many States throughout the Union, in large quantities and near the surface. It varies somewhat in color and considerably in hardness. The most of it is quite hard. It contains about 25 per cent. moisture.

[1] *Le Genie Civil*, 1906, p. 68.
[2] *Le Genie Civil*, 1906, p. 253

Plaster of paris is generally called calcined plaster, and in this article the term calcined plaster will be used hereafter. The manufacture of calcined plaster has increased very rapidly during the last few years. It is the base of all ready-made plaster, and is used in the manufacture of portland cement, and for many other purposes. The demand for it is increasing rapidly. To manufacture gypsum rock into calcined plaster, in quantities of 50 tons per day of 10 hours, the following machinery, with weight, horse-power, etc., is required:

The gypsum rock is first crushed in an ordinary 9x15-in. jaw breaker, which will crush five tons an hour. The breaker weighs 15,000 ℔. and requires 10 horse-power to drive it. From this breaker the crushed gypsum rock goes to the second crusher, called a pot crusher, and is again crushed. The weight of this crusher is 3500 ℔. The power to drive it is 10 horse-power.

From the second crusher the material goes to the drier (Fig. 1), the weight

FIG. 1.—DRIER FOR MANUFACTURE OF PLASTER OF PARIS.

of which is 21,000 ℔. It requires 8 horse-power to drive, including exhaust fan. This drier should be erected in brickwork similar to a boiler. It should have an automatic feeder. The products of the fire should not come in contact with the material being dried on account of the danger of discoloring it, if soft coal or wood be used as fuel. In some places, however, the products of the fire are allowed to come in contact with the material even when coal is used for fuel. Where this is done great care should be used in firing. On general principles it is better not to have the fire come in contact with the material at all, if coal or wood be used for fuel. Any kind of fuel can be used—oil, gas, coke, wood or coal. This drying process eliminates 10 per cent. of the moisture of the rock.

From the drier the crushed rock goes to a trommel, 36 in. in diameter and 12 ft. long, with 24-mesh wire cloth. That which passes through

the 24-mesh screen goes to the bins over the kettle. The remainder goes to the bins over the grinding mills, which have solid French buhr stones with iron frames. The stones are very heavy, 8 in. in thickness and 36 in. in diameter. The weight of the mills is five tons, and it requires 35 horse-power to drive them. The product from the stones should pass to bins over the kettle. The ground gypsum is now ready for boiling.

The boiling is done in a large kettle, 10 ft. in diameter and 10 ft. deep (Fig. 2). The weight of this kettle is 10 tons, and it requires 25 horse-power to drive it. The kettle is made with wrought-steel sides and cast-iron, or very heavy steel, bottoms made convex so as to better resist the heat without sagging. The shell of the kettle should be made of heavy

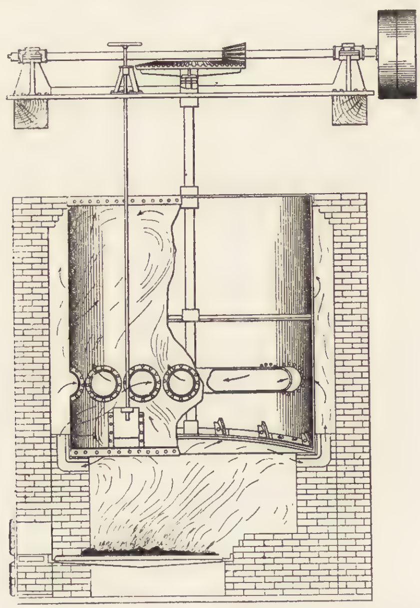


FIG. 2. KETTLE FOR MAKING PLASTER OF PARIS.

steel. There should be four flues about 12 in. in diameter running through this kettle near the bottom. Inside the kettle is a heavy shaft supported from above, with a heavy sweep or stirrers below the flues near the bottom, and some mixing paddles also above the flues near the top. This shaft should be very strong and supported from above; otherwise the weight of it would tend to break down the bottom of the kettle. The bottom of the kettle is a little smaller so there is room to raise the old one out and admit putting in a new one. A 10-ft. kettle will hold 10 tons. The kettle is set on heavy brickwork, with fire front, doors, grate bars, etc. The heat passes directly under the kettle and around the lower part of the sides, then through the flues and around the upper part of the sides,

then out at the stack. The flange ring for supporting the kettle is 8 ft. from the bottom, resting on top of the brickwork. The brickwork for the kettle should be 2 ft. 6 in. thick and 8 ft. high. It requires about 22,000 brick to erect properly one of these kettles.

The ground gypsum is fed into the kettle in quantities of about 10 tons at a time and continually stirred and boiled until the remainder of the free water is driven off. It usually takes about two hours to boil one batch. It should be brought to a temperature not to exceed 265 deg. F. When properly boiled it will settle and is ready to be discharged. This is done through a discharge gate on the side near the bottom, this gate being operated with a lever from above. This is the first boiling, or settling. A large proportion of the calcined plaster is used in this form. If it be desired to make a finer grade, it is necessary to boil it a second time, which is in reality to take out the water of crystallization. If this water of crystallization is once started, it must be taken out; otherwise the calcined plaster will be spoiled. Therefore it is important not to get the temperature above 265 deg. in the first boiling, and from 390 to 395 deg. in the second boiling. Considerable care must be taken with the boiling of the material, for here is where the chemical action takes place. After the material has been boiled it should be screened again with a trommel with about 40-mesh wire cloth. The oversize from this trommel should then be ground by a 24-in. French buhr mill.

In addition to the above mentioned machinery the necessary elevators, conveyors, shafting, belting, bins, etc., will be required. On account of the nature of the material, the bins, conveyors and elevators should be made of steel, and elevators and conveyors should have steel casings. The estimated weight of elevators, conveyors, etc., is 10 tons, and the power to drive them 25 horse-power.

To manufacture 50 tons of plaster of paris per day of 12 hours will cost about 80c. per ton for the labor and packing. Counting 10 per cent. depreciation on machinery, which is ample, and 6 per cent. interest on the investment, say $10,000, the total cost per ton, including the mining, labor, milling, fuel, depreciation and interest on the investment, will be $1 per ton. In figuring this cost, I have calculated 30c. per ton for mining, which is about the cost in most places where the gypsum is hard and of a rocky formation.

It is very much cheaper to make plaster of paris in quantities of 50 tons per day than, say, 50 bbl. per day. In making less than 50 tons per day the cost per ton increases. In fact, in my judgment, if it were desired to make less than 50 tons per day, it would be better to install a plant of 50 tons capacity and run it only part of the time. In making 100 tons per day, the cost per ton would be decreased about 10 per cent., making the total 70c. per ton, allowing 30c. per ton for mining.

To manufacture ready-made plaster, or ready-mixed plaster, calcined plaster is used as the base. It is mixed with sand, hair and retarder, or wood fiber and retarder, in the following proportions for wall plaster: Calcined plaster, 300 lb., goat hair (about), 1 lb.; dried sand, 600 lb.; retarder, from $1\frac{1}{2}$ to 2 lb.

If desired, hydrated lime can be used; and if so, one-third as much hydrated lime as there is calcined plaster. Also about 2 lb. of asbestos fiber can be used to good advantage; sometimes this adds considerably to the working of the plaster. For sand mortar and brick walls 300 lb. of calcined plaster and 900 lb. of dried sand are used; no hair is necessary. There must be either all calcined plaster and sand, or calcined plaster part and hydrated lime part and the remainder sand, in the proportion of one part calcined plaster to two parts of sand for lime mortar, and one part calcined plaster to three parts sand in brick mortar.

Wood fiber is largely used for ready-made wall plaster, and is one of the best fillers known in plaster work. It makes the plaster elastic and decreases its weight and it also helps to make it sound proof and electricity proof. The following is a good proportion when wood fiber is used: Calcined plaster, 600 lb.; clay, 150 lb.; wood fiber, 40 lb.; retarder, from 3 to 5 lb.

The color of the clay makes no particular difference, as a white coat usually follows. Some manufacturers use a low grade cement, and add as much as 200 lb. of it to the above formula.

Wood fiber can be made of almost any kind of soft wood. It is made as follows: First, the wood is cut into lengths to suit the machine, usually about 2 ft. long and about 24 in. in diameter. The block is placed in the carriage between centers and caused to revolve around the shortest diameter. The saws are set on a bias and revolve very rapidly, so that instead of cutting directly into the wood they make it into fibrous material, each piece being from $\frac{1}{2}$ to $1\frac{1}{2}$ in. long, according to the fineness of the material desired. This machine weighs 2000 lb. It requires 15 horse-power.

The proportion of materials in making ready-made plaster can be varied according to the demand of the architect and the purpose for which it is to be used. Large quantities of hemp or old rope were formerly used, but are not used much at the present time. Calcined plaster is the base of all hard plaster, and without it hard plaster cannot well be made. The retarder is used to prevent the quick setting of the plaster. It is manufactured at several places and can be bought for $45 a ton. In making a good quality of ready-made plaster it is very important either to weigh or measure the material carefully. No guessing should be allowed. It is also important to mix thoroughly the materials, and the use of a batch mixer is always recommended. It is also very important that the sand be thoroughly dried.

IODINE.

Iodine is not produced in the United States and the domestic demand has always been supplied by imports which come principally from Chile and Scotland. Other producing countries are France, Japan, and to a limited extent, Scandinavia.

The largest purchasers of iodine in this country are the manufacturing chemists and pharmaceutical firms, the chief uses of iodine being as a chemical reagent and as a constituent of various drugs and medicines. Perfectly pure iodine for these purposes is not difficult to obtain, but as purchased in the market it is likely to be contaminated with small amounts of chlorine and moisture, and it is sometimes adulterated with graphite or with powdered charcoal.

The three sources of supply are seaweed, which is the raw material used in Scotland and in Scandinavia, the nitrate deposits of Chile, from which iodine is extracted as a by-product, and the phosphate deposits of France, which contain small amounts of iodine.

AVERAGE PRICE OF IODINE PER POUND.

Year.	London.	New York.	Year.	London.	New York.
1897....	10s. 8d.	$2.53	1902.....	8s. 0d.	$1.90
1898....	10 0	2.37	1903.....	8 0	1.90
1899....	10 0	2.37	1904.....	10 2	2.41
1900....	9 6	2.25	1905.....	11 2⅔	2.66
1901....	8 1⅓	1.93	1906.....	8 0	1.90

Iodine in Foreign Countries.

Chile.—The history of the iodine industry since 1874 is a long course of formation of various combinations, followed by their disruption by new competitors, only to be renewed again by absorbing the new producers into another combination. At present the output of iodine is all in the hands of a London corporation with agents at New York, and the price of iodine at these two points is practically the same except for cost of transportation and differences in exchange. However, Consul A. A. Winslow, of Valparaiso, reports that the iodine combination in the nitrate field is having some difficulty in controlling the output. On Oct. 31, 1906, the combination had on hand 25,331 Spanish quintals, while the deliveries for the year to the same date were 7078 quintals, showing a gain of 3155 quintals over 1905. The price in 1906 held at 12 c. per oz.

Japan.—In this country the Bureau of Fisheries and Marine Products has stimulated investigations to determine the best places for collecting seaweed and the best process of extracting iodine from it, and under this government encouragement the industry has increased considerably. The chief producing centers are at Hokkaido and in the provinces of Chiba, Miga, Shiznoka and Kanagawa. The iodine content of the seaweed depends upon the species of the algae, its age and the season in the year in which it is collected. A ton of seaweed contains from 1 to 3 ℔ of iodine. The seaweed is burned to ashes in specially constructed furnaces and the ashes are leached with water to extract the soluble salts and the solution is evaporated. The residue from evaporation is heated with sulphuric acid and manganese dioxide and the iodine is distilled and collected in crystalline form in condensers. The production of Japan is uncertain, as no official statistics are available; there are reported, however, the following exports of potassium iodide: In 1902, 4036 ℔. valued at $7410; in 1903, 29,373 ℔. valued at $50,585; and in 1904, 68,812 ℔., valued at $133,400.

Analytical Methods.

A recent method of determining iodine in mixtures of halogen salts, proposed by Jannasch and Zimmerman (*Ber.*, XXXIX, 1906) is as follows: Dissolve in 120 to 150 c.c. water, add 10 c.c. pure glacial acetic, and at least 3 c.c. of Mercks 30 per cent. H_2O_2. Then distil in a current of steam. Iodine only passes over. The main receiver should contain 80 to 100 c.c. water, 10 grams crystallized hydrazin sulphate and 10 c.c. of the strongest ammonia. In the Peligot bulb tube used as a guard beyond the main receiver, 25 c.c. water, 2 c.c. conc. NH_4OH and 0.5 gram hydrazin sulphate. The solution decolorizes after heating three to five minutes, but the distillation should be continued for 20 to 25 minutes. Finally rinse the receivers into a beaker, add 30 to 40 c.c. conc. HNO_3 and precipitate with $AgNO_3$, keeping the solution warm until the solution clears before filtering.

Bibliography.

In Vol. II, there is a full discussion of the properties, sources, methods of purifying, testing and extracting iodine, and an account of industrial and market conditions. Vol. VI contains a statement of prices and production of iodine from 1893 to 1897, and a new process of extracting iodine from seaweed, patented by A. James, is described.

IRON AND STEEL.

By Frederick Hobart.

The wave of prosperity in the iron and steel manufacture continued to rise throughout 1906. Early in the year there were some predictions that the highest point had been reached, but the brief cloud soon passed over, and the second half of the year was a time of unbounded activity. The result is seen in a production of iron and steel beyond all previous records.

Notwithstanding the extraordinary output, which taxed to the full the large increase in capacity of furnaces and mills made during the past three years, it was not possible to meet promptly the demands from consumers for material. In the closing month of 1906 it became necessary to import considerable quantities of pig iron to meet the demands of Eastern foundries. Though large in comparison with other years, the quantity of these imports is really insignificant when compared with the total consumption.

IRON ORE MINED AND CONSUMED IN THE UNITED STATES.
(In tons of 2240 lb.)

District.	1900	1901	1902	1903	1904	1905	1906
Lake Superior	19,095,393	20,589,237	27,571,121	24,099,550	21,822,839	34,353,456	38,522,129
Southern States	5,100,000	4,767,667	4,850,000	5,889,000	5,450,000	7,175,000	7,450,000
Other States	1,758,000	2,530,575	2,215,000	2,483,000	2,190,000	3,050,000	3,265,000
Total	25,917,393	27,887,479	34,636,121	32,471,550	29,462,839	44,578,456	49,237,129
Add decrease in stocks	....	45,007	....	703,169	....	....	....
Add imports	897,792	966,950	1,165,470	980,440	487,613	845,651	1,060,390
Total	26,815,185	28,899,436	35,801,591	34,155,159	29,950,452	45,424,107	50,297,519
Increase in stocks	690,000	....	1,214,591	....	....	....	....
Deduct exports	51,460	64,703	88,445	80,611	213,865	208,058	265,240
Total consumption	26,073,725	28,834,733	34,499,555	34,074,548	29,736,587	45,216,049	50,032,279

SHIPMENTS OF IRON ORE FROM LAKE SUPERIOR.
(In tons of 2240 lb.)

Range.	1904	1905	1906	Range.	1904	1905	1906
	Tons.	*Tons.*	*Tons*		*Tons*	*Tons.*	*Tons.*
Marquette	2,843,703	4 210,522	4,057,187	Vermilion	1,283,513	1,677,185	1,792,355
Menominee	3,074,848	4,495,451	5,109,088	Mesabi	12,152,008	20,153,699	23,792,782
Gogebic	2,398,287	3,705,207	3,641,685	Baraboo	67,480	111,391	128,742

Iron Ore.

The total shipments of iron ore from the Lake Superior region in 1906 were 38,522,139 long tons, an increase of 4,168,683 tons over the previous year. This total includes 128,742 tons from the Baraboo range in Wisconsin, which is outside of the Lake Superior district proper, though some geological authorities trace a connection between its deposits and those of the Gogebic. It does not include 121,555 tons from the Michipicoten range in Canada, which was chiefly used in the blast furnaces at the Sault Ste. Marie.

During the half century which has passed since the first shipments of Lake ore were made, records have been carefully kept; and it is possible, as in the table herewith, to give the total shipments from each range from the beginning to the close of last season.

LAKE SUPERIOR ORE SHIPMENT TO END OF 1906

Range.	Tons.	Per Cent.	Range.	Tons.	Per Cent.	Range.	Tons.	Per Cent.
Marquette.....	80,857,801	23.9	Gogebic.....	50,476,655	14.9	Mesabi......	122,742,838	36.3
Menominee....	58,676,485	17.3	Vermilion....	25,490,255	7.5	Baraboo.....	327,846	0.1

The shipments from the region up to the close of 1906 reached the great total of 338,571,894 long tons; the equivalent of nearly 200,000,000 tons of pig iron. The Mesabi is the latest of the great ranges to be opened, but its development has been the greatest. It is the growing part of the district, and that upon which future expansion chiefly depends.

An important change affected during the year was the transfer of the so-called Hill iron ore lands, comprising nearly all the northern section of the Mesabi Range to the United States Steel Corporations. The lands have not been worked to any large extent, but include an enormous tonnage of ore which has been proved by drilling and otherwise.

In the Southern field, ore developments on a large scale were carried on in Alabama, by the Tennessee Coal, Iron and Railroad Company and the Republic Iron and Steel Company.

In all the districts outside of the leading ones above mentioned, operations were carried on steadily. In the Lake Champlain and Hudson river districts in New York; in the Lehigh Valley and Cornwall mines in Pennsylvania; in Colorado and New Mexico, much was done. No important new mines were opened during the year in these minor districts, the work done being chiefly in old mines.

The average price of Old Range bessemer ores, from the Lake Superior district, for the season of 1906, was $4.25 at Lake Erie ports. The average charges are estimated as follows: Mining, 85c.; rail freight to lake,

$1; lake freight, 75c.; taxes, 5.5c.; interest, 2c.; royalty, 40c.; total, $3.075. This leaves a net value of $1.75 per ton at the mines. The prices of Mesabi ores averaged $0.25 below those of Old Range ores. The large demand was followed by an increase in prices, and toward the end of the season higher prices were made on the small quantities not covered by season contracts. A higher price was also fixed as a base for 1907.

While this advance in the base price of Lake Superior iron ores, was from 50 to 75c. per ton, it was really greater than that, owing to a lowering of the guaranteed iron content. The accompanying table shows the base guarantees for 1907, as compared with those for the current year.

LAKE ORE PRICES.

Bessemer Ores:	Old Range.		Mesabi.		Non-bessemers:	Old Range.		Mesabi.	
	1906	1907	1906	1907		1906	1907	1906	1907
Iron %, natural	56.70	55.00	56.70	55.00	Iron %, natural	52.80	51.50	52.80	51.50
Moisture	10.00	10.00	10.00	10.00	Moisture	12.00	12.00	12.00	12.00
Iron %, 212 deg.	63.00	61.12	63.00	61.12	Iron %, 212 deg.	60.00	58.52	60.00	58.52
Phosphorus %	0.045	0.045	0.045	0.045	Base price	$3.70	$4.20	$3.50	$4.00
Base price	$4.25	$5.00	$4.00	$4.75					

Under these conditions, the actual increase in price to furnacemen is 90 to 95c. for bessemer ores, and 60 to 65c. for non-bessemers.

Pig Iron.

The statistics collected by the American Iron and Steel Association give the production of pig iron in the United States in 1906 at 25,307,191 long tons, the greatest output ever reported, and considerably more than that of Germany and Great Britain combined. Comparing 1906

PIG IRON PRODUCTION OF THE UNITED STATES.
(In tons of 2240 lb.)

Kind of Iron.	1902		1903		1904		1905		1906	
	Tons.	*Per ct.*	*Tons.*	*Per ct.*	*Tons.*	*Per ct.*	*Tons.*	*Per ct.*	*Tons.*	*Per ct.*
Foundry and forge	5,176 568	29.1	5,281,200	29.3	4,358,295	26.4	5,837,174	25.4	5,714,492	22.6
Bessemer pig	10,393,168	58.3	9,989,908	55.5	9,098,659	55.2	12,407,116	54.0	13,840,518	54.7
Basic pig	2,038,590	11.4	2,040,726	11.3	2,483,104	15.1	4,105,179	17.9	5,018,674	19.8
Charcoal	....	......	504,757	2.8	337,529	2.0	352,928	1.5	433,007	1.7
Spiegel and ferro	212,981	1.2	192,661	1.1	219,446	1.3	289,983	1.2	300,500	1.2
Totals	17,821,307	100.0	18,009,252	100.0	16,497,380	100.0	22,992,380	100.0	25,307,191	100.0

with the previous year, it is seen that foundry and forge iron showed a decrease in 1906 of 122,682 tons. All other classes showed increases: bessemer, 1,433,402 tons; basic, 913,495; charcoal, 80,079; spiegel and

ferro, 10,517 tons. The total gain was 2,314,811 tons, or 10.1 per cent. The most marked proportional change was in basic iron, which increased 22.3 per cent., while the gain in bessemer pig was 11.6 per cent. Foundry and forge irons lost 2.1 per cent.; while there were increases of 22.7 per cent. in charcoal pig and of 3.6 per cent. in spiegeleisen and ferromanganese.

These changes emphasize once more the steadily growing importance of the basic open-hearth process in this country. Nearly one-fifth of the pig iron made in 1906 was basic iron and this compares with less than 10 per cent. only five years ago. The classification, however, does not show the full growth of the open-hearth process, since no inconsiderable part of the bessemer pig—which includes low-phosphorus iron—is used in the acid open-hearth furnaces.

The extraordinary growth of our iron production is shown by the fact that in 1906 it was very nearly three times as great as that of 1896, which was 8,623,127 tons. Consumption has grown quite as fast, for even with the great output of last year, it was necessary during the closing months of the year to import some iron to meet immediate requirements.

In addition to the 300,500 tons of spiegeleisen and ferromanganese reported, there were made in 1906 a total of 142 tons of ferrophosphorus, a decrease of 1101 tons from 1905; and 5000 tons of bessemer ferrosilicon, an increase of 2250 tons. The special low-phosphorus iron, included under bessemer pig in the statement, was 228,769 tons in 1906.

PIG IRON PRODUCTION ACCORDING TO THE FUEL USED.
(In tons of 2240 lb.)

Fuel used.	1900	1901	1902	1903	1904	1905	1906
Coke (a)	11,727,712	13,782,386	16,315,891	15,592,221	14,931,364	20,964,937	23,313,498
Anthracite and coke	1,636,366	1,668,808	1,096,040	1,911,347	1,228,140	1,674,515	1,535,614
Anthracite alone	40,682	43,719	19,207				25,072
Charcoal	339,874	390,147	378,504	504,757	337,529	352,928	433,007
Charcoal and coke	44,608	23,294	11,665	927			
Totals	13,789,242	15,878,354	17,821,307	18,009,252	16,497,033	22,992,380	25,307,191

(a) Under coke furnaces are included the very few which use raw bituminous coal. The fact that 92.1 per cent. of our iron in 1906 was made with coke—not counting the proportion of coke used in connection with anthracite—shows the predominant importance of that fuel in iron metallurgy.

The number of furnaces in blast on Dec. 31, 1906, was 340, against 323 on June 30, 1906, and 313 on Dec. 31, 1905. The number of furnaces in blast at the end of 1906 was larger than at the close of any year since 1889, when 344 furnaces were active. The size of furnaces has increased since then, however, a number of very large stacks having been built. At the close of 1906 there were 98 idle furnaces, as compared with 111 idle furnaces at the close of 1905. The number of furnaces actually in

blast in the second half of 1906 was 374, as compared with 361 in the first half. In 1905 the number in blast during the last half of the year was 349, against 334 in the first half.

PRODUCTION OF PIG IRON BY STATES.
(In tons of 2240 lb.)

States.	1903	1904	1905	1906
Massachusetts	3,265	3,149	15,987	20,239
Connecticut	14,501	8,922		
New York	552,917	605,709	1,198,068	1,552,659
New Jersey	211,667	262,294	311,039	379,390
Pennsylvania	8,211,500	7,644,321	10,579,127	11,247,869
Maryland	324,570	293,441	322,096	386,709
Virginia	544,034	310,526	510,210	483,525
Alabama	1,561,398	1,453,513	1,604,062	92,599
N. Car. and Georgia	75,602	70,156		
Texas	11,653	5,530	38,699	1,674,848
West Virginia	199,013	270,945	298,179	304,534
Kentucky	102,441	37,106	63,735	98,127
Tennessee	418,368	302,096	372,692	426,874
Ohio	3,287,434	2,977,929	4,586,110	5,327,133
Illinois	1,692,375	1,655,991	2,034,483	2,156,866
Michigan	244,709	233,225	288,704	369,456
Wisconsin and Minn	283,536	210,404	351,415	373,323
Missouri, Colo., and Wash	270,289	151,776	407,774	413,040
Total	18,009,252	16,497,033	22,992,380	25,307,191

On Dec. 31, 1906, there were 27 furnaces in course of erection and five furnaces were being rebuilt. Of the building furnaces two were located in New York, 11 were in Pennsylvania, two were in Alabama, three in Ohio, five were in Indiana, two were in Illinois, one was in Wisconsin and one was in Colorado. All these furnaces when completed will use coke or mixed anthracite coal and coke for fuel. Of the five rebuilding furnaces one was in Pennsylvania, two were in Alabama, one was in Illinois, and one was in Washington. When rebuilt four furnaces will use coke for fuel and one furnace will use charcoal. In addition there were 23 furnaces projected and one furnace partly erected on Dec. 31. If built all the projected furnaces will use mineral fuel.

PRODUCTION OF PIG IRON BY DISTRICTS.
(In tons of 2240 lb.)

	1903	1904	1905	1906
New England, N. Y. & New Jersey	782,350	880,074	1,525,094	1,952,288
Pennsylvania	8,211,500	7,644,321	10,579,127	11,247,869
Ohio, Illinois, Mich., Wisconsin and Minn	5,508,034	5,077,549	7,260,712	8,226,778
Maryland	324,570	293,441	332,096	386,709
Southern States	2,912,509	2,449,872	2,887,577	3,080,507
West of the Mississippi	270,289	151,776	407,774	413,040
Total	18,009,252	16,497,033	22,992,380	25,307,191

In classifying the production of pig iron by districts, the lines cannot be drawn with absolute strictness; but it may be said that the eastern

Pennsylvania furnaces which use local ores are fully offset by those in the Northeastern group, which work on Lake ores. In fact, the increase in that group in 1906 came almost wholly from New York, and was furnished by the stacks of the Lackawanna Steel Company, at Buffalo, which run on Lake Superior ores. Maryland has been placed by itself, because its output comes from a single group of furnaces, which uses Cuban and other imported ores.

Pennsylvania, Ohio, Illinois and the Northwest use Lake ores chiefly or wholly; and the furnaces of those districts made 77 per cent. of all our pig iron in 1906. The use of local ores has made certain advances in the three years of large output, and mines are being actively worked, especially in the Lake Champlain and Hudson river regions in New York; in New Jersey and in the Lehigh valley and Cornwall districts in Pennsylvania. The fact remains that the Lake Superior region is the main supply of our iron industry, and especially of that part of the production which is converted into steel in all of its forms; and the tendency is to an increase rather than a diminution of its importance.

CONSUMPTION OF PIG IRON IN THE UNITED STATES.
(In tons of 2240 lb.)

	1905	1906
Production	22,992,380	25,307,091
Imports	212,465	379,828
Total	23,204,845	26,686,919
Exports	49,221	83,717
Approximate consumption	23,155,624	26,603,202

In the table showing the consumption no account is taken of stocks on hand, for reason that in both years there were no surplus accumulations carried over, the iron remaining at furnaces being only that necessarily carried from day to day. The demand for consumption was too urgent to permit any piling of iron in furnace yards. No general report of stocks on hand has been made for several years. One reason for this is the change in iron-making methods. Over 80 per cent. of the pig iron is now made by the steel companies, which use the iron themselves for conversion into steel and finished forms; less than 20 per cent. being made by merchant furnaces for sale as pig iron. This merchant iron is chiefly foundry iron, only a limited quantity being bought by the steel companies from outside makers.

Steel.

The total production of steel in the United States as reported by the American Iron and Steel Association, has been as follows for six years past. The proportions of the various kinds of steel in 1906 were: Besse-

mer, 52.5; basic, 47.0; crucible and special, 0.5 per cent. Comparing 1906 with the previous year, the increase in bessemer steel was 1,333,878 tons, or 12.2 per cent.; in open-hearth, 1,999,622 tons, or 22.2 per cent. in crucible and special steels, 7304 tons or 6.6 per cent.; the total gain being 3,340,804 tons or 16.7 per cent. The ratio of steel to pig iron was 92.3 in 1906.

PRODUCTION OF STEEL IN THE UNITED STATES.
(In tons of 2240 lb.)

Kinds.	1900	1901	1902	1903	1904	1905	1906
Bessemer	6,684,770	8,713,302	9,138,363	8,577,228	7,859,140	10,941,375	12,275,253
Open-hearth	3,402,552	4,656,309	5,687,729	5,837,789	5,908,166	8,971,376	10,970,998
Crucible and special	131,250	103,984	121,158	112,238	92,581	111,196	118,500
Total tons	10,218,572	13,473,595	14,947,250	14,527,255	13,859,887	20,023,947	23,364,751
Total metric tons	10,382,069	13,689,945	15,186,406	14,756,691	14,081,645	20,344,330	23,738,587

PRODUCTION OF OPEN-HEARTH STEEL.
(In tons of 2240 lb.)

Year.	Acid.		Basic.		Year.	Acid.		Basic.	
		Per ct.	Tons.	Per ct.		Tons.	Per ct.	Tons.	Per ct.
1901	1,037,316	22.3	3,618,993	77.7	1904	801,799	13.6	5,106,367	86.4
1902	1,191,196	20.9	4,496,533	79.1	1905	1,155,648	12.9	7,815,728	87.1
1903	1,094,998	18.8	4,734,913	81.2	1906	1,321,613	12.0	9,649,385	88.0

All the bessemer steel in the United States is made by the acid process; the division of the open-hearth steel is shown in the accompanying table. The year 1906 showed once more the rapid advance of the open-hearth, and especially the basic open-hearth process. The major part of the increase in 1906 was in basic steel, and the tendency to the use of the basic process gives every sign of advancing. The proportion of open-hearth steel to the total production, which was 12.2 per cent. in 1890, increased to 19 in 1895; to 33.7 in 1900; to 44.8 in 1905, and to 47 in 1906; in a short time the open-hearth output will exceed the bessemer.

The geographical distribution of the steel production in 1906, disregarding the small quantity of crucible steel, was as follows:

	Bessemer.	Open-hearth.	Total.
Pennsylvania	4,826,725	7,710,949	12,537,674
Ohio	3,769,913	816,483	4,586,396
Illinois	1,685,056	884,472	2,569,528
Other States	1,993,559	1,559,094	3,552,653
Total	12,275,253	10,970,998	23,246,251

It is of interest to compare the steel output of the three more important producing nations for 1906, classed according to processes of manufacture.

(See accompanying table.) The output of the United States and Great Britain is in long tons; of Germany in metric tons. The production of the United States exceeded by 5,567,392 tons that of its two chief competitors combined. Putting the table in a different form, we have the accompanying statement as to the respective makes of acid and basic steel. The make of basic steel was in excess of that of acid; a result chiefly due to Germany with its great supplies of ore suitable for making basic pig iron.

STEEL PRODUCTION OF PRINCIPAL COUNTRIES.

	United States.	Germany.	Great Britain.
Acid converter	12,275,253	407,688	1,307,149
Basic converter	....	6,772,804	600,189
Total converter	12,275,255	7,180,492	1,907,338
Acid open-hearth	1,321,613	230,668	3,378,691
Basic open-hearth	9,649,385	3,534,612	1,176,245
Total open-hearth	10,970,998	3,765,280	4,554,936
Crucible and special	118,500	189,313	....
Total	23,364,751	11,135,085	6,462,274
Proportion steel to pig iron	92.3	89.2	63.7

MAKE OF ACID AND BASIC STEEL.

	Acid.		Basic.	
	Tons.	Per Ct.	Tons.	Per Ct.
United States	13,715,366	58.7	9,649,385	41.3
Germany	715,952	6.4	10,419,133	93.6
Great Britain	4,685,840	72.5	1,776,434	27.5
Total	19,117,158	46.7	21,844,952	53.3

Changes and Consolidations.

Not many new consolidations were recorded during 1906. The amalgamation of the Republic Iron and Steel Company in the South with the Tennessee Coal, Iron and Railroad Company, announced in 1905, was completed.

The consequent changes have not yet been fully carried out, but will be completed during 1907. They involve the concentration of work at certain points, the enlargement of some of the works owned by the companies, and other improvements which will help to increase production and reduce costs. The great innovation brought about will be a large increase in the production of steel from Southern iron, and the consequent diversion of a large quantity of Alabama pig from the foundry trade. Already the basic open-hearth steel plant at Ensley has proved successful; so much so, indeed, that its full capacity for a year ahead is engaged

for the manufacture of rails. This plant will be much enlarged and possibly supplemented by others at different points.

The most remarkable occurrence of 1906 is found in the transfer of the Hill iron-ore lands on the northern Mesabi to the United States Steel Corporation. This transfer was foreshadowed more than a year before, when negotiations had already been begun; but it was not completed until late in 1906. The terms of the deal provide for the rental of lands in question to the United States Steel Corporation on a royalty beginning at 85c. a ton and increasing by 2½c. yearly, with a high minimum. It is calculated that the average price paid by the lessee for ore in the ground will be a little over $1.00 a ton, which is the highest figure yet reached in any leases in Minnesota. It is, of course, provided that all ores mined from these lands shall be shipped to the Lake docks over the Great Northern tracks. Computations of the quantity of ore contained in these lands vary widely, for the reason that only a small portion of them have been proved by actual operations. There is no doubt, however, that some very large mines will be developed at an early date.

The conclusion of this negotiation was foregone when Mr. Hill anouced that the lands could only be disposed of in a single block. The United States Steel Corporation was then the only possible purchaser, since no other existing concern could handle so large a transaction. This transfer practically prevents the establishment of any effective competition with the Steel Corporation on a large scale. There is no other large block of iron ore, readily accessible, which could be secured by any other company. Under our present conditions no steel company is secure of its standing or future operations unless it controls its own supply of iron ore.

Increase in Capacity for Production.

New work for 1906 was directed largely toward increase in productive capacity. The number of blast furnaces under construction, recently completed, or for which contracts have been let, was 62, the yearly capacity of which, will be, 7,300,000 tons of pig iron. This does not include the increased capacity which will be derived from old furnaces reconstructed or otherwise renewed. If we add these, the work in hand in 1906 ought to give an increase of over 8,000,000 tons yearly. Not all of this new work will be available in 1907, but more than half of it will be ready to operate during the year and the remainder will come in in the early part of 1908. There is little doubt that by the close of that year our blast furnaces will be able to turn out between 33,000,000 and 35,000,000 tons yearly. In steel production also preparations are being made for a great increase in capacity. Prominent among these are the new open-hearth furnaces added to the Carnegie steel plants around Pittsburg, to the Illinois Steel

Works at Joliet and South Chicago, to the Ohio works of the American Steel and Wire Company, and to the new works at Gary.

The most important work under construction in 1906 was the new and extensive plant which the United States Steel Corporation is constructing at Gary, Ind. This is intended to be a model plant of the largest capacity, and will include 16 blast furnaces with a corresponding number of open-hearth steel furnaces, to which the molten iron will be fed direct from the blast furnaces. These will be accompanied by rolling mills which will convert the steel from the furnaces directly into rails, plates, structural forms and other finished shapes. Only a small portion of this plant was to be ready during 1907, and the whole of it will not be in operation until after the end of 1908.

Imports and Exports.

The figures in this section are as reported by the Bureau of Statistics of the Department of Commerce and Labor; the tables include only the more important items of exports and imports:

TOTAL VALUE. (a)

	1904	1905	1906
Exports	$128,455,613	$142,928,513	$172,555,588
Imports	21,621,970	26,392,728	34,827,132
Excess, exports	$106,833,643	$116,535,785	$137,728,456

(a) Including machinery.

UNITED STATES EXPORTS OF IRON AND STEEL.
(In tons of 2240 lb.)

	1900	1901	1902	1903	1904	1905	1906
Pig iron	286,687	81,211	27,487	20,379	49,025	49,221	83,317
Billets, blooms, etc	107,385	28,616	2,409	5,445	314,324	237,638	192,616
Bars	94,665	45,105	31,549	37,182	55,472	51,870	87,655
Rails	361,619	318,055	67,455	30,656	414,845	295,023	328,036
Sheets and plates	54,865	30,832	18,300	18,093	55,204	75,034	110,654
Structural steel	67,714	54,005	53,859	30,641	55,514	83,193	112,555
Wire	78,014	88,238	97,843	108,521	118,612	142,611	172,764
Wire-rods	10,652	8,165	24,613	22,360	20,073	6,514	5,896
Nails and spikes	43,379	29,881	35,994	42,664	45,112	47,756	59,462

UNITED STATES IMPORTS OF IRON AND STEEL.
(In tons of 2240 lb.)

	1900	1901	1902	1903	1904	1905	1906
Pig iron	52,565	62,930	625,383	599,574	79,500	212,465	379,828
Billets, blooms, etc	12,709	28,164	289,318	261,570	10,807	14,637	21,337
Scrap iron and steel	34,431	20,130	109,510	82,921	13,461	23,731	19,091
Bars	19,685	20,792	28,844	43,393	20,905	37,298	35,793
Rails	1,448	1,905	63,522	95,555	37,776	17,278	4,943
Wire-rods	21,092	16,804	21,382	20,836	16,206	17,616	17,999
Tin plates	60,386	77,395	60,115	47,360	71,304	65,740	56,983

The United States Steel Corporation.

A review of the year would be incomplete without the statement of the United States Steel Corporation, which controls nearly two-thirds of the finished iron and steel production of the country. The corporation furnishes full reports of its operation, which are summarized herewith, and form a valuable indication of the course of the iron trade during the year.

The capital account shows that on Dec. 31 there was outstanding $508,302,500 common stock and $360,281,100 preferred stock. The total bonded debt was $564,670,876, a reduction of $5,801,389 during the year. The general statement of earnings and expenses may be summarized as follows:

	1905	1906	Changes.
Gross receipts....	$585,331,736	$696,756,926	I $111,425,190
Operating expenses.......	$440,013,432	$517,083,955	I $77,070,523
General expenses.	18,570,374	22,907,679	I 4,337,305
Total.........	$458,583,806	$539,991,634	I $81,407,828

	1905	1906	Changes.
Net earnings...	$126,747,930	$156,765,292	I $30,017,362
Interest on investments, etc.	6,057,134	9,159,864	I 3,102,730
Total.....	$132,805,064	$165,925,156	I $33,120,092
Account, subsidiary Co.s...	13,017,406	9,300,883	D 3,716,523
Final net earnings..........	$119,787,658	$156,624,273	I $36,836,615

The increase in gross receipts last year was 19 per cent.; in net earnings, 23.7 per cent.; in final net earnings, 30.8 per cent. The disposition of the net earnings for 1906 is reported as follows: Net earnings, as above, $156,624,273; depreciation and replacement funds, $30,-657,335; interest and sinking fund, $27,747,850; adjustment of sundry accounts, $90,501; dividends, preferred stock, 7 per cent., $25,219,677; dividends, common stock, 2 per cent., $10,166,050; special appropriations, new work, $50,000,000; total, $143,881,413; surplus for the year, $12,742,860. The total undivided surplus, as of Dec. 31, 1906, amounted to $97,720,714, which represents the working capital of the company.

The expenditures for new property and construction during 1906 were: Gary plant, $4,720,159; other manufacturing properties, $12,-895,066; coal and coke properties, $1,089,379; iron-ore properties, $3,878,295; transportation properties, $9,227,466; miscellaneous, $344,-781; total, $32,155,146. The total expenditures under these heads, from the organization of the company, April 1, 1901, to the end of 1906 were $208,231,164.

The average number of employees in the service of all companies during 1906, in comparison with 1905, was as follows, the figures for 1905 being given in brackets: Manufacturing properties (130,614) 147,048;

coal and coke properties (20,883) 21,929; iron ore mining properties (12,068) 14,393; transportation properties (14,524) 16,638; miscellaneous properties (2,069) 2,449; total (180,158) 202,457; total annual salaries and wages ($128,052,955) $147,765,540.

The offer to the employees to subscribe for the preferred stock of the United States Steel Corporation was again renewed at the end of the

PRODUCTION OF THE U. S. STEEL CORPORATION.

	1904. Tons.	1905. Tons.	1906. Tons.
Iron Ore Mined—			
From Marquette Range	934,512	1,359,722	1,442,290
From Menominee Range	1,186,104	1,871,979	1,874,680
From Gogebic Range	1,271,831	1,671,747	1,465,375
From Vermilion Range	1,056,430	1,578,626	1,794,886
From Mesabi Range	6,054,210	12,004,482	14,068,617
Total	10,503,087	18,486,556	20,645,148
Coke Manufactured	8,652,293	12,242,909	13,295,075
Coal Mined, not including that used in making coke	1,998,000	2,204,950	1,912,144
Limestone Quarried	1,393,149	1,967,355	2,227,436
Blast Furnace Products—			
Pig iron	7,210,248	9,940,799	11,058,526
Spiegel	100,025	158,071	150,044
Ferro-Manganese and Silicon	59,148	73,278	58,807
Total	7,369,421	10,172,148	11,267,377
Steel Ingot Production—			
Bessemer Ingots	5,427,979	7,379,188	8,072,655
Open-Hearth Ingots	2,978,399	4,616,051	5,438,494
Total	8,406,378	11,995,239	13,511,149
Rolled and Other Finished Products for Sale—			
Steel Rails	1,242,646	1,727,055	1,982,042
Blooms, Billets, Slabs, Sheet and Tin Plate Bars	932,029	1,253,682	1,096,727
Plates	404,422	780,717	836,399
Heavy Structural Shapes	313,779	484,048	620,823
Merchant Steel, Skelp, Hoops, Bands and Cotton Ties	577,384	982,782	1,240,548
Tubing and Pipe	710,765	911,346	1,025,913
Rods	84,934	84,049	111,488
Wire and Products of Wire	1,226,610	1,283,943	1,399,717
Sheets—Black, Galvanized and Tin Plate	757,482	924,439	1,112,542
Finished Structural Work	357,488	404,732	643,622
Angle and Splice Bars and Joints	72,470	150,265	176,730
Spikes, Bolts, Nuts and Rivets	46,003	61,496	70,233
Axles	62,981	149,596	181,913
Sundry Iron and Steel Products	25,787	28,236	79,736
Total	6,792,780	9,226,386	10,578,433
Spelter	29,963	29,781	28,884
Copperas (Sulphate of Iron)	15,805	20,040	21,933
	Bbl.	*Bbl.*	*Bbl.*
Universal Portland Cement	539,951	1,735,343	2,076,000

year 1906 on substantially the same conditions as offered in previous years, except the price was fixed at $102 per share. Under the offer subscriptions were received from 14,169 employees for a total of 27,032 shares. At Dec. 31, 1906, there were 15,568 employees who had purchased preferred stock under offers made by the corporation, and who on that date either held the certificates for the stock or were making monthly payments on account of the purchase price thereof.

An increase was made on Jan. 1, 1907 (on March 1, 1907, in case of the coke companies), in the wages and salaries of approximately 131,000 employees, or 65 per cent. of the total employees of all companies. The advance applied to practically all employees who are paid on basis of day

rates, and to a considerable number of those paid on basis of a monthly salary. The average rate of increase in wages and salaries of the employees affected equaled 6.6 per cent. The amount of increase in the total pay-roll attributable to above advance will be approximately \$6,000,000 per annum. The average rate of pay received by all employees is now the highest paid at any time since the organization of the corporation.

Following the practice heretofore adopted, a substantial amount was distributed as a bonus for the year 1906 to a large number of employees who merited the same. Included in the distribution were about 21,000 shares of common stock which were previously purchased in the market for that purpose, and which will not be delivered until the expiration of three years and the compliance with various conditions named for the protection of the companies. From the bonus fund to be ascertained at the end of 1907 in the usual way it is proposed to establish a separate fund to be known as the pension fund and to be used for pension purposes.

In common with practically all lines of industry throughout the United States, the business of the subsidiary companies during the year 1906 reached a new high level. The average prices received during the year for all steel products shipped to the domestic trade exceeded by 5.3 per cent. the average received in 1905, and were about 8 per cent. lower than the prices which prevailed in 1902. The condition of the trade has been such that prices of many of the commodities produced could have been advanced; and as wages have been largely increased and freight rates raised, it has been argued by some of the manufacturers of steel that prices of their manufactured commodities should be correspondingly advanced; but it was decided by the management of this corporation and of its subsidiary companies to make no changes in prices.

The orders for steel products of all kinds, both for immediate and for future shipments, were received in large volume up to the close of the year. Since Jan. 1, 1907, the orders received have been somewhat less than they were during the months immediately preceding that date, but they are larger than they were during the same months in 1906, and have about equaled the productive capacity of the mills, notwithstanding the tonnage of unfilled orders on the books at the close of the year equaled 8,489,718 tons of all kinds of manufactured steel and iron products, in comparison with 7,605,086 tons at the close of 1905.

The export business of the subsidiary companies continues to show material improvement. Export shipments during the year amounted to 1,079,319 tons of various products, an increase of 13 per cent. over the tonnage shipped in the previous year, while the gross receipts exceeded those of 1905 by 28 per cent. Since the export business of all the subsidiary companies was concentrated, late in 1903, in the hands of a single

selling organization, there has been steady and systematic development in all foreign markets.

With respect to the Hill ore lands, the report says that the lease is to be taken by the Great Western Mining Company, a subsidiary company of the United States Steel Corporation, and the performance of the obligations assumed by that lessee is to be guaranteed by the United States Steel Corporation. Under the lease the royalty to be paid for the ore is \$1.65 per gross ton, for ore containing 59 per cent. of metallic iron, delivered in docks at head of Lake Superior. If ore grades higher or lower than 59 per cent. in metallic iron, the royalty will be increased or decreased according to a fixed scale. The above royalty of \$1.65 per ton is for ore to be shipped in 1907, and the base price increases at the rate of 3.4c. per ton each succeeding year. The minimum to be mined and shipped is 750,000 tons in 1907 and increases by 750,000 tons per year until it reaches 8,250,000 tons, and thereafter the annual minimum continues on that basis. The lease will continue until the ore is exhausted unless on Jan. 1, 1915, the lease is terminated, under the option reserved to the lessee.

The Iron and Steel Markets.

The special reviews given herewith cover the important markets of the country. Pittsburg, of course, is the chief primary market. Cleveland is the center of an active producing and distributing region, while Chicago is the chief supply point for a great western and northwestern territory. Birmingham is the center for the Alabama iron country and the most important point in the foundry-iron trade. These and the seaboard markets cover practically the whole country.

They all tell substantially the same story of a market opening strongly; then a brief period of hesitation while crop prospects were uncertain; finally of strong and steady buying for a series of months, culminating, toward the end of the year, in a scramble by consumers to secure their supplies for the first and second quarters of the new year. The leading interests made an effort throughout the year to keep prices steady, and prevent irregular and too rapid advances. In this they were largely successful, and there was comparatively little increase in quotations.

The railroads were good customers throughout the year. While the work done on new lines was not very great, there was a great demand for rails for renewals, second tracks, etc., for new cars and locomotives, and for bridge work. The larger use of steel cars demanded a great quantity of structural shapes and plates. There was also an excellent demand for rails and equipment for electric railroads. Structural steel for new buildings, bars for machine and other work all helped to make up the great demand of the year.

Pittsburg. (By S. F. Luty.)—The year 1906 opened with the iron and steel markets strong in all lines. One of the remarkable features was that, despite the advance in the prices of pig iron, there were no material changes in the prices of finished steel products until the fourth quarter. A runaway market was prevented by conservative management all the way through. In the third and fourth quarters fancy prices were paid in some instances for small lots of both foundry and bessemer iron, but the market was not affected. Sales of foundry iron were made at higher prices than for bessemer in the closing months of the year.

Mild weather in midwinter permitted heavy production and the market in the first quarter was steady and satisfactory. In the latter part of March a lack of confidence on the part of both consumer and producer was developed. A drop of $3 a ton in the price of bar iron threatened to disturb conditions, but this did not last long, and the second half opened with the whole market stronger.

Buying was heavy, and the entire production for the rest of the year was soon under contract. The market for 1907 opened in August, and while it was not the earliest opening on record, it may be called the most sudden and unexpected. Before the beginning of the fourth quarter every producer in the Pittsburg and Valley districts was practically sold up to July 1, 1907. There was no heavy buying for third quarter, consumers evidently preferring to wait until the opening of the new year before providing for their second-half requirements.

Strength was given to the pig iron market on Jan. 15, when the United States Steel Corporation bought 115,000 tons of bessemer iron from outside producers. A leading independent producer furnished a record of all sales of bessemer iron in lots of 1000 tons or more for the entire year, and the average monthly prices from which contracts for some finished steel products based on bessemer pig-iron rates were adjusted. The tonnages and prices at Pittsburg in round numbers are as follows: January, 120,000, $18.20; February, 38,000, $18.15; March, 42,000, $18.15; April, 95,000, $18.10; May, 150,000, $18.10; June, 75,000, $18.25; July, 82,000, $18.45; August, 200,000 $18.85; September, 50,000, $19.30; October, 50,000, $20.95; November, 35,000, $22.85; December, 10,000, $23.75.

The demand for pig iron of all grades was so urgent in the last half that importations were necessary, and prices paid were several dollars higher than the average prices given in the accompanying table. Some small lots of Southern No. 2 foundry sold in this market at $23, Birmingham, equal to $27.60, Pittsburg. Northern bessemer and foundry sold above $25, Pittsburg, but these were only for small lots and were not considered in preparing the general averages.

Although the price of pig iron was high during the year, there was no change in the price of structural material, the rate remaining on the basis of 1.70c. for beams and channels. Tank plate remained at 1.60c. until December and steel bars were 1.50c. until advanced to 1.60c. in November. Wire prices were irregular until November, when an advance of $1 a ton was ordered, followed in December by an increase of $2 a ton.

There were no labor disturbances of any consequence during the year to interfere with production. The strike of the International Association of Bridge and Structural Iron Workers against the American Bridge Company and members in this district of the National Erectors' Association was continued, but was not effective. Strikes of iron molders and machinists were threatened, but the labor organizations could not muster enough strength to carry out the plans.

AVERAGE PRICES AT PITTSBURG, 1906.

Month.	Pig Iron.			Ferro Manganese.	Steel.					Nails.	
	Bessemer	No. 2 Foundry.	Gray Forge.		Bessemer Billets.	Rails.	Sheets No. 28.	Tank Plate.	Steel Bars.	Wire Per Keg.	Cut Per Keg.
	$	$	$	$	$	$	Cts.	Cts.	Cts.	$	$
January......	18.20	18.10	17.35	135.00	26.50	28.00	2.40	1.60	1.50	1.85	1.75
February.....	18.15	18.10	17.10	150.00	27.00	28.00	2.40	1.60	1.50	1.85	1.80
March........	18.15	17.85	17.00	140.00	27.50	28.00	2.40	1.60	1.50	1.85	1.80
April.........	18.10	17.35	16.60	125.00	27.00	28.00	2.40	1.60	1.50	1.85	1.80
May.........	18.10	17.35	16.60	105.00	26.50	28.00	2.40	1.60	1.50	1.85	1.80
June.........	18.25	17.35	16.35	90.00	27.00	28.00	2.50	1.60	1.50	1.85	1.75
July.........	18.45	17.60	16.60	85.00	27.50	28.00	2.50	1.60	1.50	1.80	1.75
August.......	18.85	18.85	17.85	85.00	28.00	28.00	2.50	1.60	1.50	1.80	1.75
September....	19.30	19.85	18.35	83.00	28.50	28.00	2.50	1.60	1.50	1.85	1.75
October......	20.95	21.10	19.35	78.00	28.50	28.00	2.50	1.60	1.50	1.85	1.90
November....	22.85	23.60	22.35	82.00	28.50	28.00	2.60	1.60	1.60	1.90	1.95
December....	23.75	24.35	22.85	83.00	29.50	28.00	2.60	1.70	1.60	2.00	2.05

Cleveland. (By George H. Cushing.)—The Cleveland market opened the year 1906 with a brisk demand for all commodities. In the pig iron trade prices, on the foundry grades, ranged, according to delivery, from $14 to $15.50 in the Valleys until well along toward the end of the first half of the year. In May and June there began to appear evidences that prices were going to advance and some of the more astute began to lay in a supply, on a speculative basis.

Shortly after the turn of the half-year the price of No. 2 Northern foundry began to mount and it was soon selling at $18 in the Valleys, and higher for immediate delivery, with almost as high prices on contracts. It was not far along in the third quarter when the price of foundry iron began to climb and by the opening of the fourth quarter it was a runaway market. By the close of the year No. 2 foundry for spot shipment was selling on the basis of $25@26 for immediate shipment, at about the same price for first-quarter delivery in 1907, and at $23@24 for second-quarter. At the same time sales were made for second half of 1907 delivery at $21@22

at furnace and buyers were in a scramble to obtain the amounts they might need.

In steel it was apparent the Pittsburg mills had increased the percentage of their output set aside for the use of the car companies, the railroads and the Lake shipbuilders. This was due to extraordinary demands in all these quarters, especially, as far as Cleveland is concerned, in the shipbuilding trade. This lessened the amount going into general lines, where the demand was exceptionally good, despite labor difficulties. The last half of the year saw many consumers in this territory going into the East, where small mills had reserved part of their capacity for such an emergency, and paying premiums for immediate delivery. At the close of the year all of these conditions were aggravated and there was enough business on the books of the various companies to assure prosperity through the new year.

Chicago. (By E. Morrison.)—General conditions of the iron trade in the year 1906 were satisfactory to Chicago furnace agents and dealers in finished products. The first half of the year was sluggish for pig iron, though sales of iron and steel products were heavy, particularly of railroad supplies. But the middle of July saw a remarkable recovery from the slump of the market caused by the dissolution a month previous of the Southern Furnace Association, and an unprecedented midsummer activity followed. The market grew stronger week by week, and prices advanced to such an extent that by December many smelters refused to buy except for the most urgent needs.

The following table shows the range of prices during the year, compared with the range in 1905:

	1905		1906			1905		1906	
	Highest	Lowest	Highest	Lowest		Highest	Lowest	Highest	Lowest
L Superior Charcoal......	$20.50	$16.50	$26.50	$19.00	Connellsville Coke.......	$6.40	$5.15	$6.90	$5.40
Northern No. 2 Foundry..	19.75	15.50	27.00	18.00	Bar Iron..................	1.90c.	1.50c.	1.85c.	1.665c.
Southern No. 2 Foundry..	18.65	14.65	26.90	16.90	Structural Material (*a*)...	1.865c	1.665c.	1.865c.	1.865c.

(*a*) Beams and channels, 3 in. to 15 in., and angles 3 in. to 6 in., ¼ in. or heavier.

Alabama. (By L. W. Friedman.)—Even with difficulties—labor troubles, labor shortage and last, but by no means least, a railroad car shortage—the year 1906 was a most favorable one to the iron and steel industry in Alabama. The production, in the face of serious obstacles, was satisfactory. Transportation, probably, was the greatest difficulty of the year. When the demand for the manufactured product became brisk, and there was a desire to increase production, there was a lack of cars and locomotive power was short, causing delay in handling not only the raw material, but the finished product, blocking the production at both ends. The labor problem asserted itself early also, and while there was

hope for relief in the importation of desirable labor the task was not so easy and the men brought into the district were not numerous.

As to quotations there was a good price for the product through the year. At the opening, there was some iron on the yards, but not much. In June and July there was a little depression. The market was startled when it was announced that two of the larger companies had cut prices on iron. It transpired, however, that the original cutters of price were the only ones to sell iron at the low price. With iron down to $12 per ton for No. 2 foundry, at that time, the lowest price was reached for the year. The high-water mark was reached after October, and in November No. 2 foundry, immediate delivery, was not to be had, with $21 to $17.50 for iron to be delivered within three and six months.

The Seaboard Iron Markets.—The two more important iron markets on the Eastern seaboard are Philadelphia and New York. One of the chief points in 1906 was the strong and steady demand for structural steel. In the cities an unusual amount of construction of large buildings was in progress, or planned, in which steel is needed. There was also a great number of smaller buildings in which the use of a certain quantity of steel is required by law. Bridge work throughout the country was active. The nominal, or base, prices of structural shapes were steady, but in the second half of the year—especially in the fourth quarter—there were real advances in the shape of premiums to secure delivery, while jobbers advanced their prices largely on small lots.

Foundries throughout Eastern territory were well employed. By the end of November the market was practically bare of spot iron, and only small lots from jobbers and second hands could be had. Prices were wild and irregular for a time, and considerable importations of Scotch and Middlesbrough pig from England failed to meet the demand. December was marked by heavy contracting, running over the first half of 1907, and even into the third quarter. Closing prices were high, Northern pig selling at tidewater, for 1907 delivery, at about $25.50 for No. X foundry; $24.50 for No. 2 X; $23.50 for No. 2 plain; $23 for basic pig. Southern iron for first quarter was about on the basis of $24.50 for No. 2 foundry, New York or Philadelphia. Imported iron sold at $23.50, ex-ship, for Middlesbrough and $24.50 for Scotch pig. Although there were some resales, there does not appear to have been any large quantity of speculative iron on the market.

Lake Iron Ore Trade.—The figures compiled from the dock reports by the *Iron Trade Review*, of Cleveland, O., show that in the season of 1906 a total of 85.5 per cent. of the iron ore shipped by water from Lake Superior mines reached the furnaces through Lake Erie ports. Receipts at those ports for the season, with stocks on docks Dec. 1, were, in long tons:

IRON ORE AT LAKE ERIE PORTS.

Ports.	Receipts.		Stocks, Dec. 1.	
	1905	1906	1905	1906
Toledo	1,006,855	1,423,741	368,024	281,000
Sandusky	51,202	35,847	52,977	17,467
Huron	825,278	778,453	208,023	245,499
Lorain	1,605,823	2,191,965	271,695	336,321
Cleveland	5,854,745	6,604,661	1,330,619	1,224,606
Fairport	2,008,621	1,861,498	759,961	590,783
Ashtabula	6,373,779	6,833,352	1,589,951	1,631,312
Conneaut	5,327,552	5,432,370	976,976	1,057,424
Erie	2,112,476	1,986,539	564,961	552,631
Buffalo	3,774,928	4,928,331	315,780	315,412
Total	28,941,259	32,076,757	6,438,967	6,252,455

The total shipments of lake ore by water for the season were 37,513,595 long tons. Receipts at Lake Erie ports, as shown above, were 32,076,757 tons. The balance of 5,436,838 tons is accounted for by deliveries at other points, such as South Chicago, Milwaukee and Detroit. The season movement to furnaces through the Lake Erie ports is figured as follows: Stocks on docks, May 1, 1,791,080 tons; receipts for the season, 32,076,747; total supply, 33,867,847 tons. The stocks on docks Dec. 1, were 6,252,455 tons, leaving deliveries to furnaces 27,615,392 tons, an average of 3,945,056 tons per month through the season.

Iron and Steel Production of the World.

The total production of pig iron in 1906 showed an increase over 1905 of 5,020,078 tons, or 9.3 per cent., a gain chiefly due to the increased output of the United States and Germany. The United States alone furnished 43.5 per cent.; and the three leading producers—the United States, Germany and Great Britain—together, 81.9 per cent. of the world's supply of pig iron.

PIG IRON PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1900	1,311,949	1,161,180	87,612	2,714,298	7,549,665	23,990	2,296,191
1901	1,300,000	765,420	248,896	2,388,823	7,785,887	25,000	2,869,306
1902	1,335,000	1,102,910	325,076	2,427,427	8,402,660	24,500	2,597,435
1903	1,355,000	1,299,211	269,665	2,827,668	10,085,634	28,250	2,486,610
1904	1,369,500	1,307,399	274,777	2,999,787	10,103,941	27,600	2,978,325
1905	1,372,300	1,310,290	475,491	3,077,000	10,987,623	31,300	2,125,000
1906	1,403,500	1,431,160	550,618	3,319,032	12,478,067	30,450	2,350 000

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1900	289,788	526,868	9,003,046	14,009,870	625,000	39,599,457
1901	294,118	528,375	7,977,459	16,132,408	635,000	40,950,692
1902	330,747	524,400	8,653,976	18,003,448	615,000	44,342,579
1903	380,284	506,825	8,952,183	18,297,400	625,000	47,113,730
1904	386,000	528,525	8,699,661	16,760,986	633,000	46,069,501
1905	383,100	531,200	9,746,221	23,340,258	655,000	54,054,783
1906	387,500	552,250	10,210,178	25,712,106	650,000	59,074,861

STEEL PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1900	1,145,654	655,199	23,954	1,565,164	6,645,869	115,887	2,217,752
1901	1,142,500	526,670	26,501	1,425,351	6,394,222	121,300	2,230,000
1902	1,143,900	776,875	184,950	1,635,300	7,780,682	119,500	2,183,400
1903	1,146,000	981,740	181,514	1,854,620	8,801,515	116,000	2,410,938
1904	1,195,000	1,069,880	151,165	2,080,354	8,930,291	113,800	2,811,948
1905	1,188,000	1,023,500	403,449	2,110,000	10,066,553	117,300	1,650,000
1906	1,195,000	1,185,660	515,200	2,371,377	11,135,085	409,000	1,763,000

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1900	144,355	300,536	5,130,800	10,382,069	400,000	28,727,239
1901	122,954	269,897	5,096,301	13,689,173	405,000	31,449,869
1902	163,564	283,500	5,102,420	15,186,406	412,000	34,972,497
1903	199,642	317,107	5,114,647	14,756,691	418,000	36,298,414
1904	193,759	333,522	5,107,309	13,746,051	415,000	36,148,079
1905	237,864	340,000	5,983,691	20,354,291	426,000	43,900,648
1906	251,600	351,900	6,565,670	23,738,587	420,000	49,902,079

The increase in steel production in 1906, as compared with 1905, was 6,001,431 tons, or 12 per cent. The United States made 47.6 per cent., and the three leading producers together 83 per cent. of the total steel. The ratio of steel to pig iron production was 92.3 for the United States; 89.2 for Germany; 63.7 for Great Britain; and 84.5 for the world's total.

Belgium.—Pig iron production in Belgium is reported as follows, in metric tons:

	1903	1904	1905	1906
Foundry iron	....	99,350	98,170	101,430
Forge iron	....	224,410	206,309	226,900
Steel pig	....	963,840	1,006,641	1,103,130
Total	1,216,500	1,287,400	1,311,120	1,431,460

At the close of 1906 there were 42 blast furnaces in Belgium, of which 38 were in blast and two idle. The average number in blast during the year was 37, showing an output of 38,688 tons per furnace in blast for last year. The foreign trade of Belgium for three years was as follows, in metric tons:

	Imports.			Exports.		
	1904	1905	1906	1904	1905	1906
Pig iron	347,455	507,970	694,530	39,708	41,295	31,445
Wrought iron	88,937	97,864	83,643	529,284	628,473	530,119
Steel	231,801	230,169	259,077	387,063	395,229	245,101
Total	668,193	936,003	1037,250	956,055	1,044,997	807,665

These figures include both raw and finished products.

Canada.—The production of pig iron in Canada, as collected by the American Iron and Steel Association, in 1906 amounted to 541,957 long tons, of which 525,716 tons were made with coke, 16,021 tons with charcoal, and 220 tons with electricity.

PIG IRON PRODUCTION IN CANADA.

	1902	1903	1904	1905	1906
Foundry and forge	....	....	....	146,698	130,120
Bessemer	....	....	....	149,203	165,609
Basic	....	....	....	172,102	246,228
Total	319,557	265,418	270,942	468,003	541,957

The total increase in 1906, as compared with 1905, was 15.8 per cent. Canada has not made spiegeleisen or ferro-manganese since 1899.

On Dec. 31, 1906, Canada had 15 completed blast furnaces, of which eight were in blast and seven were idle. Of the total 12 usually use coke for fuel and three use charcoal. In addition one furnace, to use coke, was being built and three coke furnaces were partly erected on Dec. 31. Work on the partly erected furnaces was, however, suspended some time ago.

The ore charged into the blast furnaces in 1906 totaled 1,204,473 short tons, of which 221,733 tons were Canadian ore, and the balance, or 982,740 tons, was imported. The production of pig iron attributable to Canadian ore amounted to 104,660 tons, which is a marked increase over the previous year, when the production amounted to only 68,170 tons. Besides the above quantity of Canadian iron ore charged into the furnaces, 74,778 tons were exported, which brings the total of iron ore produced in Canada in 1906 to 296,511 tons.

The new furnace of the Atikokan Iron Company, at Port Arthur, was nearly completed during 1906, and will be in operation in 1907. The works of the Dominion Steel Company at Sydney, Cape Breton, were active during the year. The most important iron ore producer in Canada is the Helen mine on the Michipicoten range. The following are the shipments by years, in long tons:

1900	1901	1902	1903	1904	1905	1906
53,470	232,505	298,430	203,413	118,355	169,527	121,555

This ore is used in the blast furnaces at Sault Ste. Marie, chiefly. The total to the end of 1906 was 1,197,255 tons. Developments con-

tinued on the Moose Mountain mines, but no considerable quantity of ore was shipped.

France.—Pig iron and steel production in France is reported as in the accompanying table, in metric tons:

PIG IRON PRODUCTION IN FRANCE.

	1902	1903	1904	1905	1906
Foundry	....	....	....	635,672	591,275
Forge	....	....	....	705,691	741,571
Bessemer	....	....	....	160,411	149,971
Basic	....	....	....	1,530,671	1,784,726
Special irons	....	....	....	44,267	51,489
Total	2,427,427	2,827,668	2,999,787	3,076,712	3,319,032

STEEL PRODUCTION IN FRANCE.

	1902	1903	1904	1905	1906
Wrought iron	625,826	595,831	554,632	669,841	736,918
Steel ingots	1,635,300	1,854,620	2,080,554	2,210,284	2,371,377
Finished steel	1,231,652	1,317,400	1,482,708	1,442,071	1,454,456

Of the wrought iron in 1906, sheets were 90,351 tons; bars and merchant iron, 646,567 tons. The finished steel reported in 1906 was: Castings, 30,878; forgings, 30,715; sheets and plates, 294,714; rails, 338,407; merchant and structural steel, 759,742 tons. The ratio of steel to pig iron production was 71.4 in 1906.

Germany.—The figures collected by the German Iron and Steel Union give the production of pig iron for four years as follows, in metric tons:

	1903		1904		1905		1906	
	Tons.	Per cent	Tons.	Per cent	Tons.	Per cent	Tons.	Per cent.
Foundry iron	1,798,773	17.8	1,865,599	18.5	1,905,668	17.3	2,108,684	16.8
Forge iron	859,253	8.6	819,239	8.1	827,498	7.5	854,536	6.9
Steel pig	703,130	7.0	636,350	6.3	714,335	6.5	943,573	7.6
Bessemer pig	446,701	4.4	392,706	3.9	425,237	3.9	482,740	3.9
Thomas pig	6,277,777	62.2	6,390,047	63.2	7,114,885	64.8	8,088,534	64.8
Total	10,085,634	100.0	10,103,941	100.0	10,987,623	100.0	12,478,067	100.0

The increase in 1906, as compared with 1905, in foundry iron was 203,016 tons; forge iron, 27,038; steel pig—which includes spiegeleisen, ferromanganese, ferrosilicon and similar alloys—229,238; bessemer pig, 57,503; Thomas, or basic pig, 973,649, the total gain being 1,490,444 tons, or 13.6 per cent.

Germany maintained its lead over Great Britain in 1906. The actual consumption of pig iron is estimated as follows for the year 1906, in metric tons: Pig iron production, 12,478,067; imported as pig iron, 497,240; pig iron equivalent of steel, etc., imported, 384,100; total supplies, 13,-359,407; pig iron exported, 613,527; pig iron equivalent of steel, etc., exported, 4,467,041; total deductions, 5,080,568; approximate consumption, 8,278,839. The estimated consumption was 134.96 kg., and the production of pig iron 203.43 kg. per inhabitant of the German Empire.

The Union also gives the following statement of the production of steel in Germany for the year 1905; the figures are in metric tons:

	1905				1906			
	Acid.		Basic.		Acid.		Basic.	
	Tons.	Per cent.	Tons.	Per cent.	Tons.	Per cent.	Tons.	Per cent.
Converter ingots....	424,196	4.2	6,203,706	61.6	407,688	3.6	6,772,804	60.8
Open-hearth ingots..	165,930	1.7	3,086,590	30.7	230,668	2.1	3,534,612	31.8
Special steels........	65,369	0.6	120,762	1.2	77,596	0.7	111,717	1.0
Total..........	655,495	6.5	9,411,058	93.5	715,952	6.4	10,419,133	93.6

Of the total of 11,135,085 tons in 1906 there was 64.5 per cent. made in the converter and 35.5 per cent. in the open hearth furnace.

The production compares with that of three previous years as follows:

Kind.	1903	1904	1905	1906
Acid steel..........	613,399	610,697	655,495	715,952
Basic steel.........	8,188,116	8,319,594	9,411,058	10,419,133
Total............	8,801,515	8,930,291	10,066,553	11,135,085

A large part of the finished steel produced was under control of the Steel Works Union, or combination, the output of which during 1906 amounted to 11,079,000 tons of finished and semi-finished products, being an increase of 2,127,048 tons over 1905. The production of tubes advanced 45 per cent.; forgings, railway axles, etc., 31 per cent.; bars, 29 per cent.; plates, sheets, shapes and semi-finished steel; each 22 per cent.; rolled rods and wire, 21 per cent., and railway material, 19 per cent.

Imports and exports of iron ore for the year were as follows:

	1904	1905	1906
Imports........................	6,061,127	6,085,196	6,730,636
Exports........................	3,440,846	3,698,563	3,212,977

The imports of iron ore are chiefly from Sweden and Spain. The exports are largely of minette ores from Luxemburg to France and Belgium.

Exports and imports of iron and steel in Germany were as follows, in metric tons:

	1904	1905	1906
Exports	2,770,276	3,349,968	3,619,796
Imports	344,967	322,907	690,081

As compared with the previous year there were considerable increases both in exports and imports, in 1906.

Mexico.—The production of Mexico is still limited, and the new iron-making enterprises under way are making slow progress.

Japan.—The production of iron ore in 1906 approximated 71,000 metric tons. The government iron works, the largest in Japan, draw their chief supplies of ore from China. Some iron deposits have been reported in the island of Formosa.

Russia.—Some recovery in iron and steel is reported though the industry is still affected by the unsettled condition of the country. It is difficult to secure authentic figures, but estimates give the following production:

	1902	1903	1904	1905	1906
Iron ore	3,987,303	4,218,600	5,272,300	4,050,000	4,580,000
Pig iron	2,597,435	2,486,610	2,978,325	2,125,000	2,350,000
Steel ingots	2,183,400	2,410,938	2,811,948	1,650,000	1,763,000
Steel rails	382,152	332,367	401,541	275,000	315,000

The industry in Poland and South Russia is still in an unsettled state, but the Ural district was comparatively undisturbed.

Spain.—Exports of iron ore in 1906 were 9,311,325 metric tons, an increase of 720,842 tons over the previous year. The production of iron and steel in Spain for five years was as follows, in metric tons, 1906 being estimated:

	1902	1903	1904	1905	1906
Pig iron	330,747	380,284	386,000	383,100	387,500
Wrought iron	53,252	53,288	53,177	52,250	57,100
Bessemer steel	103,389	105,263	93,100	113,664	116,200
Open-hearth steel	60,175	94,379	100,659	124,200	135,400
Total steel	163,564	199,642	193,759	237,864	251,600

The exploitation of the iron sierras was active, owing to the high prices paid for ores.

Sweden.—Iron and steel production showed only moderate increases. The government has taken in hand the regulation of iron ore imports, and proposes to acquire a controlling interest in the great mines of Swedish Lapland. These mines are to become State property in 25 years, and during that period the quantity to be exported each year will be strictly prescribed. The object is to prevent too early exhaustion of the iron ore reserves needed by the Swedish industry.

United Kingdom.—The British Iron Trade Association has reported the production of pig iron for four years as follows, in long tons:

	1903	1904	1905	1906
Forge and foundry..	3,875,826	3,841,975	4,276,943	4,587,606
Bessemer..........	3,700,422	3,362,883	4,070,222	3,990,820
Basic..............	991,610	1,192,120	1,057,999	1,263,317
Spiegel and ferro...	183,346	165,680	187,573	307,645
Total............	8,811,204	8,562,658	9,592,737	10,049,388

The total number of blast furnaces standing was 517, and the average number working in 1906 was $367\frac{3}{4}$. This makes the average output per furnace 27,598 tons. The largest average make was 45,127 tons, in the Cleveland district; the smallest was 14,158 tons, in Derbyshire.

The make of steel for four years is reported by the Association as follows:

	1903		1904		1905		1906	
	Tons.	Per cent.	Tons.	Per cent.	Tons.	Per cent.	Tons.	Per cent.
Open-hearth........	3,124,083	62.0	3,245,346	64.6	3,879,748	65.9	4,554,936	70.5
Bessemer...........	1,910,018	38.0	1,781,533	35.4	2,009,712	34.1	1,907,338	29.5
Total..........	5,034,101	100.0	5,026,879	100.0	5,889,460	100.0	6,462,274	100.0

The table shows a steady gain in the proportion of open-hearth steel.

The division of the make according to the acid and basic processes for two years was as follows:

	1905			1906		
	Acid.	Basic.	Total.	Acid.	Basic.	Total.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
Open-hearth....................	3,084,510	795,238	3,879,748	3,378,691	1,176,245	4,554,936
Converter.......................	1,117,731	891,981	2,009,712	1,307,149	600,189	1,907,338
Total	4,202,241	1,687,219	5,889,460	4,685,840	1,776,434	6,462,274

There was some decrease in 1906 in the proportion of basic steel, chiefly in converter steel.

A partial report of the make of finished steel for two years is as follows:

	1905		1906	
	Bessemer.	Open-hearth.	Bessemer.	Open-hearth.
Blooms and billets	286,082	402,535	277,845	498,656
Rails	951,552	108,953	854,740	94,626
Bars, including tin-plate bars	288,980	839,415	242,706	939,087
General merchant steel and shapes	187,973	....	82,246	198,389
Plates and sheets	....	1,765,111	....	1,734,446
Total	1,714,587	3,116,014	1,457,537	3,465,204

This statement omits some important items, such as forgings and tires.

Imports of iron ore into Great Britain in 1904 were 6,100,556 long tons; in 1905, 7,344,786; in 1906, 7,823,086 tons. Spain is the chief source of supply, furnishing 5,949,361 tons in 1906. The British foreign trade is of great importance, and is more extended than that of any other nation. Its total value, including machinery, is estimated by the Board of Trade returns as follows, for four years:

	1903	1904	1905	1906
Exports	£54,741,296	£53,587,013	£60,524,755	£77,298,496
Imports	8,662,481	12,529,212	13,128,270	13,487,322

The total values of the exports in 1906 were made up as follows: Iron and steel, £39,880,563; machinery, £28,732,693; new ships, £8,685,240. The values of the imports were: Iron and steel, £8,360,135; machinery, £5,127,187. The chief items of the iron and steel exports were as follows, in long tons:

	1903	1904	1905	1906
Pig iron	1,065,380	810,934	981,891	1,664,442
Wrought iron	203,619	170,505	183,406	200,316
Sheets	....	385,408	407,021	443,131
Plates	161,722	152,337	204,503	275,058
Rails	604,076	525,371	546,644	463,240
Steel shapes, etc	156,821	122,930	151,809	275,143
Tin plates	292,800	359,634	354,951	375,414
All other kinds	....	735,723	891,290	992,102

The chief items of the imports in 1906 were: Steel billets and blooms, 486,029 long tons; structural steel, 138,660; bars and shapes, 55,025;

wrought iron, 107,713; pig iron, 89,246; all other kinds, 340,043 tons. The larger imports are semi-finished materials from Germany and Belgium.

Other Countries.—In Australia efforts continue to start the manufacture of iron, offers of government contracts being the chief agency on which reliance is placed.

In Brazil little has been done to develop the known deposits of iron ore, though there has been a good deal of mining of manganese ores.

No material changes in the minor countries can be reported.

Lake Superior Iron Ore.

By Dwight E. Woodbridge.

The full statement of iron ore production from the Lake Superior ranges will be found under the head of iron ore on a preceding page. Shipments by the various railroad system that handle ore from mines to upper lake ports are given in the accompanying table.

SHIPMENTS OF IRON ORE.

Railroad.	1905		1906		Railroad.	1905		1906	
	Tons.	Per Ct.	Tons.	Per Ct.		Tons.	Per Ct.	Tons.	Per Ct.
Duluth, Missabe & N.	8,804,443	25.9	11,220,218	29.5	Duluth, S. S. & Atlantic..........	1,243,388	3.6	1,074,045	2.9
Duluth & Iron Range	7,778,768	22.9	8,205,128	21.6	Wisconsin Central.	799,394	2.3	693,852	1.9
Chicago & Northw'n..	6,729,975	19.8	6,706,986	17.7	Wis. & Mi. (car ferry)..........	....	......	62,757	0.2
Great Northern......	5,118,385	15.1	6,133,057	15.8	All rail, not included	198,000	0.6	....	......
Chic., Mil. & St. Paul.	1,310,021	3.9	1,931,244	5.1	Algoma Central...	169,527	0.5	121,555	0.3
L. Sup. & Ishpeming.	1,844,823	5.4	1,889,631	5.0					

These figures emphasize the importance of the Mesabi, which furnished 62.4 per cent. of the year's total, and all but 100,000 tons of the increase over 1905.

There was one overshadowing event of the year, which was not only important in itself, but had its influence on almost everything that has taken place on the iron ranges. That was the so-called "Hill deal" for the sale of the Great Northern and associated ore deposits on the western Mesabi to the United States Steel Corporation. This transaction placed in the hands of the Steel Corporation the great bulk of remaining independent iron-ore tonnage, thus making more difficult than before the creation of additional competition in steel making. So far as the buyer was concerned, this was the compelling cause of the purchase. So far as the Great Northern road is concerned, it is more difficult to determine what the impelling reason was. To be sure, the seller gets a higher royalty than had before been given for any Mesabi ore and secures a high annual product. But there is little question that the road could mine this ore for the market and make a larger profit than 85c. a ton it secures from the Steel Corpor-

ation if it cared to undertake the complex operation of mining, concentrating and marketing these ores. In the manner of annual minimums, others than the Steel Corporation stood ready to make contracts that would have insured a high product and as favorable terms as were finally obtained. The estimates of tonnage in sight from these lands vary greatly, but it is generally understood that it is less than has been frequently stated. That there are possibilities for a tremendous addition to reserves upon complete exploration along the miles of Mesabi formation held under this lease is well recognized, and one of the important matters to be considered by the lessees is the immediate development of that land.

Royalties of 85c. a ton, increasing annually by the amount of the interest charge, and minimums of 750,000 tons the first year, increasing by the same volume annually, have set a new pace for Mesabi range owners. About the same time this deal was closed another was made to an interest in need of good ores, for even a higher price—$1 a ton—and now the owners of undeveloped lands are putting up their figures to something far above any rate ever made in the past.

The Hill deal was one of the factors directly responsible for the construction of the West Mesabi branch of the Duluth, Missabe & Northern road, the other being the Steel Corporation's purchase of the Canisteo lands, made the year before. Upon portions of these lands the ores are quite sandy, and will be concentrated before shipment. Experiments looking to the solution of the problem of concentration are under way. The Oliver Iron Mining Company is building at Coleraine, on the western Mesabi the first 1000-ton unit of a series of washeries, The problem is not regarded as especially difficult, but it is taking considerable study to determine the most economical process. These ores, once washed, are desirable both from their grade chemically, and their structure physically. The character of the western Mesabi enterprise was such that it naturally fell to the lot of a great company to carry it forward; no small concern could undertake operations of the nature and extent required to make it a success. There was the construction of long lines of railway, of suitable machinery for concentration, while the attendant experiments, the opening of enormous deposits for steam-shovel mining on a large scale, the handling of large quantities of water, and the building up, from the beginning, of every item going toward the makeup of a vast mining enterprise. As an instance, one of the stripping operations consequent upon this undertaking covers a total of more than 13,000,000 cu.yd. of material

The marked tendency toward the substitution of open pits for underground mines is still in evidence. Labor costs are advancing, and what is worse, it is hard to get labor commensurate with the amount of work to be done. Timber is rising in price very fast, now in a greater ratio than ever, and the two items have completely changed former conditions

governing the depth to which stripping might be carried. The deepest open pits now on the Mesabi are about 90 ft., but a mining company operating in a large way is to let contracts for stripping an ore body of 108 ft. of overburden, and another company is considering the stripping of 125 ft. of surface. The question of depth of overburden that may economically be taken from over ore is somewhat complicated, far more than by the mere ratio of thickness of stripping to depth of ore. Questions of location of pit, grades into it, spoils banks, regularity of the ore chemically, probable tonnage to be produced annually, are among the more important questions, and lead many to continue underground operations rather than adopt what might seem far more suitable for their situations. Many Mesabi deposits can be economically stripped, so far as depth of overburden is concerned, but there will doubtless be underground mines so long as the district is a producer. Many large and far-reaching plans for stripping operations have been made of late, and the Oliver Iron Mining Company especially is undertaking, or is soon to begin, the stripping of important groups of mines from which an annual tonnage of enormous proportions may be secured. Costs of stripping have been steadily lowered. With the adoption of standard gage equipment and large shovels one development of the stripping of deep mines is sure to come soon—that of hoisting the ore by other means than by the standard railway car attached to a locomotive. In the milling process ore is hoisted in skips, but where ore is sticky when wet—and in open-pit mines it must be wet whenever there are rains—this is no slight difficulty. The time will soon come, doubtless, when instead of the long and costly approach on grade into a mine there will be a stationary engine above the pit and railway cars will be loaded by shovels and hauled out by cable, to be made into trains above.

During 1907 from 35 to 40 large steam-shovels will be constantly employed in earth excavation on the Mesabi range, about half by contracting firms, and half by mining companies. The average work of a shovel is about 35,000 yds. per month. These shovels should move during the season of operation about 10,000,000 yds. of material. One contracting firm is moving at the rate of 3,000,000 yds. per annum.

The concentration of ore into few hands was maintained during 1906, though with the exception of the Hill deal there were few large purchases. Indeed there are few deposits to be sold. It is well within the mark to say that there is not one 40-acre tract on the Mesabi that has not been explored at least once, and hundreds of them many times. Ore-bodies that have escaped this search are few and small. With the lowering of grade that has been going on during the last few years properties are now workable that a short time ago were useless, and most of the so-called new discoveries are of this nature.

Lake iron-ore ships have grown steadily and rapidly. The lake leviathan of 1897 is a pigmy beside that of 1907. In lake shipping, as in ore mining the wildest expectations of one year are weak beside the realizations of the next. Vessels are now carrying upward of 13,000 gross tons of iron ore, or 380,000 bushels of grain. Together with additional size has come more rapid handling of cargoes. The record cargoes of 1906 were those of steamship "J. P. Morgan," from Lake Superior, 11,868 gross tons, and "H. H. Rogers," from Escanaba, 13,333 gross tons. These ships are 600 ft. long. The steamship "Earling," with about 10,000 gross tons, was loaded at the Duluth, Missabe & Northern docks in 135 minutes, of which period only 70 minutes were consumed in actually running ore into the vessel. The fastest time ever made in loading was in 1905, when the ship "Wolvin" took 10,000 tons in 90 minutes. Unloading records were speeded up also. Several times in 1906 ships left the head of the lakes Wednesday morning with cargoes for eastern Lake Erie, and started back up lakes again on Sunday morning. The "W. E. Corey," with 9574 gross tons of ore, was at Conneaut docks but four hours and 29 minutes while the net unloading time was three hours and 32 minutes. Automatic electric unloaders of the Hulett and Brown types handled out of this ship at the rate of 389 and 284 tons per hour, each, respectively. Of this cargo more than 7000 tons was put on board railway cars, as part of the unloading operation, the rest being dumped into dock stocks.

The average iron-ore freight for 1906 was 75c. per gross ton, the distance carried being 880 miles. This was 0.076c. per net ton-mile. Probably freights are carried nowhere else at such a figure. The average rate for the last 10 years has been 88.5c. per ton. For 1907 it has been fixed at the same price as during the season now closed.

Exploration proceeded during 1906. Exploring syndicates and drilling firms were never more busy and were limited not so much by the amount of work offered as by the lack of suitable drill runners. The Mesabi range, as for many years, furnishes the bulk of drill operations. This is carried on, not alone in a search for mineral, but in redrilling of previously explored properties, with the idea of development for mine operations in a more scientific and accurate manner than has been the general rule in the past. No great tonnage of new ore was uncovered on the Mesabi during the year; deposits found were small and scattered. The great bulk of ores found during the year on this range was at the west end, and development proceeded west of the Mississippi river, where some good indications of ore-bodies have been cut into.

On the eastern Mesabi, where original discoveries of the district were made 30 or 40 years ago, developments during the late months of 1906 indicate that ore will be found in workable quantity. These ores are flat,

and lie in thin beds, chiefly of a hard hematite. Exploration is now proceeding with considerable vigor.

On the Cuyuna range, 100 miles west of Duluth, there is little to report. Another year's operations have not been such as to satisfy iron-ore men that merchantable ore exists in workable quantity on the Cuyuna. From 20 to 40 drills were driving holes along the formations, but few claim the range looks better than it did a year before. Operating concerns, which had taken lands there, abandoned explorations, and in some cases gave up their options. The facts as to the Cuyuna seem to be that it contains a large amount of lean iron ore, running from 35 to 45 per cent., largely non-bessemer, and frequently manganiferous.

Work was carried on during the year west of the Mississippi river, in an attempt to connect the Mesabi and Cuyuna districts. This was without result. Some persons worked farther south along the Missisippi river, in Morrison and Crow Wing counties, where float indications are present, but found nothing.

In the Palmer district, Cascade range, south of Marquette, several large interests were drilling all the year, and one of these discovered more than 40,000,000 tons of ore, while others found less amounts. There is, doubtless, an enormous quantity of ore in that district. It is low-grade, silicious ore, in part under the bessemer limit, and quite dry. In addition to this, it is close to surface and can be cheaply mined. Nothwithstanding the tonnages discovered, the use to which these ores can now be put is rather limited, and none of the operators is able to mine to capacity.

Considerable work was done along the Menominee during the year, and a few good deposits were opened, while several others give promise enough to warrant careful exploration. Some work is under way along the western Gogebic, which has been a discarded territory for years.

The Gunflint district of northeastern Minnesota, idle for many years, was under exploration. That ore exists in the district is well known, but it is magnetic; some of it carries titanium and some sulphur, though neither of these elements is present where development is now proceeding. The discovery of a minable property at Gunflint would stimulate exploration over a large area where surface conditions are most alluring for exploration.

On the Canadian side, work lagged. The Atikokan Iron Company is still preparing to open its mine west of Lake Superior, for its new furnace at Port Arthur. The Lake Superior Corporation was unable to carry forward exploration on the Michipicoten range, but mined 121,555 tons from its Helen mine. There were discoveries in the Port Arthur neighborhood the final significance of which cannot yet be determined. The United States Steel Corporation and other parties are exploring in northern Ontario, where the surface indications are excellent. In the Moose Moun-

tain region, little was done further than to test ground and crosscut formations. Indications are that this will be one of the important newer iron-ore fields of the northern country.

A fact, of which the importance can scarcely be overlooked, is the diminishing iron content of ore shipped from Lake Superior. It has been a common complaint for a year or two that Mesabi ores were becoming high in silica. Of course, this is not due to any change in the character of Mesabi ores, but to the shipment of a leaner ore than heretofore. It is still the fact that merchant firms shipping off that range are unable to dispose of an ore much under 56 to 57 per cent. dry, which would be, say, 50 to 51 per cent. natural iron. To be sure, concerns shipping to their own furnaces are able to dispose of an ore considerably leaner, and are doing so, to an ever increasing extent, conserving as far as may be their richer ores.

Sales of Lake Superior iron ores for 1907 were practically all closed before Dec. 1, 1906, and at that time open bessemer ores were sold up for the coming year. Few of these ores are now on the market, and the demand from furnace concerns not sufficiently fortified with ores on Lake Superior was urgent. Even with sales made and most of the mines utterly unable to accept more orders for ore than they have disposed of, not a few furnaces are insufficiently supplied for the year. It is no wonder, therefore, that of new steel-making plants, the proportion of bessemer works is so small.

Progress in the Metallurgy of Iron and Steel in 1906.

By Bradley Stoughton.

Iron Ore Developments.

Great interest was aroused during 1906 by the consummation of the so-called "Hill ore deal" which is probably the largest transaction of the kind ever made, and by which lands estimated by Mr. Hill and others to contain about half a billion tons of Lake ore, largely non-bessemer, are leased to the U. S. Steel Corporation. Second in interest was the growing scarcity of bessemer ores, the increasing basic price of Lake Superior ores—75c. a ton for bessemer ore and 50c. a ton for non-bessemer ore—the lowering of their guaranteed basis of metallic content by 1.3 to 1.7 per cent., and the use with the ores of increasing amounts of limestone—about 20 ℔. more per ton of iron,—bringing the average for the country up to a little over 1150 ℔. There has been also an increasing tendency in Lake Superior and other districts to wash the ores.

Bessemer Ores.—The amount of pig iron used in America for the bessemer process is about one-half of all that made, but this enormous demand has made heavy drafts upon the reserves of ore sufficiently low in phosphorous to supply a suitable pig iron. The result has been, as

is well known, the steady encroachment of the basic open hearth process. Before this advance the bessemer process has, however, continued to maintain the predominant position in point of tonnage produced, but the yearly increases in output have been due to improvements in plant and operating efficiency instead of new installations. Nevertheless, no important step was taken in the abandonment of plant until 1906, when the Carnegie Steel Company decided to dismantle its Duquesne and Homestead bessemer plants and replace them with open hearth furnaces. This is the greatest blow the bessemer process has suffered in America in its history, and is the first instance where a powerful company has actually abandoned a bessemer plant to be immediately replaced by the open hearth. Other important signs of the times are the decision to install only open hearth steel furnaces at the new plant of the U. S. Steel Corporation at Gary, Ind., even for making rails; to build an open hearth plant at the Edgar Thomson works to supply the new rail mill to be erected there; re-entry into the railroad rail field, with open hearth rails, of the Bethlehem Steel Company, whose bessemer plant has long been idle, etc. It is freely predicted that, unless very unexpected and extensive bessemer ore deposits are discovered bessemer plants will continue to decrease and that the amount of open hearth steel made in the United States in 1908 will exceed the amount of bessemer steel made. There are about 62 standard converters now operating in the United States.

Ore Reserves.—During 1905 a spirit of pessimism as to the world's available ore seemed to exist and in my previous review, I ventured to express opposite views. That my opinion is the more generally accepted one in the United States is shown by the many estimates that have been made during the past year of the long life which the American iron industry may reasonably expect in spite of the fact that it now makes more than one-third of the world's iron, although its ore reserves are estimated at not more than one-ninth of the world's known deposits. Important discoveries were made on all the Lake Superior ranges in 1906, and the future there appears very bright. This is especially true of the Mesabi range where deep drilling, even in well-prospected regions has located new deposits of importance; in the Gogebic range where mining is now at some depth, and especially in the Swanzey district, off the Marquette range. It is evident that the Alabama reserves have barely been touched and that the red ores of that district are likely to be of importance. There have been several important discoveries in Mexico which will be developed by United States iron masters, especially in Guerrero, where deposits that may be among the largest in the world have been located, running in some instances 65 to 69 per cent. iron and usually 0.007 to 0.006 per cent. phosphorus. They are said to be capable

of being worked very cheaply by quarrying. All the siderurgical nations, with the exception of England, have been very active in developing ore or obtaining control of ore properties. In the United States the Geological Survey, whose work in the Lake Superior region has resulted in developing the properties very successfully, has now become active in other localities where there seems a good prospect for large deposits to be developed. A beginning is to be made by exploration in Alabama, Tennessee, New York, New Jersey, Virginia, Wyoming, Colorado, Utah, New Mexico and Washington.

Briquetting, Nodulizing, etc.—A good deal of interest has been manifested by large steel companies in the briquetting of ores, etc., and some of the principal processes have been described in the technical literature[1]. A new process for desulphurizing and nodulizing has been exploited by the National Metallurgic Company of Newark, N. J.[2] In this process burnt pyrites from sulphuric acid manufacture, or fine magnetites, flue dust, hematites (Mesabi), etc., are fed into an inclined rotary kiln, being mixed meanwhile with 1 per cent. or so of a binder consisting of tar or some similar substance. The binder coheres the fine ore into nodules which pass through the kiln meeting a gradually increasing temperature. At about 1200 deg. C. the tar unites with the sulphur in the ore and both are volatilized. At the end of the kiln (2000 deg. C.) the ores frit together with the heat. These nodules have been used at six different blast furnaces with success and over 24,000 tons of fine ores and by-products have been converted. The company controls over 80 per cent. of the pyrite cinder output of the country for the next five years.

Transportation.—During 1906 in the Lake ore trade there was a tendency to revert to smaller sizes of ships than the recent 600-ft. vessels carrying upwards of 10,000 tons of ore each. The Hulett electric unloader is gaining in popularity.[3]

The Blast Furnace.

The most important innovations in blast furnace practice in America in 1906 were greatly increased use of the gas in engines, an extension of the air drying process, improvements in methods of distribution by automatic charging devices, increased employment of slag for useful purposes and great increase in capacity, both by the erection of new furnaces and by increasing the output per furnace. There were 62 new blast furnaces authorized during the year with a collective annual capacity of more than 7,000,000 tons. About one-half of this number is already completed or under construction. The average output per furnace was also increased to 77,000 tons. The significance of these figures may

[1] *Stahl und Eisen*, XXVI, pp. 2–8 and 76–82, H. Wedding; also *Eng. and Min. Journ.*, LXXX, pp. 969, 1022–1023.
[2] *Journ.* of the Iron and Steel Institute, 1906, No. III, pp. 358–376.
[3] For a description of this see *Iron Trade Review*, Nov. 15, 1906, XXXIX, p. 32.

be seen by comparing them with the corresponding figures for other countries, as follows: Germany, 43,000 tons[1] per furnace per year; Belgium, 36,000 tons[1] per furnace per year; England, 27,600 tons[1] per furnace per year; France, 25,500 tons[1] per furnace per year; Sweden, 5,700 tons[2] per furnace per year. The United States figure includes charcoal furnaces, which pull the general average down, as indicated by the Swedish average.

Charging Devices.—In the old method of hand charging it was possible to empty the barrows at any point in the circumference of the hopper. With automatic charging this regulation is not possible and there has been difficulty with blast furnace working on that account. The result has been the invention of devices whereby the upper bell in the automatic charging rig shall deliver the material to any point of the lower bell, or can be regulated to deliver at intervals around, according to the calculations of the furnace manager. The Baker and Baker-Neuman rotary charger has a deflecting plate on the bell of the upper hopper which distributes the charge to one side. After each skip has been emptied into the upper hopper, the bell is automatically turned through any predetermined angle, an indicator at the foot of the hoist showing its position. Eight blast furnaces are equipped with this device and five are equipping, including four at the Gary plant of the United States Steel Corporation, where A. B. Neuman is chief engineer. The experience at furnaces equipped with this device shows a saving of fuel of 5.4 per cent. an increase of iron production of 21.6 per cent., greater regularity, of working, less flue dust and better maintenance of linings.[3] The McKee revolving furnace top[4] is installed at 11 blast furnaces and is in successful operation. No description is given and patents are pending. A rotary device, said to be an improvement on that of Baker, and invented by Engineer Geuze at Trith-Saint-Leger is described.[5]

Stock Line Recorder.—J. E. Johnson, Jr., describes a form of stock line recorder[6] which indicates and automatically records the movement of the stock with accuracy, not only telling the regularity of the charging but also whether the stock is settling regularly or slipping, and how much, etc. The device consists of a suspended rod resting upon the stock and descending with it except when the bell is opened; there is an attachment for recording the motion of this rod on a scale at the bottom of the furnace and a mechanism for automatically lifting the rod out of the way of the incoming stock when the bell is open.

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[1] Estimated.

[2] For the year 1905.

[3] This top is described in the *Journ.* of the Iron and Steel Institute, 1906, No. IV, pp. 600–602; and *Iron Age*, July 26, 1906, LXXVIII[1], pp. 214–215.

[4] *Iron Trade Review*, Dec. 27, 1906, XXXIX, p. 32.

[5] *Revue de Metallurgie*, 1906, III, p. 190. See also a general description of charging apparatus, *Stahl und Eisen*, 1906, XXVI, pp. 1303–1311.

[6] *Trans.* A. I. M. E., 1906, XXXVI, pp. 79–89.

Kennedy Closed Top Furnace.—Julian Kennedy has built 24 furnaces without any explosion doors or other relief doors on the top of dust catchers or gas mains.[1] The top of the furnace is securely fastened down and every effort made to resist the pressure when a so-called explosion or slip occurs. The furnaces have been operated on all character of ores and have undergone severe slips without any dire result. No men have been burned or injured by material thrown from the top of the furnace and no indications given that the structural strains have been increased.

Stoves.—A number of advantages are obtained from the Hartman-Kennedy stoves[2] of which there are now 83 in operation in different parts of the country. Combustion takes place in an auxiliary combustion chamber beside the stove and the gases thence pass into the first regenerator, having thick flue walls, and then into the second regenerator, having thin flue walls. The combustion chamber is always kept hot, because the cold air never passes through it, and this is said to give more perfect combustion. The gas dust is also said to be released in the combustion chamber and falls to the bottom where it is not exposed to so high a temperature and so does not fuse. It is blown out when changing the stove from air to gas. Thick walls in the regenerator flues absorb more heat but do not attain such a high temperature; therefore they do not glaze. When the burning gases have been cooled off somewhat, however, i.e., after they have passed through the first regenerator, they are brought in contact with thinner walls without this danger. The gases passing out of the stove are said to be at a temperature of 450 deg. C. and there is 5 sq. ft. of flue surface per cubic foot of air.

Use of Gas Engines.—The action of the United States Steel Corporation in ordering gas engines to the extent of 102,000 h.p. to be operated by blast furnace gas places the United States, which was very much behind in this field of progress (there being previously only 40,000 h.p. at the Lackawanna Steel Company), second only to Germany, where a total of about 385,000 h.p. is obtained by means of gas engines, about two-thirds working on blast-furnace gas and one-third on coke-oven gas. In Belgium 103,550 h.p. are obtained from gas engines operated by blast-furnace gas, and in England 4601 h.p. from blast-furnace gas and 95,000 h.p. from other gas. If all the available blast-furnace gas in the United States were used in gas engines instead of under boilers, the amount of power developed would be about 1,500,000 h.p. which is equivalent to about one-third the total power of Niagara Falls if fully developed, or about 20,000,000 tons of coal burned under boilers. About one-half of this 1,500,000 h.p. would be available for use at the blast furnace for driving the blowing engines and other power purposes, while

[1] *Proc.* Eng. Soc. of Western Penn., Presidential Address, Feb. 1907, XXII, pp. 1–15.
[2] *Iron Trade Review*, Sept. 20, 1906, p. 27.

the other half would be available for operating steelworks, rolling mills, etc.

As is well known, the difficulty from carbon monoxide poisoning has been met more frequently in plants using the blast-furnace gas in gas engines than elsewhere, and the following simple carbon monoxide indicator may be very useful for testing the atmosphere at intervals to indicate leaks:[1] "The air to be examined is aspired through a tube packed with glass wool, which frees it of dust. This is accomplished by allowing water to flow from an aspirator bottle. The filtered air then traverses a U-tube filled with anhydrous iodic acid which is maintained between 70 and 80 deg. C. by a small alcohol lamp. By such passage the monoxide is oxidized to carbon dioxide and free iodine is liberated. This iodine, carried forward by the current of air, is arrested by solution in a little chloroform contained in another tube. The rose color produced in the chloroform, by the iodine becoming dissolved therein, is compared with a set of standard colored tubes previously prepared and kept on hand. To prevent evaporation of the chloroform it is covered with a layer of distilled water."

Dried Air.—Mr. Gayley's dried air process is being adopted at the Carrie, Duquesne and Chicago furnaces of the United States Steel Corporation; by the Warwick Iron and Steel Company, of Philadelphia, and at the Dowlais Iron Works, Cardiff, Wales.

Fixation of Atmospheric Nitrogen.—As pointed out in my last review, and as also mentioned by other writers recently, blowing into the blast furnace air enriched in oxygen, a process patented by J. E. Johnson, Jr., would affect the same results in the regularity of working, fuel economy, etc., as drying the air. The fixation of atmospheric nitrogen which gives promise of furnishing the enriched air at a low enough price for this purpose made important industrial advances during 1906[2]. There are two processes of fixing the nitrogen, one of which causes a combination of the oxygen and nitrogen of the air and would therefore not produce the by-product wanted by the blast furnace plants. The other process, however, invented by Dr. Frank, of Charlottenburg, Germany, makes calcium cyanamide—a fertilizer—in an electric furnace from calcium carbide and atmospheric nitrogen.

Slag Handling.—There is an increasing tendency to granulate the slag at the blast furnace and then load it into gondola railroad cars by mechanical devices. This is advantageous where room is lacking or a long haul is necessary for dumping the liquid cinder, and the granulated slag makes much better filling and road bed. The cost of handling up to the gondola is said to be less than a cent per ton.

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[1] *Engineering News*, 1907, LVII, p. 132.
[2] *Electrochem. and Met. Ind.*, IV, pp. 23, 295–296, 327–328.

Cement From Slag.—There is a constantly increasing amount of portland cement made from blast furnace slag, and the industry is to be greatly increased because of the decision of the United States Steel Corporation to treble its output and make about 6,000,000 bbl. of cement per year. In the process operated by this company the slag is granulated and mixed with limestone in proper proportions. The mixture is then dried, pulverized and burned to a clinker in rotary kilns, after which it is again pulverized and put up into bags. The contemplated output by the United States Steel Corporation will exceed 10 per cent. of the total cement production of the country, which in 1906 amounted to about 50,000,000 bbl. valued at about $54,000,000 and divided as follows: portland cement 46,610,822 bbl.; natural cement 3,935,151 bbl.; puzzolan cement 481,224 bbl.; total 51,027,321 bbl.

Another process is in use by the Struthers Furnace Company in Ohio.[1] The slag must contain 31 to 34 per cent. silica, 12 to 15 per cent. alumina, 46 to 49 per cent. lime and not more than 1.3 per cent. sulphur. It is granulated at the furnace and transferred to the cement factory where it is screened to pass a $\frac{3}{4}$-in. sieve, dried at a red heat and pulverized. It is then ground with lime to form cement which analyzes 30.40 per cent. silica, 12.86 per cent. alumina and iron oxide, 50.30 per cent. lime and 0.80 per cent. sulphur.

Coke Supply.—Already there are indications of the ultimate exhaustion of the Connellsville seam. In some localities ovens have been abandoned, and the price of Connellsville coke has risen steadily. At the Gary plant of the United States Steel Corporation by-product coke ovens are to be employed. Neither the Otto-Hoffman nor the Semet-Solvay will be used, but ovens designed by engineers of this company. A great deal of interest has been aroused also by the final decision of the United States Steel Corporation to build a plant near Duluth. This is the first instance where large producers have decided to carry the coke to the Lake Superior region instead of carrying the ore to the coke.

Oxygen to Remove Obstructions.—In the *Journal* of the Iron and Steel Institute, No. I, 1906, pp. 125-132, is an account of the use of oxygen for melting iron. To open the frozen tap hole of a blast furnace, for example, or a tuyere, the metal will first be heated by the oxyhydrogen flame and then the oxygen gradually increased until a pressure of 30 atmospheres or more is employed, the hydrogen being correspondingly diminished. The iron is burned and a block of 16 in. thick will be cut through in one or two minutes, the pressure of the oxygen keeping the hole clean. For tap holes from 8 to 40 cu. ft. of oxygen will be used at a total expense of from 75c. to $2.50. There are about 80 blast furnaces in England, Germany, Austria, France and Belgium using the process, but it

[1] *Iron Trade Review*, June 28, 1906, XXXIX, p. 17.

cannot be used in the United States unless the law be changed so that gases can be shipped confined under pressures of more than 250 lb. Oxygen is used in the same way for cutting steel, such as removing risers from castings, cutting up steel work after the wrecking of a building, opening the pipes in the top of ingots so that hot metal may be poured in, etc., etc.

Records.—Four of the 11 blast furnaces of the Edgar Thomson works made collectively 82,301 tons of pig iron in October, 1906. This is an average of nearly 700 tons per furnace per day. One of the Lackawanna blast furnaces was relined and put in operation in 21 days, while another Lackawanna furnace was erected complete in five months, ground being broken for foundations on Sept. 15, 1906, and the first cast of iron made on Feb. 14, 1907.

The Manufacture of Steel.

The most important developments in this field in America during 1906 were the abandonment of the bessemer process in favor of the basic open hearth by important interests, which as previously pointed out, is rather a question of ores than of steel manufacture, the increase in the so-called "standard size" of open hearth furnaces from 50 tons to 60 or 70 tons capacity, the increase in the price of natural gas in the Pittsburg district which has finally resulted in the installation of gas producers by the Carnegie Steel Company, the continuation of the increase in size of steel mixers up to 700 and 750 tons capacity, the increase of the use of the Talbot furnace, of the Duplex process and of the proportion of pig to scrap used in the basic open hearth.

Basic vs. Acid Steel.—The tonnage of basic open hearth rails made in 1906 was probably very close to 250,000 tons, an increase of over 100,000 tons since 1904. This fact is not as significant, however, as is the installation of open hearth furnaces for the production of rails at the Edgar Thomson steel works, the largest rail mill in the world, at Youngstown and at Bethlehem, and that the new plant of the United States Steel Corporation at Gary is to make rails altogether by the basic open hearth process. The Tennessee Coal, Iron and Railroad Company, the pioneer in the manufacture of open hearth rails on a large scale, raised its price of rails \$1 a ton above the current price for bessemer rails. A committee of the American Society of Civil Engineers declared in favor of open hearth rails over bessemer rails for quality.

The basic open hearth process has also again invaded the tubing field, which had been almost entirely filled with bessemer steel, the National Tube Company having authorized the erection of open hearth furnaces at McKeesport and at Lorain. These furnaces will be used chiefly for the production of tubes higher in carbon than those made in the bessemer process. In this sense therefore it has not replaced bessemer

metal from its particular field, because the ability to produce easily low carbon steel of good quality is the chief reason why the bessemer process has maintained its position so long. To offset this gain of the basic open hearth, the National Tube Company added another converter at McKeesport and new bessemer plant, containing two 10-ton converters was built at the plant of the Youngstown Sheet and Tube Company. This is the only bessemer plant built in the United States for some years.

The open hearth has also made inroads into the tin plate and wire field, where the bessemer has long been predominant on account of its ability to make low carbon metal easily, and, in the case of tin plate, because the basic open hearth metal was too low in phosphorus to be rolled double without danger of the plates sticking together. During 1906 open hearth furnaces were built especially for the purpose of making steel for wire, for example, a plant authorized by the Pittsburgh Steel Company at Monessen, Penn. The difficulty of plates sticking together has been avoided in some cases with open hearth metal by increasing the phosphorus with ferro-phosphorus during recarburizing. In addition to the advantage of being able to use pig iron made from high phosphorus ores, the basic open hearth process has encroached upon its rival because the steel is preferred for its quality and the difference in price is now so small that it does not cause a barrier. Furthermore the basic furnace has the advantage of being able to modify the analysis as regards phosphorus and, to a limited extent, sulphur according to varying specifications.

That the gains made by the basic open hearth process are to be attributed mainly to the basic lining rather than to the process itself is emphasized by the fact that the basic open hearth is supplanting the acid open hearth even more rapidly than it is the bessemer process. In fact, acid open hearth steel is actually on the decrease, even in the field of steel castings where it has admittedly important advantages, such as producing a better surface, on account of sand sticking to the basic metal, and a much less liability to blow holes, on account of the lower silicon in the basic steel. Nevertheless, for foundries doing a regular line of work with not too frequent changes in types of patterns each of which is a study in itself, the basic process is reasonably satisfactory; and, on account of its great cheapness, is largely used, so that in a few years time, we may expect to see more basic than acid castings made. Furthermore the employment of the acid and basic processes side by side in the same foundry is a decided advantage, since the scrap from the basic castings is very suitable for use in the acid furnace.

In structural steel also there is a tendency to use more basic metal and one authority states that the controversy among engineers that basic steel is not so good is practically dead. Nevertheless I have heard

of no bridges of large size and first importance in which acid steel is not required by the specifications, and I believe that the statement quoted is, let us say, premature.

In Germany also basic steel is increasing much faster than acid, and especially in castings.

Increased Length of Open Hearth Furnace.—At the works of the Lackawanna Steel Company five new basic open hearth furnaces were built with a nominal capacity of 60 tons[1] and with a length between ports of 43 ft. and a port construction such that after they have burned back the maximum distance, they will still retain sufficient length to properly direct the currents of gas and air. The result of the greater furnace length is better combustion within the laboratory of the furnace and a consequent important reduction in fuel used and in the life of the furnace. Since combustion takes place before reaching the ports, the latter are not burned back so rapidly and since they direct the streams of gas and air effectively, the roof and walls do not suffer as much from the heat. These innovations are due to T. S. Blair, Jr., manager of the open hearth plant, and after their demonstration, several plants increased the length of furnaces building: for example, the Pennsylvania Steel Company has installed five 75-ton furnaces, the Gary plant and the Duquesne plant of the United States Steel Corporation and the Monessen plant of the Pittsburgh Steel Company have authorized collectively over forty 60-ton furnaces, etc.

Fuels.—The price of natural gas in the Pittsburgh district has been increased at intervals during the last few years, there being a final advance of 10 to 12 per cent. in December, 1906. This resulted in an order for 64 Hughes mechanically-stoked gas producers, equivalent to 160 hand-poked producers, for the Duquesne plant.[2] The same type of producer will also be used at the new furnaces of the Pennsylvania Steel Company. It is interesting to note that there are now about 100 Morgan producers in England and Europe, this being the only American producer that has been able to make an entry there.

Petroleum.—In the six years previous to 1905, the output of petroleum in the United States more than doubled, the increase being in the heavy fuel oils, whose price greatly decreased at the same time. The result has been a constant increase in the use of this fuel in the open hearth furnaces removed from the natural gas and coal districts, because of the economy with which oil may be transported and also because of the operative advantages of its use. It requires no regeneration, but is applied directly in blowpipes, atomized with steam or compressed air, and on account of the ready control of the combustion and the flame,

[1] Described in *Iron Age*, Jan. 3, 1907, LXXIX[1], pp. 10–12.
[2] The Hughes producer is described in *Iron Trade Review*, Nov. 22, 1906, XXXIX, p. 32; and *Iron Age*, 1906, LXXVIII[2], p. 1372.

it does little damage to the roof of the furnace, which results in an important increase in the life of that part. It has one disadvantage, however, in that the flame is liable to spread laterally and thus cut the furnace walls, and it gives a longer flame than either natural gas or producer gas. It requires no labor except general supervision of the pumps. The American fuel oils usually average from 7.8 to 8.3 ℔ per gal. with a calorific power of 1400 to 1700 B.t.u. per ℔. H. H. Campbell states[1] that 50 gal. of oil are equivalent to 1000 ℔ of soft coal in gas producer or regenerative furnaces. This value seems somewhat high, however, and estimates that I have made would indicate that from 35 to 60 gal. of oil will be required per ton of steel treated in the open hearth furnace, depending upon practice. This figure is not very different from the published results of the makers of an oil burner for open hearth furnaces with which eight furnaces have been equipped, viz., that 38 to 42 gal. of oil are required per ton of steel.[2] It is true that the use of oil in locomotives[3] would indicate much less efficiency than the figures I have mentioned, but there is a very important difference, in that the introduction of oil into a very hot furnace gives a very much greater efficiency than when it is introduced into a colder chamber.

Talbot Furnaces.—There were 20 Talbot furnaces installed in January, 1906, and ten more were installed during the year: six of 200 tons each at the Jones & Laughlin plant, two of 200 tons at the plant of the New York State Steel Company in Buffalo, one of 175 tons at Cargo Fleetworks, England, and one of 160 tons at Senelle, France, which is the first to be installed on the Continent.

Duplex Process.—The Duplex process is to be extended at Alabama and 20-ton bessemer converters, the largest in the country, are to be used in connection with basic open hearth furnaces of 6000 tons monthly capacity each. The process is also being worked at a very important steel works in the Pittsburg district whose name I may not publish.

Open Hearth Scrap.—The amount of scrap used in a basic open hearth furnace has been decreasing and the amount of scrap bought by large steel works operated in connection with rolling mills is decreasing even more, which indicates that there is more produced during the rolling process, probably as a result of more cropping on account of the researches on segregation demonstrating the extent of this in the upper part of the ingot. The amount of basic pig iron made was 40 per cent. of the amount of basic steel made in 1901 and 48.6 per cent. in 1905. The intention is to use about 60 per cent. pig iron and 40 per cent. scrap at the open hearth plant at Gary.

Premelting Additions.—The Carnegie Steel Company has acquired

[1] "The Manufacture and Properties of Iron and Steel," 4th Edition, 1907, p. 168.
[2] *Iron Age*, 1907, LXXIX, pp. 192–193.
[3] See *Engineering News*, 1906, LVI, p. 133, and 1907, LVII, pp. 33–35.

the right to patents for a process which will presumably be used in connection with its Monell process and whereby ore and lime are melted and then added to an open hearth bath. This process avoids the wear of the lining that occurs when these materials are melted in the furnace as in the Monell process and also saves time of the furnace. It is also stated that less oxide of iron is carried off mechanically than occurs when the Monell reaction takes place and the slag runs from the furnace.

Mechanical Treatment.

The most important developments in the mechanical treatment of steel have been the applications of electric motors for the driving of roll trains, the increasing application of steam turbines and rope drive, a continuous wire drawing system, a continuous bloom heating furnace and increased application and new processes for the fluid compression of ingots.

Electric Motors in Rolling Mills.—All American rolling mills of most modern equipment now use electric power for all purposes except the driving of the roll trains, and in this latter respect very important installations were made during 1906. In Europe there are many motor-driven trains of smaller size, including mills in England, several in Sweden and in Germany, notably at the Peine, Remschield and Phoenix works. Most of the motors for this purpose are of but a few hundred horse power, although at the last named works, a reversing motor of 1200 h.p. was recently installed. At the Hildegarde work in Silesia, the first large reversing mill in the world was started in July, 1906. It is a cogging mill for reducing two-ton ingots, with a maximum speed of 140 r. p. m. of the rolls and operated by three motors on one shaft with a total maximum capacity of 10,350 brake-horse-power and balanced, by the Ilgner system whereby the extreme variations, caused by rolling and reversing, are compensated. In America there are a few small roll mills operated by electricity and in the latter part of 1905, an electric roll train of 1500 h. p. was installed at the Pittsburg Reduction Company (rolling aluminum). At the beginning of 1906 two 1500 h.p. motors were installed to drive one of the rail trains at the Edgar Thomson works and shortly afterward a new rail mill for small sizes at the South Works of the Illinois Steel Company was installed, with electric drive. At the same works a new two-high, reversing universal mill capable of rolling plates from 6 to 30 in. wide and 80 ft. long to the extent of 7500 tons per month and to be operated completely by electricity was authorized. This is the first electric reversing mill in the country. The rolls are operated by two motors on the same shaft which, when running at full speed (150 r.p.m.), will develop approximately 10,000 h.p. This mill is capable of reversing from full speed forward to full speed backward in the space of three seconds. Another innovation in this mill is the arrangement

whereby the vertical rolls can be brought much nearer the horizontal rolls than was hitherto possible, and they can be removed without disturbing their drive shaft, or else they can be lifted out vertically. Another very powerful electric installation will be the rail mill at the Gary plant of the United States Steel Corporation, which will be the first electric rail mill of standard section in the world. There will be two stands of bloom rolls, the first being operated by two motors of about 2250 h.p. each and the second by one motor of about 6000 h.p.; two roughing mills each operated by one motor of 4500 h.p. and a finishing mill operated by a motor of 2250 h.p. All auxiliary machinery, tables, etc., will also be operated by electricity.

The advantages of electricity over steam are low operative cost, great security of operation and a more flexible connection between the prime mover and the load. The advantage of steam is that it adjusts itself better to the extreme variations in load that always occur in rolling mills and especially in reversing mills.

Large Steam Engines.—Large steam engines for three high roll trains with fly wheels of 180,000, 200,000 and 224,000 ℔. each have been erected during 1906. Two such engines with an indicated horse power of about 9000 each have been erected, one at the Edgar Thomson and the other at the Homestead works of the Carnegie Steel Company.

Turbines.—Small steam turbines of the Rateau system using waste steam have been installed by the Steel Company of Scotland, Hallside, and in the International Harvester Company's works, South Chicago. A steam turbine of 24,000 h.p. is being built by Brown, Boveri & Co. for the Krupp rolling mill at Rheinhausen, Germany.

Rope Drive.—Rope drive for connecting several stands of rolls was installed at a number of works during 1906.

Rolling of Special Shapes.—The Grey process for rolling I-beams up to 30 in. depth, with flanges up to 15 in. wide, is being installed in the works of the Bethlehem Steel Company by Henry Grey & Son. I thas been in operation at the Differdingen works in Germany. This universal mill[1] forms the web of the beam with a pair of horizontal rolls, and forms the flanges with lateral, auxiliary, vertical rolls. This gives the possible variation in sizes and produces a more uniform amount of reduction in all directions. Since the web can be made of any desired thickness irrespective of the other dimensions, beams can be rolled with the same strength as standard sections and about 10 per cent. less weight, or of the same weight and greater strength.

In the *Journal* of the Iron and Steel Institute, No. 3, 1906, pp. 307 to 332, are described the York universal bloom mill and transverse mill. The first produces I-beams of varying sections, without using grooved

[1] *Iron Age*, 1906, LXXVIII[2], pp. 1142–1146.

rolls, but by means of auxiliary, lateral rolls. The second produces a number of useful shapes, hitherto impossible by rolling processes.

Rolling Round Bars.—A new process for rolling round bars, invented by W. Tafel and in use for two years at the Nuremberg iron works, Germany, causes an oval rod, supported by oval guides, to pass through two sets of rolls placed tandem with a close fitting guide between them. The rod cannot twist and comes out true to shape and with a smoother surface, because the pressure in the last pass is small. Furthermore the two ends of a rod rolled in this way are the same size. The process is of great advantage for rods for wire drawing and other purposes and may be used for square rods, if desired, or other shapes.[1]

Continuous Wire Drawing.—The Horton continuous wire drawing process[2] is in use by the Shenandoah Steel Wire Company and is being installed by the Southern Steel Company, both in America. This process draws wire through from three to nine dies at one operation, there being a drum between each two dies, driven by power, and around which the wire passes in order that it may be pulled through the previous die. The wire may be drawn wet or dry, and speeds up to 500 linear feet per minute are obtained.

Pressed Steel Shapes.—The number of forms produced by hot or cold pressing, with or without dies, and the tonnage produced in this way, and also by drop forging, is increasing very fast, indicating the growing preference of engineers for mechanically treated over cast material. Some of the new pressed shapes of importance are pressed steel car wheels and tires,[3] bath tubs[4] and radiators.[5]

New Chain Making Process.—The continuous making of chain with one furnace and one machine has never before been accomplished, but machines for this purpose have been invented by Emile Lelong, of Belgium.[6] The machines are made in three sizes to produce a chain up to 4. in and over and it is stated that the cost of manufacture is less than one-half of that by the ordinary method.

Fluid Compression of Ingots.—The interest in the fluid compressing of steel ingots in the molds, both for producing pipeless ingots and for avoiding segregation which was referred to in my last review, was if anything, increased in 1906. For many years the Whitworth method of compression was alone in the field and it continues to occupy an important position because of the excellence of the ingots made, and it is stated that an ingot weighing 120 tons was compressed in 1906. The Harmet process also continues to be of importance. John Illing-

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[1] *Iron Age*, 1906, LXXVIII², pp. 1664–1666
[2] *Iron Age*, 1907, LXXIX¹, pp. 18–25.
[3] *Iron Age*, 1906, LXXVII¹, p. 428; and *Journ.* of the Iron and Steel Institute, 1906, No. I, pp. 179–207.
[4] *Iron Trade Review*, Oct. 11, 1906, XXXIX, p. 17; and *Iron Age*, 1906, LXXVIII², p. 923.
[5] *Iron Age*, 1906, LXXVIII², p. 1507.
[6] *Iron Age*, 1906, LXXVII¹, p. 856.

worth, an American inventor, joined forces with Robinson & Rodger of Sheffield, England, there being already great similarity between their methods. This combined process uses divided ingot molds. The process itself forms the casting pit and the steel is teemed into all the molds simultaneously by means of multiple runners. Pressure is then applied from the end, but only sufficient to take up the contraction of the ingots, and amounts finally to about two tons per square inch of surface.[1] The process is in use by William Jessop & Sons, Ltd., England.

Another new process has been invented by N. Lilienberg, of Philadelphia, whereby the ingots are cast on cars in molds as in the ordinary method. They are then stripped and drawn between the jaws of the compression apparatus. These jaws are forced together by driving wedges alongside of one of them. Just enough pressure is applied to take up the contraction and when the ingots are entirely solid, they are released and drawn away on the cars.[2]

Several valuable researches upon segregation in steel ingots were published during 1906, which are summed up in an article by Prof. Henry M. Howe with other very valuable data in the *Transactions* of the American Institute of Mining Engineers, 1907.

Continuous Bloom Heating Furnace.—The Guettler continuous bloom heating furnace has several advantageous features. In many respects it is similar to the Morgan continuous billet furnace, but the hearth is divided about the middle of its length, the first half being several inches above the level of the lower half. The furnace is 32 ft. long and will heat blooms 7⅛ in. sq., weighing 660 lb. each. They rest on water cooled pipes and are pushed along towards the fire by means of a hydraulic pusher which has an adjustable stroke up to 16 in. When the first bloom comes to the edge of the upper level, it rolls down turning over 90 deg. and thus bringing the two opposite sides, exposed to the greatest heat. There is then a second pusher operating on the lower level which pushes the line of blooms along that hearth. As the line of blooms is only half as long as the furnace much less power is required to slide them over the pipes. The furnace is to be introduced at the Peine works, Germany.[3]

Founding.

The most important tendencies in foundry practice during 1906 were the increasing attention given to science and education by foundrymen and the greater willingness to exchange information, increasing use of ferro-silicon added to the ladle of iron, of mechanical labor-saving devices, of fluorspar as a cupola flux and of steel castings in place of iron castings.

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[1] *Journ.* of the Iron and Steel Institute, No. I, 1906, pp. 28–47.
[2] *Trans.* A. I. M. E., 1906.
[3] *Iron Age*, 1906, LXXVIII[1], p. 735.

Ferro-Silicon in the Ladle.—At the June meeting of the American Foundrymen's Association[1] was described the practice of adding powdered ferro-silicon in the proportion of about 1 lb to 200 lb of iron in the ladle with the object of increasing the silicon in the metal and thereby softening it and reducing its melting point, both of which objects are accomplished better by adding the alloy in the ladle than by introducing it with the cupola charge. From information published, as well as certain facts that have come to my attention privately, it appears that this practice is not new, even though a patent was obtained upon it. Nevertheless great benefit will certainly result to the foundry industry through its attention having been attracted to the advantages of this process.

Fluorspar Flux.—The use of fluorspar as a cupola flux is not new, but a good deal of attention has been devoted to it by writers in the technical journals, some for and some against the practice. The custom is to add only a few pounds of fluorspar with the limestone in order to make the slag melt at a lower temperature and be more fluid. That it accomplishes these purposes can hardly be doubted, and therefore that it causes more rapid melting and presumably carries off a little sulphur, its action in this respect in the open hearth furnace, blast furnace, etc., being well recognized. It is also stated that the use of fluorspar reduces the amount of slag made on account of a small volatilization of silicon.

Machines in the Foundry.—American foundries are already ahead of most of the rest of the world in their use of mechanical apparatus to perform the suitable operations and almost all large modernly equipped plants use extensively mechanical sand mixers and temperers, sand conveyors, tumbling barrels, pneumatic hammers and chippers, sand blast for cleaning, and molding machines for at least a part of the work, if they are proved to be more economical. One step seems to have been neglected and that is the installation of a number of motors instead of one steam engine which must be combined with cumbersome transmission of power or else a lack of the most economical and convenient arrangement of plant. Besides their greater cost for operating, steam engines are out of place in foundries, on account of the great inconvenience when overhead shafting is employed.

Core Box Machines.—The Thomes core box machine, designed to produce core boxes of unusual shapes at very low cost, would seem to have important advantages.[2]

Molding Machines.—There was a very great extension in the use of molding machines in 1906, partly due to their intrinsic advantages and

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[1] See also *Iron Trade Review*, June 7, 1906, XXXIX, p. 34.
Iron Trade Review, Sept. 6, 1906, XXXIX, p. 37.

partly to labor troubles which have given both the opportunity and the incentive to install the mechanical molders. Two new machines have aroused a good deal of interest: The first is the Bonvillain molding machine which seems to mold intricate shapes with great effectiveness and is being rapidly introduced into France, England, Russia, Mexico, Sweden and Spain.[1] The second is a gravity molding machine in which sand is conveyed to a hight and dropped progressively upon the pattern around which it is packed by gravity.[2]

Multiple Molding.—There was also, during 1906, a great deal of interest in the Rathbone multiple molding machine which produces several flasks of gated patterns at one pouring, there being several layers of gates and six or more cheeks between the cope and drag. The result is indicated by Fig. 1. The Rathbone multiple molding machine forms

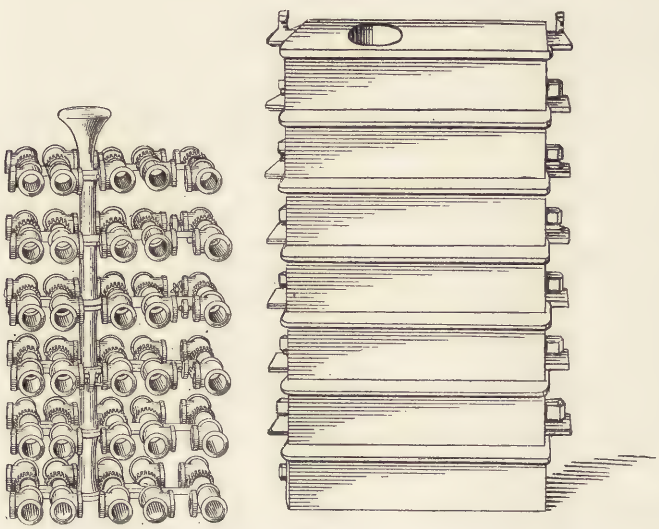

FIG. 1.

the impression for the cope part of one gate at the same stroke as the drag part of the gate next above it. It is stated that economy in molding and in production of scrap is obtained by this method and that the pressure upon the sand in the bottom layer is never excessive, because the gates of each pattern are made so small that they will chill soon after the patterns are poured, and before the metal in the main runner has risen to any great hight and so established a head.[3]

Air Furnace Practice.—The secrets of air furnace practice have been guarded very jealously by foundrymen, but in this respect a spirit of

[1] *Iron Trade Review*, Nov. 29, 1906, XXXIX, p. 25
[2] *Iron Trade Review*, Jan. 17, 1907, XL, pp. 116–118.
[3] *Foundry*, July, 1906, XXVIII, pp. 359–362.

great liberality was displayed during 1906 and a number of very interesting facts were published. At the June meeting of the American Foundrymen's Association,[1] Ralph H. West published data furnished by a large number of American foundries. From these facts it appears that the fuel used is 2¾ to four times the weight of the iron melted and that the melting time varies from about five hours, in the case of nine-ton heats, to nine hours, in the case of 35-ton heats. The size of furnaces varies from about 10 to 45 tons and the practice of mechanical charging is approved, one of the most economical methods being to have the charging door situated on the end of the furnace, away from the fire and to load material by means of a long beam which can be introduced at this point from the outside of the building, the chimney being on one side of the furnace and connected by a flue. Another method of charging is described in *Foundry*, Jan., 1906, XXVII, p. 227, in which suggestions are also made for regenerating the air. Regeneration would, of course, give better control and fuel economy, but the cost of the apparatus would be large and it is doubtful if a regular open hearth furnace would not be the best solution in case regeneration were to be adopted, there being several such furnaces used for the purpose at the present time and apparently with great success.

Dr. R. Moldenke, the popular Secretary of the American Foundrymen's Association, has also collected data on air furnace bottoms in malleable practice by sending out a number of questions to which answers were received from 20 foundries. All of these plants use fire sand containing over 95 per cent. silica with just enough base to "set" it at the heat of the furnace. The bottom is made up from this by a method similar to the making of open hearth bottoms. the sand being spread in layers of about one inch thick and fired for about 15 minutes. Bottoms are about 5 in. thick and last, as a rule, from three to 12 heats. At two foundries where ground fire brick was mixed with the fire sand, however, a life of 29 and 30 heats respectively was obtained.[2]

Open Shops.—Difficulties between the Molders' Union and the National Founders' Association has resulted in there being fewer shops where only union labor may be employed, with the result, also, of increased use of molding machines. An important step has been taken by the National Founders' Association in publishing a list of the open shops,[3] listed by localities, so that non-union molders will know where to look for work.

Foundry Supply Exhibit.—During the convention of the American Foundrymen's Association at Cleveland, Ohio, in June, 1906, an exhibit was made by many of the large dealers in foundry supplies, machines,

[1] See also *Iron Age*, LXXVIII[1], pp. 16–19.
[2] *Iron Trade Review*, April 5, 1906, XXXIX, p. 36.
[3] *Iron Trade Review* Dec. 6 1906, XXXIX, pp. 24 and 25

etc. This exposition was of incalculable benefit, both to the foundrymen and the dealers, from a business and educational standpoint. The exhibits were photographed and described in some of the technical journals, and similar exhibits are to be made on even a more advanced scale for the next annual meeting in Philadelphia, in May, 1907.

Foundry Preparatory Schools.—The National Founders' Association has established at least two foundry preparatory schools in which men's hands are to be trained for foundry work. The enterprise has proved itself a great success and has apparently been the means of bringing a number of new men into the foundry business. In order to give a man an opportunity to decide whether or not he would care to enter this kind of work permanently, he is obliged first to serve three days as a helper, at regular wages, in some foundry. If satisfactory, he then returns to the school for 10 days, where he is instructed in the manual operations and in the management of the various modern devices with which the schools are fully equipped. During his schooling he is paid at the rate of $1.50 per day, which he is expected to return at the rate of $1.50 per week after he gets to work.

Cupola Blast Pressures.—For some time foundrymen have been warned that in their striving for large output from too small cupolas, they have been using blast pressure too high to give the best and most economical results. They have been led on largely by the makers of cupolas, for, in this regard as in many others, the maker has been the foundrymen's "guide, counselor and friend" and almost his only instructor. Pressures of a pound for 4 ft. diameter cupolas and of 1.5 lb. for 6 to 7 ft. cupolas have not been, and still are not, rare, although about half that pressure is the maximum that is advisable. The higher pressure gives a larger output, because it raises the melting zone, but for the same reason it burns more fuel, oxidizes more metal and, therefore, gives a larger loss in silicon and manganese, larger wear of lining, more slag, more melting loss, more power needed in proportion to iron produced and poorer quality of castings because of more sulphur.

More Steel Castings.—Steel castings are replacing iron castings in many lines of work, there being a much greater increase in the tonnage of the former than of the latter. In 1906 there was noticeable, in locomotive work especially, such a replacement. About three-quarters of all the locomotive frames are now made of steel and there is almost no part in which steel has not been tried. Especially, advances have been made in the use of steel castings for cylinders and particularly of large size such as compound cylinders, etc.

Use of Thermit.—The use of the Goldschmidt process continues to increase rapidly for the repairing of broken or defective castings. The

chief progress of the year has been in the observation that the welding of the thermit metal is more regular if the mold around the broken part, into which the thermit metal is poured, is made absolutely dry and also if the casting is pre-heated to a good red heat. This result is obtained by forcing hot air from a Well's burner or from a coke basket upon the metal. After the mold is made, the hot air is forced through the riser and escapes by the gate. In welding gray iron castings, good results can be obtained by mixing up to 45 per cent. of punchings in the thermit steel. In welding pipe lines an improvement has been introduced by making the molds of cast iron, which will require no sand backing and last indefinitely.

The most important Operations performed with thermit in 1906 are the following:—The welding of the stern post of the steamship "Puritan" (1547 tons), April, 1906, in Chicago; the welding of the stern post of the steamship "Alaska" (1288 tons), Buffalo, in May; the stern post of the tug boat "Murrell" (311 tons), Boston, in July; the main shaft of the stern wheelers "J. D. Peters" and "Capt. Weber," at Stockton, Cal., and of several large bucket arms—one in particular, on a dredger bucket weighing 12 tons, with 6 cu. yd. capacity.

Expansion of Cast Iron.—It is well known that when cast iron solidifies, the graphite separates from it and this precipitation causes an expansion of the metal, in relation to the amount of graphite and the size of the flakes. Prof. Thomas Turner[1] has devised an apparatus, whereby this expansion and the subsequent shrinkage of cast iron may be autographically recorded. The apparatus is very simple to make and use and gives very valuable information as to the analysis and quality of the pig iron in relation to the rate of cooling and other details of practice. There is probably no other single test which gives us such extended information about the metal, while it also tells us the net result of several counteracting influences.

Miscellaneous.—Progress has been made in molding by the use of collapsible patterns,[2] in cleaning molds by sucking the fallen dust away instead of blowing it off, whereby there is less danger of injuring other parts of the sand.

Miscellaneous.

Electric Smelting.—There are now 20 electric furnaces in operation in Europe on the production of iron or steel, as follows: France, nine; Germany, four; Sweden, Italy and Switzerland, two each; Spain, one. There is one in Sheffield, England, and five in the United States and Canada, one of which is not operating at present. The different processes are described in a continued series of articles by John B. C. Kershaw, in

[1]*Journ.* of the Iron and Steel Institute, 1906, No. 1, pp. 48–74.
[2]*Foundry*, March, 1907, XXIX, p. 116

Vol. XXXIX, 1906, of the *Iron Trade Review*. The greater number of those operating are of the induction type similar to the designs of Colby and Kjellin, and of the Héroult type. Very early in 1907 the Court of Appeals of the District of Columbia, U. S. A., rendered a decision of great importance in favor of Dr. Héroult, whereby he is allowed to have his American patents re-issued. As they were originally drawn by attorneys in Paris comparatively ignorant of American law, with American correspondents, their application was very limited. The claims of the re-issue allowed are very broad.

Roe Puddler.—A new mechanical puddling furnace, patented by James P. Roe, of Pottstown, Penn., was described at the July meeting of the Iron and Steel Institute.[1] One of these furnaces has been in operation at Pottstown for two or three years with success, and those who have seen it in operation speak favorably of its probable future. A possible bar to its early introduction is the rather high cost of installation.

Alloy Steels.—A great deal of research has been carried on upon alloy steels without any very important extension of their application. The special steel of Hadfield, with extraordinary magnetic permeability, is being employed increasingly, and also Guillaume's "invar," which has passed at least one very thorough test by the U. S. Geodetic Survey and proved that measuring instruments made of it have an accuracy not approached by ordinary steels[2].

Corrosion.—There has been a large amount of interest shown in the corrosion of steel,[3] and especially in the relative rate of corrosion of cast iron, wrought iron and steel. It cannot be said that any very reliable data exist on this subject, but it is evident that cast iron corrodes the least and that steel, as ordinarily made corrodes faster than wrought iron, though not as much faster as some would have us believe on the basis of observations that are not accurate or trustworthy. There is also fear expressed that concrete made from cinder will cause a rapid rate of corrosion of steel used for re-inforcement, and that this is a new danger in the use of this class of structural material which is now increasing with great rapidity.

Galvanizing.—The wet and dry galvanizing processes have been much discussed, and some new processes have been developed.[4]

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[1] *Journal*, 1906, No. III, pp. 264-306.

[2] For an account of these results see, *Proceedings* of the American Academy for the Advancement of Science, Dec. 1906. For some other notes on alloy steels see: *Revue de Metallurgie*, 1906, III, pp. 155, 462, 558; 1907, IV, p. 5.
Iron Age, 1906, LXXVII[1], p. 666; LXXVII[2], pp. 1273, 1401; 1906, LXXVIII[1], pp. 268, 680, 1668; 1907, LXXIX[1], pp. 8, 822, 987.
Iron Trade Review, May 24, 1906, XXXIX, pp. 21 and 29; Nov. 1, 1906, p. 27; 1907, XL, p. 136.
Electrochem. and Met. Ind., 1906, IV, pp. 105, 195, 247: 1907, V, p. 9.
Engineering News, 1906, LVI, p. 580.

[3] *Iron Age*, 1906, LXXVII[1], p. 207; LXXVII[2], pp. 1262 and 2072; LXXVIII[1], pp. 80, 152, 277, 287; 1907, LXXIX[1], pp. 141, 478, 731, 816. *Engineering News*, 1906, LV, p. 715.

[4] *Iron Trade Review*, Feb. 15, 1906, XXXIX, p. 17. *Iron Age*; 1906, LXXVII, pp. 260, 352, 514, 882, 1980; LXXVIII, p. 838; *Foundry*, 1906, XXVII, p. 245.

Autogeneous Welding.—Very useful progress has been made in autogeneous welding processes.[1]

Uses of Steel.—Probably the most important expansion in the uses of steel in America is the constantly increasing amount of production and consumption of wire and wire products, which now amounts to about one-half that of railroad rails. Bank vaults and safes are now being made of armor plate, which serves to give employment when the Government is not having this material made. In this connection I may refer to the interest in America because an order was given in this country for armor plate for the Italian government.

Steel Ties.—Each year there has been more tonnage of steel ties made, both because of the gradually increasing price of wood, the short life of that material and the cheapness with which metal ties may be made by re-rolling old shapes. This has occupied a large amount of space in technical journals during 1906, especially in the *Iron Trade Review, Engineering News, Iron Age, Railway Maintenance and Structures* and other railway magazines. The steel tie received a bad set-back when the Pennsylvania ordered no more to be put in its lines and that the existing ones be taken up. The reason for this action was because it was believed that a wreck due to the shearing of tie-bolts by a train running at high speed around a curve was rendered more serious by the easy slip of the rails along the steel ties. In a letter to the *Railroad Gazette* after the accident, Chief Engineer A. C. Shand says that some substitute for wooden ties must be found in the near future and that he has no doubt that experiments with steel ties will go forward on his (Pennsylvania) line as well as on others.

Art of Cutting Metals.—I cannot conclude this review without referring to an extraordinary article written by Fred W. Taylor, one of the inventors of what is now called "high speed steel," and presented as a presidential address to the American Society of Mechanical Engineers, December, 1906. For thoroughness, for frankness and for the amount of valuable information given the article is almost without a parallel in the subject treated and those allied to it.

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[1] *Iron Age*, 1906, LXXVIII[2], pp. 1149, 1437; *Iron Trade Review*, 1907, XL, p. 305; *Foundry*, 1907, XXIX, p. 65; *Electrochem. and Met Ind.*, 1906, IV, p. 284.

LEAD.

By W. R. Ingalls.

The production of lead in the United States in 1906 showed a further increase, but it was insufficient to supply the requirements for consumption, and a considerable amount had to be imported. The statistics of production are given in the accompanying table, which is based on reports from all the producers except one, whose output has been estimated on the basis of its ore supply. These statistics represent the production of refined lead. The smelting industry is so complex that it is difficult to make an accurate distribution among the States. A fairly close separation can be made, however, between the lead of domestic origin and that of foreign origin, because the latter is dutiable. Even under that circumstance, however, the separation is not precise, because the Government assesses duty on only 90 per cent. of the work-lead and lead content of ore imported, allowing 10 per cent. for loss in smelting and refining; but the actual loss in those processes is not so large as that, wherefore the smelters and refiners make a certain gain on their imports, which they are able to market as domestic lead. The statistics reported herewith are accurate so far as they represent the quantity of lead actually produced in final marketable form. They include a certain amount of old lead which comes back to certain of the refiners for reconversion into refined lead.

STATISTICS OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Produced from Domestic Ores.				Imported in Ores and Bullion.	Total Production and Imports.	Exported in all Forms.
	Desilverized.	Soft. (a)	Antimonial. (b)	Totals.			
1897	144,649	45,710	7,359	197,718	92,117	302,859	60,353
1898	169,364	50,468	8,643	228,475	89,209	348,845	78,168
1899	171,495	40,508	7,377	217,085	76,423	317,196	74,944
1900	221,278	47,923	9,906	279,107	114,397	425,824	100,288
1901	211,368	57,898	10,656	279,922	112,471	458,033	100,026
1902	199,615	70,424	10,485	280,524	107,715	458,456	82,228
1903	188,943	78,298	9,453	276,694	106,407	418,601	81,971
1904	200,858	90,470	10,876	302,204	112,852	415,056	84,142
1905	205,665	105,623	11,186	322,474	98,378	420,852	59,741
1906	220,095	118,000	10,120	348,215	84,134	432,349	47,574

(a) In 1905 and 1906 a large part of the so-called soft lead was desilverized, but this (being of Missouri origin) has been included in the old classification. (b) The entire production of antimonial lead is entered as of domestic production, although part of it is of foreign origin. In 1905, the first year in which the separation has been possible, the antimonial lead from foreign sources amounted to 2730 tons. In 1906 the antimonial lead from foreign sources was 2686 tons.

METALLURGICAL PRODUCTION OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Domestic Origin.						Foreign Origin.		Grand Total.
	Desilverized	Antimonial	S. E. Mo.	S. W. Mo.	Miscel.	Total.	Desilverized	Antimonial	
1905	205,665	8,456	81,299	21,324	3,000	319,744	83,504	2,730	405,978
1906	220,095	7,434	100,492	16.528	980	345,529	67,441	2,686	415,656

IMPORTS OF LEAD IN ORE, BASE BULLION, PIGS, BARS AND OLD. (a)
(In tons of 2000 ℔.)

Source.	1901.	1902.	1903.	1904.	1905.	1906
United Kingdom	201.3	396.3	776.4	247.3	795.0	4,926.4
Germany	335.6	476.4	704.9	365.6	125.1	1,003.2
Other Europe	1.2	671.1	225.7	82.8	58.8	1,960.5
Canada	26,065.0	9,732.5	9,600.4	8,951.9	8,181.5	9,257.1
Mexico	81,726.8	93,742.3	93,068.3	102,903.0	87,583.8	66,756.3
South America	4,108.7	2,690.1	1,947.8	290.0	1,577.2	157.6
Other Countries	32.5	6.1	83.2	11.0	56.3	73.7
Total	112,471.1	107,714.8	106,406.7	112,851.7	98,377.7	84,134.8

(a) Refined lead, i. e., in pigs, bars and old, is a small part of the total. It was in 1901, 604 tons; 1902, 2,529 tons; 1903, 3,023 tons; 1904, 8,724 tons; 1905, 5,720 tons; 1906, 11,763 tons.

As to the classification in the first table of this article, it is to be further noted that it is to a large extent conventional. The quantities credited to Southeast and Southwest Missouri represent the lead actually smelted in those districts, not the ore production of the districts. The smelters of Southeast Missouri obtain considerable lead ore from the Joplin district, and also obtain small quantities from outside of Missouri. A considerable portion of the lead produced in the Southeast Missouri district (in which we include the two large smelters in Illinois, near St. Louis) is desilverized, which is done, however, rather to improve the quality of the lead than to extract the small quantity of silver which it contains. The lead entered as "desilverized" in the above table is only the product of the high-grade argentiferous work-lead of the Far West.

The average price of pig lead (desilverized) at New York in 1906 was 5.657c. per ℔. against 4.707c. in 1905. Upward of 90 per cent. of the lead produced from domestic ores is refined and sold by three interests, chief among which is the American Smelting and Refining Company.[1] The market for pig lead is largely controlled by the latter company, and at present the price is governed by it, inasmuch as the selling price fixed by it is adopted by the other large producers. The price established by the American Smelting and Refining Company is for nearby shipments of desilverized lead in 50-ton lots, or larger. The bulk of all the lead produced goes into consumption on that basis, the large consumers making contracts for their supply and paying for it at the average of the settling prices for the month. A small amount of lead was sold during 1906 by the smaller producers and out of warehouse by independent dealers at a premium, giving rise to quotations higher than those of the trust, but the total volume of this business was insignificant in comparison with the great production of lead which was marketed on the terms stated above.

In another table an attempt has been made to show the productio

[1]The precise percentage of the output in 1906 controlled by these interests was 91:5. In 1907 the United Sta Smelting, Refining and Mining Company will be another important factor in the trade.

of lead according to States of origin, the authorities for the figures being given in the foot-note to the table. The total of these figures differs from the total of refined lead production, which of course is to be expected. The two tables are compiled in radically different manners. The first represents the refined lead actually produced, but it includes some "exempt" lead of foreign origin, and also includes about 1500 tons of antimony (content of antimonial lead) of which no account is taken in statistics based on ore production as reported by mines, or on the production of work-lead as reported by smelters. Difficulties are involved in statistics based upon either of those methods. So long as ores are bought on the fire assay their actual lead content is known only approximately, and the allowance to be made for loss in smelting is uncertain. The work-lead of the smelters is not pure lead and it is subject to a loss in refining, wherefore statistics based upon it do not show the actual production of the United States. Discrepancies in the lead statistics of the United States are therefore inevitable. It is considered most accurate to adopt the total production of refined lead, viz, 345,529 tons, and to regard the distribution by States as only an approximation, which, however, in the case of several of the States is close.

CONSUMPTION OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

	1905	1906		1905	1906
Supply:			Deduction:		
Total production desilverized	289,169	287,536	Re-exports of foreign	58,631	47,223
Domestic production soft lead	105,623	118,000	Exports of domestic lead	63	74
Production antimonial lead	11,186	10,120	Stock, domestic lead, Dec. 31	4,000	4,000
Imports foreign refined lead	5,720	11,763	Stock, foreign in bond, Dec. 31	8,148	5,691
Stock, domestic lead, Jan. 1	10,000	4,000			
Stock, foreign in bond, Jan. 1	11,481	8,148	Total deductions	70,842	56,988
Total supply	433,179	439,567	Apparent home consumption	362,331	380,122

PRODUCTION OF LEAD BY STATES.
(In tons of 2000 lb.)

State	(a) 1901	(a) 1902	(b) 1903	(b) 1904	(d) 1905	1906
Colorado	73,265	51,833	43,276	49,290	(f) 57,856	(f) 52,992
Idaho	79,654	84,742	94,611	103,411	(h) 107,000	(i) 121,584
Missouri-Kansas	(c) 67,172	(c) 79,445	(c) 86,439	(c) 92,119	(g) 102,500	(k) 115,103
Montana	5,791	4,438	3,138	3,454	(e) 3,500	(k) 2,485
Utah	49,870	53.914	48,573	53,647	44,500	(k) 56,268
Others	8,452	6,425	6,365	5,283	(e) 4,900	(l) 6,877
Totals	284,204	280,797	282,204	307,204	320,256	355,309

(a) Statistics of the U. S. Geological Survey, representing "lead content of ore smelted." (b) Statistics of the U. S. Geological Survey, representing production of merchant lead. (c) Includes also the production of Wisconsin, Illinois, Iowa, Virginia and Kentucky. (d) Production of crude lead, distributed according to State of origin. (e) Estimated. (f) Report of State Commissioner of Mines. (g) Includes 1500 tons from Iowa, Illinois and Wisconsin; this may not represent the total production of these States, anything in excess appearing under the classification of "Others." (h) Partly estimated. (i) Report of State Inspector of Mines, less allowance of 5 per cent. for loss in smelting. (k) Smelters' eports. (l) Chiefly the production of Wisconsin, Arizona and Nevada.

LEAD PRODUCTION OF THE WORLD. (*a*)
(In metric tons.)

Year.	Australasia.	Austria.	Belgium.	Canada.	Chile.	France.	Germany.	Greece.	Hungary.
1897......	22,000	9,860	17,023	17,698	370	9,916	118,881	16,468	2,527
1898......	67,000	10,340	19,330	14,477	13	10,920	132,742	19,193	2,305
1899......	87,600	9,736	15,700	9,917	171	15,981	129,225	19,059	2,166
1900......	87,100	10,650	16,865	28,654	14	15,210	121,513	16,396	2,030
1901......	90,000	10,161	18,760	23,452	455	21,000	123,098	17,644	2,029
1902......	90,000	11,300	19,504	11,478	(*e*)500	18,817	140,331	15,668	2,243
1903......	141,446	12,162	22,263	9,070	71	(*c*)23,332	145,319	16,093	2,057
1904......	118,979	12,645	23,470	19,000	(*e*) 50	(*c*)18,288	137,580	14,320	2,103
1905......	(*c*)104,639	(*c*)12,968	22,885	25,650	(*e*) 50	24,148	152,590	10,447	(*c*)2,100
1906......	(*d*) 93,000	(*f*)16,400	(*d*)22,200	24,560	(*e*) 50	(*d*)24,300	150,741	(*d*)12,100	(*g*)

Year.	Italy.	Japan.	Mexico.	Russia.	Spain.	Sweden.	United Kingdom. Foreign Ores.	United Kingdom. Domestic Ores.	United States.	Totals. (*h*)
1897...	22,407	1,737	71,637	450	189,216	1,480	13,312	26,988	179,369	721,167
1898...	24,543	1,705	71,442	241	198,392	1,559	23,239	25,761	207,271	798,615
1899...	20,543	1,989	84,656	322	184,007	1,606	17,571	23,929	196,938	820,873
1900...	23,763	1,877	63,827	221	176,600	1,424	10,738	24,762	253,204	854,407
1901...	25,796	1,806	94,194	156	169,294	988	19,639	20,361	253,944	892,861
1902...	26,494	1,644	106,805	225	177,560	842	9,113	17,987	254,489	926,870
1903...	22,126	1,728	(*b*) 94,181	225	175,109	678	14,900	19,958	250,992	951,710
1904...	23,475	(*e*)1,700	(*e*)103,000	(*e*)225	185,862	589	6,888	19,838	274,132	962,144
1905...	(*c*)19,077	(*e*)1,500	(*e*) 87,393	(*e*)200	(*c*)185,693	(*e*)500	(*e*) 5,000	(*e*)20,613	292,519	967,972
1906...	(*d*)21,400	(*d*)4,000	(*e*) 66,610	(*e*)300	(*e*)190,000	(*e*)600	(*e*) 4,000	(*e*)19,000	315,869	965,130

(*a*) From official reports of the respective countries, except the United States. (*b*) Net exports. (*c*) Commercial statistics of Julius Matton, London. (*d*) As reported by the Metallgesellschaft, Frankfurt am Main. (*e*) Estimated. (*f*) Including Hungary. (*g*) Included with Austria. (*h*) The totals are somewhat too high, because of certain duplications which it is impossible to eliminate.

The prominent features of the lead industry in 1906 need be only briefly mentioned. Almost all of the important lead-producing districts showed gains and the increase in the production was attributable especially to them. viz, the Cœur d'Alene, Southeastern Missouri, the Joplin district, and Park City, Utah. The operation of new smelters in Idaho, New Mexico, and Arizona added to the total. Leadville, Colo., showed a decrease. The lead production by districts was as follows: Cœur d'Alene, 119,533 tons; Southeast Missouri, 92,986; Leadville, 23,918; Joplin district, 27,431. The figure for the Joplin district is computed as 70 per cent. of the ore production, and includes the ore that was used for making sublimed white lead.

The conditions of lead mining in Missouri and Idaho are described in following articles. Outside of the leading lead-producing districts, the most important event of the year was possibly the reopening of the old mines at Eureka, Nev., which are now being unwatered. In the meanwhile low grade ore and slag from the old dumps are being shipped to Salt Lake City. If these mines, operated under modern conditions, are able to approach their former record, they will add something to the much needed supply of lead ore.

Attention may be called to the increasing competition in the market for lead ores. The United States Smelting Company, of Salt Lake City, the Ohio & Colorado Smelting Company, of Salida, Colo., and the Pennsylvania Smelting Company, of Pittsburg, Penn., are now decidedly more than insignificant competitors of the American Smelting and Refining Company. The Bingham Consolidated Mining Company, of Utah, was planning to erect lead furnaces in connection with its copper smeltery, but although these plans may be deferred on account of the smoke injunction, it is likely that this company will eventually go into the business, inasmuch as interests identified with it have been investigating and optioning lead mines in the Cœur d'Alene. Moreover, there is an increase in the number of small lead producers in various States.

California.—The new smelter of the Great Western Ore Purchasing Company at Keeler, Inyo county, went into operation in 1906, smelting old slag from Cerro Gordo, where furnaces were run on a rather extensive scale in 1869-1876. The total production of refined lead of California origin in 1906 was 306 tons.

Colorado.—According to the State Commissioner of Mines, the production of lead in Colorado in 1906 was 52,992 tons, Leadville contributing 23,918 tons, and being as usual the largest producer of the State. The total production of refined lead of Colorado origin according to the reports received by us from the smelters and refiners was 50,488 tons. Included in this total is the lead obtained from zinc smelting residues received from Pueblo, Colo., and Iola, Kan., the origin of which is partly Colorado and partly other States. The United States Zinc Company of Pueblo, Colo., produces a small amount of pig lead as a by-product in refining spelter. The United States Smelting Company, of Cañon City, Colo., produces some lead in the form of pigment, made directly from ore. The latter is not included in statistics based on work-lead or refined lead production.

LEAD PRODUCTION OF COLORADO.
(In tons of 2000 lb.)

County.	1900	1901	1902	1903	1904	1905	1906
Clear Creek	2,497	1,945	1,641	1,726	1,981	1,631	1,439
Hinsdale	4,689	3,705	3,107	230	521	446	442
Lake	31,300	28,180	19,725	18,177	23,590	26,424	23,918
Mineral	7,476	5,260	4,646	4,300	6,673	5,940	7,443
Ouray	4,739	3,952	2,131	1,675	1,022	2,674	2,861
Pitkin	13,726	16,375	12,487	16,635	9,441	10,987	8,781
San Juan	8,789	7,736	3,850	3,485	4,644	3,223	2,070
Others	8,921	6,813	5,565	4,529	5,901	6,531	6,038
Total	82,137	74,056	53,152	50,757	53,773	57,856	52,992

Idaho.—According to our reports based on the refined lead actually produced by the smelters as originating from Idaho ore, the output of this

State in 1906 was 115,526 tons. According to the statistics of Robert N. Bell, State inspector of mines, based on reports from the mines, with an allowance of 5 per cent. for loss in smelting, the output was 121,584 tons, of which the Cœur d'Alene contributed 119,533 tons.

(By Robert N. Bell.)—The lead-silver mines of this State continue to progress. The most noteworthy development during 1906 was the performance of the Bunker Hill & Sullivan mine, at Wardner; in spite of a nasty fire in one of its principal stopes during the early months of the year, which greatly retarded production, the dividends paid by this property during the year rank it among the most profitable metal mines in the United States. The other great mines of the district all continued to show remarkable improvement in their lower levels, particularly the big mines of the Federal Mining and Smelting Company, whose combined shipments of high grade mineral during 1906 aggregated 135,000 tons and yielded a net profit of over $2,500,000.

Several new lead-silver producers entered the shipping list during 1906, and when more fully equipped are likely to make paying mines. Noteworthy among these are the Tamarack & Chesapeake, Callahan, Rex, Bear Top, Pittsburg, and the Success, the last being responsible for the bulk of the zinc ore output of the district as high grade concentrate from a new mill which is now in very successful operation. The Hunter mine at Mullan also made an important production; it is being developed at depth by a long drain-tunnel which will greatly facilitate its output, and is likely to open higher-grade ore.

There were fully 4000 men at work in the district. Many new companies were incorporated, and new ore bodies of both lead and copper will unquestionably be disclosed in the near future. Some extensive preliminary work has already been done in the Park copper belt, south of Mullan, which had been lying dormant until the recent disclosures of the Snowstorm revived interest in that section of the district. The Frisco mine is expected to enter the shipping list during 1907 and should make an important yield of both lead and zinc mineral.

Among the other lead-producing districts of Idaho the output for 1906 was not much improved. The principal producer of the Wood River district, the Minnie Moore, suspended its output early in the year. This mine has yielded high-grade ore both in lead and silver, and has a record of over $8,000,000 produced. Development was cut off by a fault last year, and the ore has not been found beyond it, although a large amount of work has been done for that purpose. The total shipments from the Wood River district during 1906 amounted to 200 cars, ranging from 10 to 60 per cent. lead, and from 20 to 130 oz. silver per ton. There were several hundred men at work on development in the district. None of

the old mines has been completely bottomed, and it is likely that deep work on its more important fissures will disclose new ore bodies.

In Lemhi county, the Gilmore mine shipped over 1500 tons of high grade lead-carbonate ore and concentrates during 1906 containing an average of 60 per cent. lead and 30 oz. of silver per ton, and had 500 tons of similar grade on hand at the close of November, which it was unable to get hauled to the railroad. This property has been worked conservatively with little capital investment. The mine carries a pronounced fissure in blue limestone, which has been developed to a depth of 300 ft., by a vertical shaft. It has yielded in the neighborhood of $300,000 gross, and its ore now exposed (principally between the 200- and 300-ft. levels) is estimated at a like value. On the opposite side of the valley from the Gilmore the Viola mine produced ore to the value of $5,000,000 in the early '80s.

One of the most extensive and promising silver-lead prospects in Idaho has recently been taken over by Wayne Darlington, former State Engineer, on a working option for the Bagdad-Chase Mining Company. These are the South Mountain mines, situated near the extreme southwest corner of Idaho, in Owyhee county, 25 miles southwest of Silver City. The property consists of 15 patented claims owned by George A. Sonnemann, of Spokane, Wash., and is being developed by a force of 20 men. The Sonneman property covers the choicest mineralized section of the limestone lode for two miles of its course, and comprises the best old claims that formed the basis of an excitement in 1873, when South Mountain had a population, for a short period, of 1000. A small 30-ton smelter was built and $250,000 worth of bullion run out; a like amount of crude ore was shipped to the railroad at Winnemucca, Nev., which was the nearest shipping point at that day.

Kentucky.—The fluorspar mines, in the vicinity of Marion, Ky., and Rosiclare, Ill., produced a small amount of galena ore as a by-product in 1906.

Missouri and Kansas.—With the increased production of zinc ore in the Joplin district, the production of lead ore naturally showed an increase also. The industrial conditions of the district are discussed under "Zinc" in a subsequent part of this volume. The statistics of lead ore production are given in the accompanying table.

PRODUCTION OF LEAD ORE IN THE JOPLIN DISTRICT.
(Tons of 2000 lb.)

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1894........	32,190	1897....	30,105	1900....	29,132	1903.. .	28,656	1906	39,189
1895........	31,294	1898....	26,687	1901....	35,177	1904 ...	34,362	....	....
1896........	27,721	1899....	23,888	1902....	31,625	1905....	31,679	....	....

PRICE OF LEAD ORE AT JOPLIN.
(Per ton of 2000 lb.)

Year.	1899	1900	1901	1902	1903	1904	1905	1906
Highest	$55.00	$56.50	$47.50	$50.00	$60.50	$62.00	$80.00	$87.00
Average	51.34	48.32	45.99	46.10	54.12	54.80	62.12	77.78

In Southeastern Missouri, the bulk of the production was made by the smelters at Collinsville and at Alton, Ill., and at Herculaneum, Mo. These smelters treated considerable ore from other States than Missouri. At the Herculaneum works (St Joseph Lead Company) a third water-jacket furnace, 42x160 in. was erected and the last of the old circular furnaces was put out of commission. The total lead production of Southeastern Missouri in 1906 was 92,986 tons, against 83,500 tons in 1905. Lead mining in Southeastern Missouri is reviewed in the following article.

(By H. A. Wheeler.)—The output of the Southeastern Missouri lead district approximated 93,000 short tons of pig lead in 1906. Of this, St. Francois county, or the Bonne Terre and Flat River districts, contributed 90 per cent. and Madison county, or Fredericktown and Mine la Motte, produced about 9 per cent., while the small mines scattered throughout the adjoining Washington, Jefferson and Franklin counties produced much less than usual, or only about 1 per cent. This is about a 5 per cent. increase over 1905 and is the largest output in the history of the district. The prices received for the lead were also at high-water mark, as they ranged from $5.25 to $6 per 100 lb. in the St. Louis market. The year was prosperous in every respect and labor shared in the prosperity, as it never received such high wages as in 1906. There were no labor troubles, but men were scarce; however, the large companies managed to finish the year without serious curtailment. The outlying districts of Washington, Franklin and Jefferson counties were seriously affected, as the tributers, or small leasers, who produce the lead in those districts were attracted and drawn off by the high wages paid at the big mines, so that their output was much below normal. The latter has become such a small factor, since the opening up of the deeper disseminated ore bodies, that the deficiency was more than made up in the progress of the St. Francois county mines. The increased output was not due to the development of any new companies, but merely to the greater activity of the established mines that was stimulated by the high price for lead.

The energetic prospecting for new ore bodies by the large companies inaugurated in 1905 continued through 1906 and many thousands of acres were optioned and drilled, though in most cases the drilling was inadequate to demonstrate the value of the land. One of the older properties

acquired by the Federal Lead Company under such an option was the land of the St. Louis Prospecting Company, for which $500 an acre was paid.

Central Mine.—The most important event of 1906 was the successful unwatering of the main mine of the Central Lead Company (which was drowned out three years ago during the miners' strike) and the erection of a new mill. The old Central mill was started as a 200-ton mill and later was enlarged to 400 tons at a time when capital was scarce; it has done excellent work in grinding out dividends. Now it is to be superseded by a fine, modern, steel-frame mill that will be of 1000 tons daily capacity. A new shaft has been put down on the 40-acre tract in Flat River of the Central company that will prove a valuable acquisition in maintaining the success of this property.

St. Joseph Lead Company.—This company materially strengthened and improved its old plant at Bonne Terre, which is still the largest producer of the district. A new shaft was completed at North Bonne Terre, No. 13 or B, that will assist the old mine in maintaining its big production, although the latter is holding up its output admirably in spite of having been worked continuously for nearly 40 years. The new plant at Leadwood, or Owl Creek, was also enlarged, 10 roasting furnaces having been added, and it is now producing almost as much lead as the Bonne Terre plant, but with a much finer, electrically operated, mill. A new shaft is being sunk that will come into production in 1907.

Desloge Consolidated Lead Company.—This company suffered a temporary set-back in the loss of its No. 3 shaft house (its principal producer) by fire in the autumn. The output of this company was therefore smaller than usual, although shafts Nos. 2 and 4 were pushed more energetically to make up for the loss of No. 3. No. 3 will be rebuilt and will be producing again before spring. A new shaft is being completed on the recently acquired land adjoining Washington county that should materially increase the output of 1907.

Doe Run Lead Company.—This company had a very prosperous year and is contemplating the erection of a new, modern mill. It is sinking a new shaft on its Mitchell tract and is considering the re-opening of its old mine at the town of Doe Run, where there is not only a large deposit of low grade ore, but recent discoveries show plenty of high grade ore that was overlooked in the earlier and supposed thorough drilling of the original property.

National Lead Company.—This company has begun to look around for new territory by optioning several tracts that are now being prospected with the diamond drill. It sunk a new (No. 5) shaft at Esther that will open up the eastern end of its property, and is arranging to make new improvements in the mill that will add to its efficiency.

Federal Lead Company.—This company had a very busy year in recovering the principal mine of its new acquisition, the Central Lead Company, and in erecting the mill for it. The latter should largely increase the output of 1907 if its completion is not delayed. The company is also sinking two new shafts on the southern portion of the property that it acquired from the old Missouri Leadfields Company. Additional property has been judiciously acquired and several other tracts have been optioned that are now being prospected. By the end of 1907 this company will be the largest producer in the district with the exception of the St. Joseph Lead Company, and if its present rate of growth continues it will even eclipse that famous property.

Other Companies.—The Columbia Lead Company has eliminated the internal legal troubles that put it into the hands of a receiver, by the majority interests buying out the minority holders, but beyond doing some prospecting, it has not yet started up its mines. The Union Lead Company's property is under option to the Doe Run company, which is drilling it, while the Penicaut lands are under option to the St. Joseph company.

In Madison county, the North American Lead Company is going into the nickel business by utilizing the large body of mixed nickel, cobalt, copper and iron sulphurets that have hitherto been neglected. It is now erecting a large smelting and refining plant that will cost $250,000 and is expected to be ready by July 1, 1907. It will not only work its own ores, but also will treat similar ores that are found on the Catherine and Mine La Motte properties in the same district.

The Madison Land and Mining Company enlarged its plant by putting in a 20-drill air compressor and erecting a new 200-ton mill. Later on it expects to enlarge the latter to 400 tons and then to abandon its first mill, which is only 150 tons capacity. It installed a Hancock jig in the old mill, replacing the Harz jigs, and is so well pleased with it that it will use Hancock jigs in the new mill.

The old Mine La Motte property is being rejuvenated under the new Pittsburg management and is recovering from the blight inflicted on it by the Dougherty & Albers stock scheme. The old copper-mine lease to the Hudson Valley Mining Company was actively and successfully worked during 1906, producing from 18,000 to 20,000 ℔. of concentrates daily.

New Mexico. (By Charles R. Keyes.)—Much of the lead produced in New Mexico is derived from complex ores. In statistics the tonnage of lead ore alone does not appear so large as it should. Many of the zinc ores and copper ores contain large percentages of lead. The largest production of lead ores alone is from Las Cruces, in Doña Ana county, the chief mines being in the Organ mountains about 15 miles distant. The deposits are in contact zones of limestone and intrusive granite. New

and important developments were made during 1906 at Cooks Peak, Granite Gap, Magdalena, and a number of other places. In the Caballos mountains, west of Engle, some large deposits were opened during the year, the ore being a pure galena in a fluorite gangue. Taking into consideration the numerous new developments in complex deposits in which there is also a considerable lead content, it may be expected that during the next few years the lead production of this Territory will greatly increase.

Nevada.—The Richmond-Eureka Mining Company, formed in 1905 by the consolidation of the old Richmond Consolidated and Eureka Consolidated, vigorously prosecuted the unwatering of the old mines, and in the meanwhile made considerable shipments of ore and slag from the dumps, these materials being sent to Salt Lake City for smelting. A small amount of lead ore was produced at White Pine, but the total production of lead in Nevada, which 30 years ago led all the other States, is still insignificant. However, it showed a considerable increase in 1906, amounting in that year to 1669 tons.

Utah. (By L. H. Beason.)—While the mining of copper ore represents the principal resource of Bingham, the production of lead ore is by no means unimportant. The United States company finds it profitable to extract lead ore from its Galena mine, and the Bingham Consolidated from its Lark and Lead-mine veins. Lead ore is a factor in the shipments from the Phœnix, Utah Apex, Utah Development, New England and the Bingham-New Haven mines. The lead ores shipped range from 15 to 35 per cent. lead and from 5 to 15 oz. silver. Since about the holidays the available ore reserves of the last named mine have been greatly augmented by the development of an ore body of large dimensions in its lower tunnel of a character similar to the ore produced in the Utah Consolidated's Highland Boy mine. The company has the construction of a smelting plant under consideration.

Park City has not yet recovered from the Ontario tunnel disaster of almost two years ago, when the avenue through which practically the whole camp is drained was obstructed by caving. The Ontario company, however, has been endeavoring to clear the way and its efforts are about to be crowned with success. The tunnel stoppage prevented the resumption of development and of ore extraction on the lower levels of some of the older mines, but a number of newer exploitations have more than offset the bad results of the tunnel difficulty. The more important developments were made in the Daly-Judge mine, which is now a dividend payer. In February, 1906, this company had an indebtedness of $232,000, which has since been cleared up, and it begins 1907 with $300,000 in its treasury. Other important developments occurred in the Thayne's Cañon section of the camp. Three mills are being operated in the camp at present—by the Daly-Judge, Daly West and Keith-Kearns companies. The Daly-

Judge has realized considerable revenue from the shipment of zinc ore. Since Christmas, 1906, a controlling interest in the Ontario, Daly and Naildriver companies has been purchased by Jacob E. Bamberger, president of the Daly West Mining Company. Although no official statement has been given out to that effect, it is probable that these mines will be merged with the Daly West in some form. The Silver King Mining Company has purchased the properties of the Magnolia-St. Louis and Pinyon Ridge Mining Companies as well as those owned by James McGregor, adding several hundred acres to its already large area and ending a law suit brought by the Magnolia-St. Louis Mining Company to recover $900,000 for ore alleged to have been unlawfully taken by the defendant.

Alta, in the Big Cottonwood district, 25 miles from Salt Lake City, forged ahead in 1906 and promises to be a large producer of gold, silver, copper and lead ores from now on. The Columbus Consolidated is the best developed property in the camp, but large bodies of mill-ore have been proved in the Albion and City Rocks mines. The Albion will, no doubt, provide facilities for treatment during the coming year. Both the City Rocks and Albion, however, contain considerable high-grade ore and shipments are being made. Several important deals were consummated during 1906. The old Flagstaff and Emma mines were sold to Salt Lake and Eastern syndicates, resulting in the formation of the Consolidated Flagstaff and the Emma Copper companies. A consolidation of the South Columbus and Alta Quincy mines, under the title of the South Columbus Consolidated, was effected and the new company has mapped out a broad campaign of development.

Next to Bingham, the Tintic mining district sent the largest tonnage of ore to the smelters. Tintic has the deepest mines in the State and several of the older companies, the Mammoth, Centennial Eureka, Eagle and Blue Bell, and Gemini, have sought lower levels with gratifying results. The leasing system, inaugurated in the Bullion Beck & Champion mine a little over a year ago, has been conducted successfully in other mines and has contributed largely to the increased activity in the camp. The only milling was conducted by the Uncle Sam Consolidated and the Godiva mining companies; the other mines shipped direct to the Salt Lake smelters.

Announcement was made recently of the formation of the Tintic Smelting Company backed by some of the strongest financial interests in Utah. It is the intention of this concern to begin the construction of a lead smelter at a convenient point within the Tintic district. The initial capacity will be about 400 tons of ore per day. This new smelting concern is closely allied to the Utah Smelting Company, operating a small copper smelter near Ogden.

Virginia.—The production of lead ore in this State in 1906 was only 50 tons.

Wisconsin.—The production of lead ore in the Wisconsin-Illinois-Iowa district in 1906 was 2100 tons. The smelting works at Dubuque, Iowa, were operated during the year. The smelter at Dodgeville, Wis., was not operated in 1906, the company owning it having been engaged in the erection of a new plant, which will be completed early in 1907.

Lead Mining in Foreign Countries.

Statistics of the lead production of the world are more tardy than those of the other principal metals, and we are not yet able to present the complete data for 1906. It appears, however, that except in the United States there were no large increases, and in Australia at least there was a decrease.

Australia.—In spite of the great boom at Broken Hill, the exports of lead and lead ore showed a decrease. The pig lead production of Australian smelting works in 1903-1906 is reported by Julius Matton, of London (in tons of 2240 lb.), as follows:

Works.	1903	1904	1905	1906
Broken Hill Proprietary Co..	61,375	68,513	67,062	55,892
Sulphide Corporation........	15,680	23,094	22,246	21,033
Smg. and Ref. Co. of Australia	7,500	10,599	502	....
Tasmanian Smelting Co.....	6,800	7,800	9,000	8,900
Fremantle Smelter..........	250	5,053	2,104	2,850
Queensland................	3,795	2,046	2,422	2.461
Totals..................	95,400	117,105	103,336	91,141

According to the official statistics of the Department of Mines of New South Wales, the production has been as follows, in tons of 2240 lb.:

Lead.	1903	1904	1905	1906
Base bullion	92,293	106,038	93,182	79,925
In ore exported	29,706	59,507	69,044	58,683
Totals................	121,999	165,545	162,226	138,608

Canada.—According to the Dominion Geological Survey the production of lead in 1906 was 27,100 short tons, against 28,280 in 1905. About 95 per cent. of the above figures of production is to be credited to British Columbia, the great bulk being derived from the East Kootenay district. However, when arrangements are completed which will permit of mining the bodies of zinc-lead ores of the Slocan district there is no doubt that a much larger production will be recorded. The St. Eugene increased

its output of lead-silver ore and the Sullivan was also an important producer. On the other hand, the Slocan district fell off. The Blue Bell mine in the Ainsworth division will likely produce a considerable quantity of lead when arrangements for treating its big body of lead-zinc ore are completed.

Germany.—The production of lead in Germany in 1906 was 150,741 metric tons, which was a small decrease from the previous year. The imports and exports of lead ore to and from Germany during the last four years have been as follows (in metric tons):

Year.	Metal.				Ore.			
	1903	1904	1905	1906	1903	1904	1905	1906
Imports	52,440	61,388	78,528	71,040	67,573	83,807	92,667	89,979
Exports	30,243	23,169	32,515	27,039	1,270	1,312	1,496	1,916

Mexico.—The imports of lead into the United States from Mexico were 66,756 short tons in 1906, against 87,584 in 1905. This indicates a falling off in the production, inasmuch as practically the whole of the Mexican production is represented by the American importation.

(By C. A. Bohn.)—The prediction that lead must soon give way to copper as a base for collecting the gold and silver in the ores in smelting does not seem to be proved in the work of 1906. The existing plants that have been using lead continue to do so; and of the new plants, in the building of which there seems to have been somewhat of an epidemic during 1906, the tonnage was about equally divided between copper and lead. Among these the more important custom plants are the Magdalena in Oaxaca, of 100 tons, blown in in July; the Oaxaca Smelting and Refining Company, of Oaxaca, with a lead plant of 250 tons capacity, which was to have been blown in by the end of December, but was held back several months, pending the further development of the company's mines; the Mexican Smelting and Refining Company, Taxco, Guerrero, with a 40-ton lead furnace, blown in in September; the Michoacan Metallurgical Company, at Augaugueo, with a 250-ton copper plant blown in last July; the Guaymas, Sonora, 500-ton lead and copper plant; and the 500-ton lead plant of the American Smelting and Refining Company in course of construction in Chihuahua. The largest new plant to go into commission in 1906 was that of the American Smelters Securities Company, at Velardeña, Durango, intended for 600 tons of lead ore and 900 tons of copper ore, though as yet only two copper stacks of 600 tons daily capacity have been installed and blown in. At this plant all is automatic from the time the ore leaves the railroad cars, belt con-

veyors taking the ore to the mill, thence to the beds, and from the beds to the 5-ton, electrically-driven charge cars, which dump directly into the furnaces. As yet no converters have been erected, the matte being shipped to the American Smelting and Refining Company's plant at Aguascalientes.

Portugal.—Argentiferous lead ores, associated with zinc, prevail in the Avièro and Vizen districts. Native silver is found at the Varzea de Trévoés mine in the former district, but the principal output of this mine is galena and blende. The Terramonte mine of Douro has sent important amounts of lead sulphide ore to Germany.

Rhodesia.—Important deposits of zinc and lead ore have been developed at Broken Hill. The zinc ore is already being exported to Wales. The lead ore, which is of high grade, will be smelted on the spot.

Spain.—Spanish exports increased about 5000 tons, indicating an increased production.

The Lead Markets in 1906.

New York.—Not for 30 years has the lead market ruled at as high an average as during 1906. Its high level was due not only to the fact that a very large proportion of the output is controlled by one interest, but also to the underlying conditions, which were such as fully to warrant the high market. Consumption overtook production, and not only was a large amount of foreign lead, refined in this country, retained for domestic consumption, but also during the summer several thousand tons were actually imported from Europe. More than the normal expansion occurred in all lines of manufacture; an unprecedented tonnage was used up by cable makers.

The increase in the production of lead, particularly in the Far West, was not commensurate with what could have been reasonably expected on account of the market conditions. This was due, in the first place, to the non-discovery of large new lead deposits, and, secondly, to the obstacles in the way of utilizing present facilities, owing to the scarcity of labor and other causes which hampered industrial activity throughout the country. Supplies from Missouri became more plentiful and there was a great deal of prospecting going on there as well as in the Far West, which may be expected to show results during 1907.

The year 1906 opened with lead standing at 5.60c., New York, at which price it remained until Feb. 14, when, on account of a decline in the European market, the quotation was reduced to 5.35c. No change was made until the latter half of April, when the price was advanced to 5.50c., and then in quick succession there were further advances in the early part of May to 5.60c. and 5.75c. The latter price ruled until about

the middle of December, when the demands upon the sellers became so heavy that the price was raised to 6c., at which the year closed.

AVERAGE MONTHLY PRICE OF LEAD PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*
1897......	3.04	3.28	3.41	3.32	3.26	3.33	3.72	3.84	4.30	4.00	2.96	3.70	3.58
1898......	3.65	3.71	3.72	3.63	3.64	3.82	3.95	4.00	3.99	3.78	3.76	3.76	3.78
1899......	4.18	4.49	4.37	4.31	4.44	4.43	4.52	4.57	4.58	4.58	3.70	4.64	4.47
1900......	4.68	4.68	4.68	4.68	4.18	3.90	4.03	4.25	4.35	4.35	4.58	4.35	4.37
1901......	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.15	4.33
1902......	4.00	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.069
1903......	4.075	4.075	4.442	4.567	4.325	4.210	4.075	4.075	4.243	4.375	4.218	4.162	4.237
1904......	4.347	4.375	4.475	4.475	4.423	4.196	4.192	4.111	4.200	4.200	4.200	4.600	4.309
1905......	4.552	4.450	4.470	4.500	4.500	4.500	4.524	4.665	4.850	4.850	5.200	5.422	4.707
1906......	5.600	5.464	5.350	5.404	5.685	5.750	5.750	5.750	5.750	5.750	5.750	5.900	5.657

London.—The year opened rather hopefully for trade generally, soft foreign lead being then obtainable at £17 10s. ex-ship. The first business consisted mostly of speculative purchases, which tended to strengthen the market, but the early pressure of spot parcels checked the advance. Disappointed holders then pushed sales, and prices became irregular. More speculative buying set in, followed by American and Russian orders, which absorbed large quantities, and a gradual improvement was recorded. The month closed quietly with foreign at £16 5s., and English at £16 8s. 9d. per ton. February found the available supplies rather scarce, and spot parcels in request. Later, dealers' purchases, "bear" covering, improved demand for home consumption, and the reported outbreak of a troublesome fire in the Broken Hill mines combined to stiffen prices, which rose to £16 1s. 3d. for foreign, and £16 3s. 9d. for English whereat the month closed. March found buyers rather reserved, particularly when it became known that the Broken Hill mines had resumed their normal output. During the first eight days prices dropped about 15s. under pressure of sales; but a recovery set in, prompted by combined speculative purchases and bear covering, and the month closed with foreign at £15 17s. 6d., and English at £16 2s. 6d.

April started with a downward tendency induced by the pressure of large accumulations in dealers' hands. Then came a gradual recovery. Toward the close of the month the buying became more determined, particularly for forward delivery for American account. English consumers being taken unawares followed with substantial purchases, and the closing prices were £16 5s. for foreign, and £16 7s. 6d. for English. May found consumers encouraged by marked improvement in the building trade. Their purchases coincided with larger inquiry from the Continent, and with substantial orders from America. Spot lots, which at first appeared plentiful, were thus readily absorbed. Toward the end of the month news

came of serious subsidences in Broken Hill Block 10 and Central mines, which pointed to curtailment of the Australian production, and caused an improvement in prices to £16 17s. 6d. for foreign, and £17 for English. June found buyers still active under the influence of the Australian reports, consumers willingly covering requirements a good way forward. Toward the end of the month, a good demand set in for export. Closing prices were £16 13s. 9d. for foreign, and £16 18s. 9d. for English.

July opened with a downward tendency, due mainly to the public marketing of various holdings. Toward the middle of the month spot parcels were scarcer, and prices hardened somewhat. The last two or three days of the month found demand satisfied, and sales could be effected only by making concessions, the last business recorded being £16 12s. 6d., English being held for £16 16s. 3d. August found the market quiet, but without any pressure to sell. Consumptive demand soon asserted itself, and was followed by important orders for export, notably for Russia. At the close foreign brands were quoted £17 10s. and English £17 12s. 6d. September was an active month, with a slow but steady advance. Trade demand was excellent. Offerings were scarcer from day to day, and it soon became apparent that supply was unequal to demand. Closing prices were £18 10s. to £18 12s. 6d. for foreign brands, according to time of delivery, and £18 15s. to £18 17s. 6d. for English.

AVERAGE MONTHLY PRICE PER 2240 lb. OF LEAD AT LONDON. (*a*)
(In pounds sterling.)

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1897......	11.717	11.708	11.562	11.787	11.837	11.912	12.250	12.679	13.650	13.575	13.100	12.600	12.367
1898......	12.508	12.362	12.650	13.062	13.700	13.437	12.950	12.800	12.800	13.050	13.412	13.100	12.983
1899......	13.375	14.350	14.150	14.375	14.146	14.283	14.385	14.733	15.267	16.179	17.096	16.883	14.933
1900......	16.296	16.542	16.612	16.733	16.900	17.225	17.533	17.633	17.667	17.596	17.229	16.233	16.987
1901......	15.925	14.667	13.379	12.421	12.275	12.342	12.150	11.692	11.954	11.600	11.267	10.533	12.521
1902......	10.567	11.617	11.508	11.596	11.600	11.271	11.233	11.121	10.892	10.746	10.717	10.754	11.262
1903......	11.304	11.708	13.225	12.404	11.800	11.437	11.383	11.146	11.167	11.108	11.108	11.179	11.579
1904......	11.558	11.592	12.037	12.254	11.754	11.521	11.667	11.737	11.787	12.187	12.892	12.775	11.983
1905......	12.875	12.462	12.296	12.658	12.762	13.000	13.608	13.958	13.950	14.679	15.337	17.050	13.719
1906......	16.850	16.031	15.922	15.959	16.725	16.813	16.525	17.109	18.266	19.350	19.281	19.609	17.370

(*a*) The statistics for 1897–1905 are from the report of the Metallgesellschaft, Frankfurt am Main. Those for 1906 are from the *Engineering and Mining Journal*.

October brought evidence of a serious depletion of available supply. The larger consumers were quick to apprehend the situation, and bought with more than customary boldness. The price rose steadily until £20 was reached, Oct. 11. Later the metal markets generally were adversely affected by financial trouble, and a general decline ensued. The month closed with foreign brands £19 2s. 6d. to £19 7s. 6d., and English £19 5s. to £19 10s. November revealed a dwindling supply for early delivery. Toward the middle of the month this scarcity was relieved by fresh arrivals, and prices eased off. Fluctuations were narrow throughout the month. Closing prices were £19 5s. to £19 7s. 6d. for foreign

brands, and £19 10s. to £19 12s. 6d. for English. December revealed some scarcity of supply at the outset, but this was counteracted by early arrivals, and for about 10 days the markets showed little animation. The undercurrent, however, was strong, it being evident that the consuming trades were but poorly covered. Dealers began to buy freely and demand was stimulated by postponement of the law prohibiting the manufacture of white lead in France. The result was an advance to £20 for foreign brands, £20 7s. 6d. being asked for English, March delivery commanding 5s. premium. At these figures the market remained steady till the close.

White Lead, Red Lead, Litharge and Orange Mineral.

The production of these products is given in the accompanying table. In addition to the quantity of white lead stated therein, there was an output of 7988 tons of "sublimed white lead" a mixture of lead oxide and sulphate obtained by the volatilization of galena ore, and 5749 tons of "zinc-lead," a pigment abtained by the smelting of blende-galena ore at Cañon City, Colo. A similar product is produced at Coffeyville, Kan., but is included in the statistics of zinc oxide production (see Zinc).

PRODUCTION OF LEAD PIGMENTS IN THE UNITED STATES.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1897	7,798	$744,709	105,804	$9,522,360	8,591	$773,190	477	$76,320
1898	9,160	916,000	93,172	9,391,738	7,460	710,192	541	108,200
1899	10,199	1,070,895	103,466	10,812,197	10,020	1,032,060	928	139,200
1900	10,098	1,050,192	96,408	9,910,742	10,462	1,067,124	825	100,650
1901	13,103	1,448,550	100,787	11,252,653	9,460	979,586	1,087	224,667
1902	11,669	1,262,712	114,658	11,978,172	12,755	1,299,443	867	138,349
1903	12,300	1,385,900	112,700	12,228,024	12,400	1,326,800	1,000	168,000
1904	13,938	1,672,569	(a) 126,336	13,896,913	12,487	1,248,691	1,125	168,681
1905	16,269	1,919,767	(a) 122,398	12,068,443	12,643	1,422,616	1,000	120,000
1906	13,693	1,874,448	(a) 123,640	15,234,297	13,816	1,890,050	2,927	421,488

(a) The output of "sublimed white lead," a mixed sulphate and oxide of lead, is not included in 1904, 1905 and 1906.

IMPORTS OF LEAD PIGMENTS INTO THE UNITED STATES.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1896	1,543,262	$47,450	1,183,538	$52,400	51,050	$1,615	1,359,651	$51,077
1897	1,386,070	46,992	1,101,829	48,988	60,984	1,931	1,486,042	67,549
1898	682,449	25,780	506,739	24,334	56,417	2,021	795,116	37,74[illegible]
1899	776,197	30,479	583,409	30,212	55,127	3,614	1,141,387	58,14[illegible]
1900	549,551	25,532	456,872	28,336	77,314	2,852	1,068,793	61,885
1901	485,466	19,369	384,671	21,226	49,306	1,873	977,644	52,40[illegible]
1902	1,075,839	37,833	506,423	25,320	88,115	2,908	997,494	49,060
1903	1,152,715	40,846	453,284	24,595	42,756	1,464	756,742	36,407
1904	836,077	30,115	587,383	33,788	44,541	1,500	766,469	37,178
1905	704,402	26,553	597,510	34,722	117,759	4,139	628,003	31,106
1906	1,093,619	50,741	647,636	41,233	87,230	3,737	770,342	42,519

Although the production of white lead in 1906 showed a small increase, it is probable that the consumption did not increase in like ratio. This is attributable in a measure to the loss of the Canadian business through the imposition of a prohibitory tariff by the Dominion government, in its effort to assist in the development of a domestic white-lead industry. The effect thus far, however, has been to leave the Canadian consumers of lead pigments largely dependent upon foreign supplies, and to advance prices there very much above what white-lead from Europe or the United States would cost in bond at Dominion ports.

The prices on lead pigments in the United States established late in December, 1905, continued exactly 12 months, when the first change of the year was made, this being an advance of $5 per ton on white-lead and the oxides, on Dec. 20, 1906.

In the face of the stocks, or contracts for deliveries during the year at the prices current late in 1905, held by grinders, the possibilities of competition from graded lead and mixed paints was such as to render corroders unwilling to advance the price on pure lead in oil, although the cost of pig lead had steadily risen. But the last advance made by smelters in December brought the cost of pig lead to $8 per ton above the figures ruling at the opening of the year, and this led to the advance of $5 per ton on the products above referred to.

A strenuous effort is being made by the manufacturers of zinc, in co-operation with the makers of mixed paints and graded leads, to popularize zinc and the so called "inert pigments" such as barytes, silica and whiting, at the expense of white-lead. This is being done through magazine and other forms of advertising which appeal to the public, and the effort forces upon corroders a problem in the solution of which is involved the margin they shall allow between dry lead, used as a base for all paints competing with pure lead in oil, and their price for the latter product. The margin in 1906 was narrower than usual, and although grinders and makers of mixed paints were pretty well supplied with dry lead at low prices, they have foreseen a changed condition in 1907, and have made strong protests to the corroders against so close a relation between the dry pigment and lead in oil. In view of the fact that the interests which control probably 85 per cent. of the production of pig lead now control nearly a like proportion of the production of lead pigments, the relation which they will chose to assume toward the industries that have been prominent factors in the distribution of those products, becomes an interesting question.

The United Lead Company, which was owned by American Smelting and Refining Company interests, was merged with the National Lead Company during the year, and the latter also obtained control of the Davis Lead Company, of Pittsburg, and the Carter White Lead Company, of

Chicago. This leaves six corroding-works outside, with a capacity of about 30,000 tons of white lead and oxides.

The Metallurgy of Lead in 1906.

By W. R. Ingalls.

Roasting.—The chief feature in the metallurgy of lead in the United States in 1906 was the further introduction of the lime roasting process, which has been modified, however, by the metallurgists connected with the American Smelting and Refining Company, so that as used by them it is rather a process of "pot" roasting than "lime" roasting, the distinction hanging upon the point as to whether lime does or does not play a chemical part in the process, i.e., any part other than merely serving as the diluent of the charge, which function may be fulfilled by ferric oxide or other material. A third alternative is that lime or other material may possibly act as catalyzers. The nature of the reactions which occur in these processes of desulphurizing galena is still a disputed point. Pot roasting, as practiced by the American Smelting and Refining Company, is performed on charges low in lead, and free or nearly free from limestone. However, this modification does not appear to be an unqualified success, the time required for the working of a charge being rather long and the proportion of fines produced, which must be re-treated, being rather large. Certainly the results reported are much inferior to those obtained with the Huntington-Heberlein process on high-grade galenas in Australia, Europe, and elsewhere.

Dust and Fume Collection.—In the construction of dust chambers, there appears to be a general reversal of fashion, the flues of reinforced concrete construction, which were so much in vogue a few years ago, being now out of favor. It has been found impossible to prevent them from cracking, which of course is highly undesirable. In new constructions, the flues are built of brick in the form of an inverted catenary.

Another new feature has been the restoration to fashion of the bag-house which has been before the attention of silver-lead smelters since 1880 and found its leading exponent in Doctor Iles. However, it never came into general use. An installation was made at the East Helena works, but a few years ago was abandoned. In 1906, however, the bag-house at East Helena was put into commission again, and the United States Smelting Company at Salt Lake began the installation of a similar fume-saving device.

Electrolytic Refining.—A prominent feature of 1906 was the extension of interest in the Betts process of electrolytic lead refining. The installation of the Consolidated Mining and Smelting Company at Trail, B. C., was materially increased in size, and a new refinery was erected and put in operation by Locke, Blackitt & Co., at Newcastle-on-Tyne, England.

Most important of all was the erection of a plant of 2500 tons per month capacity, near Chicago, by the United States Metals Refining Company. This plant went into operation early in December, but at the end of the year it had not come to good running order and had made no production of refined lead. The various difficulties were overcome early in 1907 and an output of lead was commenced which goes into the market under the brand of "electrolytic." There seems to be no question as to the success of this process of lead refining in so far as the electrolytic work is concerned, but the problem of handling the anode slime does not seem yet to have been satisfactorily solved. Mr. Betts has patented a new process for this purpose, but both at Trail and at Chicago the slimes are being worked up by independent processes. Anyway, the treatment of this material is only an ordinary metallurgical problem from which an economical method will doubtless be developed by a little experimentation.

Recent Improvements in Lead Smelting.

By H. O. Hofman.

Introductory—Physical Properties—Alloys.

New Publications.—W. R. Ingalls, "Lead Smelting and Refining, with some Notes on Lead Mining," *Engineering and Mining Journal*, New York, 1906, pp. vii+327, ill., $3. This book is a reprint of articles on lead smelting and refining, prefaced by a few papers on lead mining, which have appeared in the *Engineering and Mining Journal* and the *Transactions* of the American Institute of Mining Engineers, mainly during the last three years; some papers go as far back as 1900. The table of contents comprises: Notes on lead mining, p. 24; roast-reaction smelting, p. 11; sintering and briquetting, p. 15; smelting in the blast furnace, p. 35; lime-roasting, p. 60; other methods of smelting, p. 9; dust- and fume-recovery, p. 29; lead refining, p. 11; smelting works and refineries, p. 26. The interesting process of lime-roasting receives most attention. While the papers referring to smelting and refining have been reviewed in this publication, it is convenient to have the originals classified and collected in book-form in which they can be readily consulted. The idea of giving quick, handy information regarding new developments in technology is ever to be recommended and will be appreciated, especially by men engaged in practice.

Specific Heat of Lead.—N. Stücker[1] gives the following data for the specific heat of lead for a range of 20–350 deg. C.:

SPECIFIC HEAT OF LEAD.

Deg. C.	20–100	150	200	250	300	350
Spec. Heat	0.03055	0.03094	0.03141	0.03213	0.03289	0.03314

[1]*Sitzungsberichte* der Oesterreichischen Akademie der Wissenschaften, May, 1905; through *Metallurgie*, 1906, III, 487.

Volatility of Lead Oxide.—F. O. Doeltz and C. A. Graumann[1] carried on experiments to determine the volatility of lead oxide. They found that slight losses in weight occurred at 800 deg. C.; these grew quickly at 900 deg. C., when the oxide became liquid, and more slowly at 950 and 1000 deg. C., the highest temperatures reached in the tests. As with melted litharge the volatility is proportional to the exposed surface, the rate of volatilization will depend upon the ratio of surface and mass. Thus the authors varied the loss at 900 deg. C. from 8.8 to 0.04 per cent. by changing the above ratio. Therefore in cupelling, the loss by volatilization would be much greater than it is, if the litharge were not drawn off from the surface of the bath more or less continuously.

Volatility of Lead Sulphide and Decomposition of Lead Sulphate.—Lodin had shown[2] that lead sulphide was volatile at a temperature below its melting point. F. O. Doeltz and C. A. Graumann[3] found that heating lead sulphide in a current of nitrogen for one hour to 860 deg. C. gave a loss in weight of 18 per cent.; doing the same for two hours at 950 deg. C. gave a loss of 38 per cent.

SODIUM-LEAD ALLOYS.

Composition.	Freezing-Point, °C	Na. % Found	Na. % Calculated.	Eutectic Alloy.	Freezing-Point, °C	Na. %
Na_4Pb	386	30.0	30.8	$Na_4Pb + Na_2Pb$	373	25.8
Na_2Pb	405	18.5	18.2	$Na_2Pb + NaPb$	329	13.6
$NaPb$	367	9.8	10.0	$NaPb + Na_2Pb_5$	301	6.2
Na_2Pb_5	319	4.0	4.3	$Na_2Pb_5 + Pb$	307	2.7

It has always been held that lead sulphate was stable at a white heat. The authors found that decomposition began at 900 deg. C. and increased to 46 per cent. by heating 1½ hours at 1000 deg. C.

Lead-Wool.[4]—This material is soft lead shredded to the thickness of heavy thread and collected into bundles of convenient size and length. It is used instead of molten lead for calking hub and spigot-pipe joints. For this purpose a bundle of strands is somewhat twisted and calked into the joint against a back of hemp or tarred yarn with dull chisel-pointed tools. It is stated that only 75 per cent. as much lead-wool is required as molten lead and that the joint is stronger. Tests made showed that in a 3-in. pipe the joint began to give way at an internal hydrostatic pressure of 3450 lb. per sq.in.

Lead-Alloys.—C. H. Mathewson[5] investigated the alloys of sodium with tin, lead, aluminum, magnesium, zinc, cadmium, bismuth and antimony.

[1] *Metallurgie*, 1906, III, 406.
[2] *Comptes Rendus*, 1895, CXX, 1164.
[3] *Metallurgie*, 1906, III, 441.
[4] *Eng. and Min. Journ.*, 1906, LXXXII, 592.
[5] Doctorate Thesis, Metzger and Wittig, Leipsic, 1906.

In view of the addition of sodium to lead that is being made at present, the constitution of sodium-lead alloys has a practical interest. He found that sodium and lead formed four chemical compounds and four eutectic alloys as shown in the accompanying table.

J. Goodman[1] investigated the strength and elasticity of three lead-antimony alloys with Pb, 90, 87, and 85 per cent., and Sb, 10, 13, and 15 per cent., and one lead-antimony-tin alloy with Pb, 80 per cent., Sb, 15 per cent. and Sn, 5 per cent. He found that in their mechanical properties they differed but little from one another. It may be noted that their compositions approach that of the lead-antimony eutectic with Sb 13 per cent.

Magnolia Metal.—F. C. Stanley[2] analyzed the alloy going by the name of magnolia metal and found Pb, 78.27 per cent., Sb, 17.81 per cent., Sn, 3.88 per cent., Cu, 0.04 per cent. As copper is an accidental impurity, the alloy may be said to be composed of Pb, 80, Sb, 15, and Sn, 5 per cent.

Nonpareil.—This is the name of an anti-friction alloy which has been in the market for some time. An analysis[3] gave Pb, 78.35, Sb, 16.70, Sn, 4.95 per cent., Cu, trace.

Lead-Magnesium.—W. F. Moffet[4] patented an alloy, consisting of Pb, 98.75 and Mg, 1.25 per cent., to be used for antifriction purposes. This alloy is said to have a melting-point higher than those of the lead-antimony and lead-tin-antimony alloys commonly employed for this purpose. An alloy of 96 per cent. lead and 4 per cent. magnesium is brittle, while the alloy of 98 per cent. lead and 2 per cent. magnesium is sufficiently hard to serve as a bearing metal. The alloy is prepared by melting lead under a cover of fluorspar and then adding the necessary amount of magnesium, allowing to cool to the melting-point of magnesium, and pouring into ingots.

Lead-Nickel.—A new alloy has been patented by G. F. Allen[5]. It is made up of 96.5–99.95 per cent. lead and 3.5–0.05 per cent. nickel. It is claimed to be stronger and less corrodible than other lead alloys of a similar nature.

Constitution of Lead Compounds.—K. Friedrich[6] has traced the freezing point curve of lead-arsenic compounds with from zero to 34.4 per cent. arsenic. He finds that the two metals form a eutectic alloy which freezes at 292 deg. C. and contains 2.5–3 per cent. arsenic, and that there are no indications whatever of any chemical compounds. Microphotographs substantiate the thermal evidence. The paper contains interesting ob-

[1] *Engineering*, 1906, LXXXII, 376.
[2] *Metallurgie*, 1906, III, 607.
[3] *Brass World*, 1906, II, 132.
[4] *Metallurgie*, 1906, III, 226.
[5] U. S. Patent No. 834,099, Oct. 23, 1906.
[6] *Metallurgie*, 1906, III, 41.

servations in regard to the eliquation of lead from the alloys richer in lead than the eutectic. The freezing-point curve furnishes a nice explanation of the fact that in softening base bullion in the reverberatory furnace, 75 per cent. of the arsenic remains with the liquated lead, while only 25 per cent. goes into the softening dross.

K. Weidmann[1] traced the freezing-point curve of iron sulphide and lead sulphide mixtures. He found that they form a eutectic (of the composition FeS, 25.8 per cent., PbS, 74.2 per cent.) with a freezing-point at 782 deg. C. and show no indication of a chemical compound. Freezing-points of the eutectic were clearly noticeable within 5.1 and 90.2 per cent. PbS, thus giving the limits of the eutectic line. Seven microphotographs accompany the paper. This paper, as well as that by Hofman, Caypless and Harrington[2] on iron sulphide and copper sulphide mixtures, seems to indicate that the constitution of matte is that of a ternary eutectic.

Lead Ores.

A. Hofmann[3] has investigated the distribution of silver, antimony and tin in the galena of the Pribram silver-lead mines. It has always been held that the silver content of the galena increased with the depth of occurrence. The present investigation shows that such a regularity does not exist. Thus, from the 13th (1010-ft.) to the 32d (3600-ft.) level the galena contains Pb, 81.5–83.7 per cent., Ag, 0.510–0.578 per cent. and Sb, 0.41–0.86 per cent., without showing any regularity in the changes. Tin begins to appear in the 17th (1400-ft.) level and shows a range of 0.020 to 0.200 per cent.; it occurs as a sulphide (stannite, plumbostannite, kylindrite).

Purchasing Lead Ores.—E. A. De La Vergne and F. Guiterman discussed, October, 1906, before the meeting of the American Mining Congress at Denver, Colo., the mutual relations and grievances of the smelting trust and the ore producers from the standpoint of the ore producer and the smelter respectively. De La Vergne complained that owing to the absence of competition in Colorado the smelting trust has laid down arbitrary and unjust rules in valuing ores; Guiterman on the other hand maintained that, since the formation of the American Smelting and Refining Company, the charges for treatment had been reduced and with them the margin of profit, that the present rules for valuing ores were the outcome of careful investigations proving that with the former competition practice the smelter was paying for more than he received, that the smelting charges had to vary with the desirability of the ore for making up blast-furnace charges, and that the smelter trust had made an honest effort to identify its interest with that of the producers.

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[1]*Metallurgie*, 1906, III, 660.
[2]*Trans.*, A. I. M. E., April, 1907.
[3]*Oest. Zeit. f. B. u. H.*, 1906, LIV 129.

Smelting Rates in Nevada[1].—The smelting rate for 1906 on silver ore is $5 a ton on $14 ore or under; on ore between $14 and $20 the charge is $6; on ore between $20 and $25 it is $6.50; with ore of higher grade the rate increases. For ore carrying Cu 0.2–5 per cent. the allowance is $2.25 per unit; with Cu 5–10 per cent. it is $1.75–$2.50; with Cu over 10 per cent. it is $2.75.

Smelting Rates in British Columbia.—W. R. Ingalls states[2] that there is an abundance of lead ores available in British Columbia in comparison with silicious ores; hence the former have to pay a high smelting charge. Since February, 1906, the rates have been as follows: Galena ore—Pay for 95 per cent. of the silver at New York quotation, and for 90 per cent. of the lead, by fire assay, at the London quotation for soft Spanish, less 1c. per lb.; deduct $12 per 2000 lb. of ore for freight and treatment, plus 50c. per unit for zinc in excess of 10 per cent. The freight and treatment charge obtains on ore containing 50 per cent. and upward of lead. Below that tenor there is a deduction of 20c. per unit of lead. Lead is not paid for unless it be in excess of 5 per cent. Silicious Ore.—On silicious, or dry, ores, the freight and treatment rate is $6.50–$10.00 per ton, depending upon the analysis of the ore and its situation; terms for silver, zinc penalty, etc., being as above. Iron Ore.—Magnetite is paid for at the rate of 5c. per unit of excess of iron and lime over silica, together with allowance for silver at the regular rate, less cost of freight to smelter, and no treatment charge.

Smelting Practice.

Works Accounts.—P. H. Argall has written a valuable paper[3] on metallurgical accounts in which he chooses as example for illustration a lead blast-furnace plant which refines its own base bullion and parts the resulting doré silver. The paper can not be intelligently discussed without reproducing the 27 forms accompanying it. It is to be hoped that the paper will be reproduced in pamphlet form and thus made available to all readers.

Pot-Roasting.—L. S. Austin describes[4] a new method of roasting, in operation at several plants of the American Smelting and Refining Company, which he calls pot-roasting. As far as apparatus and mode of working are concerned, pot-roasting is the same as lime-roasting with the lime left out. The word "pot-roasting" may be accepted as the generic term for an operation of which lime-roasting forms a special phase.

The apparatus, shown in Fig. 1, consists of a cast-iron kettle A, 8 ft. 6 in. in diameter, with detached hood c, connected by means of movable

[1] *Eng. and Min. Journ.*, 1906, LXXXII, 1079.
[2] Report of Canadian Zinc Commission, 1906, pp. 72–74.
[3] *Min. and Sci. Press*, 1906, XC, 573, 722, 750.
[4] Ibid. 1906, XC, 511.

off-take *d*, with dust-flue *e*. The kettle has a cast-iron plate *f*, with $\frac{3}{8}$-in. holes serving as grate, and a baffle-plate *g*, which distributes the air-blast arriving through the branch-pipe from an air-main. The kettle or pot having been lowered, into the position shown, by means of an overhead traveling pulley, receives a barrow-load of ashes which is distributed over the grate to form a thin layer and thus prevent adhesion of the charge. Calcareous ores free from lead have been used instead of ashes. Then follow 1–2 tons of hot roasted ore which are to ignite the remaining 8–9 tons of the charge. The ore-mixture for roasting is made up to contain Pb, 7–10, SiO_2, 30, S, 20–25 per cent. The roaster is a

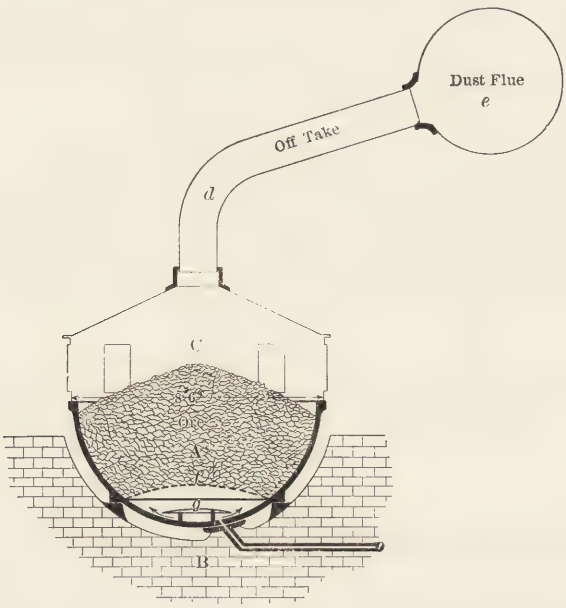

FIG. 1

Godfrey furnace, a single revolving hearth 20 ft. in diameter which (without the use of extraneous fuel) treats 20–25 tons of ore in 24 hours, reducing the sulphur content to 12–14 per cent. The 8–9 tons of charge are slightly moistened as in lime-roasting. Hood and off-take are now adjusted, the blast is started and the rest of the charge is given, either at once or after the preliminary 1–2 tons of ore have become thoroughly ignited, the former being more convenient, the latter probably more efficient. The blast-pressure (1–2 oz. at the start) is raised gradually to 4 oz. When a charge has been blown, the hood is removed, the pot picked up and transferred to the dumping floor at the end of a row of pots, inverted and the sintered cake of blown ore dumped. During the interval between charging and dumping, the overhead pulley

raises blown ore to the mouth of a 24x36 in. Blake crusher. The sulphur content of blown ore ranges from 1.5–2.0 per cent.

According to other reports this process is not quite as satisfactory as one might be led to believe from the above abstract. The time required for blowing a charge is said to be excessive, and the proportion of unsintered ore, to be retreated, rather large.

Lime-Roasting.—W. R. Ingalls published a paper[1] on lime-roasting galena in which he considers mainly the economic advantages of lime-roasting lead ores, with subsequent blast-furnace smelting, over the usual methods of treating non-argentiferous galena in the Mississippi Valley and argentiferous lead ores in the Rocky Mountain region and farther West; some details of the three industrial lime-roasting processes, the Huntington-Heberlein, the Savelsberg and the Carmichael-Bradford, are also given.

In southeastern Missouri the galena concentrate averages 65 per cent. lead. The reverberatory-furnace method of smelting costs $6.50–$7 per ton of ore and yields 90–92 per cent. of the lead. Smelting in the ore-hearth without recovery of fumes comes to $5.75–$6.50 per ton of ore and extracts 87–88 per cent. of the lead; with recovery of fumes the yield in lead is 98 per cent.; and with the improved methods of handling and smelting ores at the new plants of Alton and Collinsville, Ill., the cost has been reduced below the figures given. Roasting in the reverberatory furnace and smelting in the blast furnace costs $6–$7 per ton of ore, of which $2–$2.50 go to roasting; the yield in lead is 90–92 per cent. In the treatment of argentiferous lead ores with roasting in the hand reverberatory furnace at $2–$2.50 per ton of ore and smelting at $2.50 (coke $0.84; labor, power and supplies $1.66), to which general expense of about $0.16 has to be added, the total cost is $3.50, assuming that 0.4 ton is roasted for every ton that is to be smelted. The yield in silver is about 98 per cent. and of lead 95 per cent., both figures based upon the fire-assay.

In the Huntington-Heberlein process, an improved roasting furnace, 26 ft. in diameter, roasts 40 tons of charge per 24 hours with 22.5 per cent. coal and 0.5 man on an eight-hour shift, and reduces the sulphur content from 20–22 to 10–11 per cent. at a cost of 63c. per ton on the basis of two furnaces treating 80 tons. This is comparable to the work of a Brückner cylinder in Colorado roasting at a cost of $1 @ $1.45 a ton of ore and reducing the sulphur content of the charge to 4.5–6.0 per cent. An 8-ton converter treats its charge in about 16 hours at a cost of 60c. per ton. The total cost per ton of charge, treating an ore assaying Pb, 50, Fe, 15, S, 22, SiO_2, 8, other elements 5 per cent., the mixture to form a slag, SiO_2, 30, FeO, 40, CaO, 20, RO, 10 per cent., is estimated

[1] *Trans.* A. I. M. E., 1907, XXXVII, p. 528.

as: crushing one ton @ $0.10, $0.10; mixing one ton @ $0.10, $0.10; roasting one ton @ $0.63, $0.63; delivering 1.1 ton to converter, @ $0.12, $0.13; converting 1.1 ton @ $0.60, $0.66; breaking 0.9 ton @ $0.60, $0.54; total $2.16, or $2.70 per ton of ore. The total estimated cost of lime-roasting is therefore higher than roasting in the hand reverberatory furnace, or perhaps the same after introducing improvements in breaking the desulphurized ore and shortening the time of the blow. The main advantage of the process lies then in the increased smelting power of the blast furnace, in the diminished matte-fall and in the greater direct output of silver and lead.

With the galena ores of Missouri, that are slag-roasted in the reverberatory furnace and smelted in the blast furnace, no increase in smelting power of the blast furnace is to be expected; the saving must come in the increased yield of lead, 98 vs. 92 per cent., and in the diminished matte-fall. With the ore-hearth treatment and recovery of fumes little advantage is to be expected by lime-roasting.

In the Savelsberg process the expense of roasting, 63c. per ton of charge or 79c. per ton of ore, has to be deducted, as here the mixture of raw ore and crushed limestone goes straight to the converter. This makes the Savelsberg method of treatment a serious competitor of the Huntington-Heberlein. The objection that the blow in the Savelsberg process lasts very much longer than in the Huntington-Heberlein has still to be proved.

The low cost of $1.64 for lime-roasting a ton of galena concentrate by the Carmichael-Bradford process, proclaimed by the company controlling the patents, must be attributed to the character of Broken Hill ores and to a different basis of estimating.

Doeltz' paper "Experiments on the Metallurgical Behavior of a Mixture of Lead Sulphide and Calcium Sulphate," which forms the basis of the Carmichael-Bradford process of lime-roasting and has been given in abstracts in last year's issue of THE MINERAL INDUSTRY (Vol. XIV, p. 403), has been translated and published in the *Engineering and Mining Journal*, 1906, LXXXI, 175. T. Huntington and F. Heberlein have written a general article[1] on their process which is disappointing, as it gives little information, a policy consistently followed by these patentees and their representatives. This paper, as well as that of Doeltz referred to above, is published in full in Ingalls' "Lead Smelting and Refining."

A. W. Dyer[2] gives a brief outline of the operations in the Hall smelter at Nelson, B. C., which has a daily capacity of about 200 tons of ore. The ore is crushed and sampled in a Vezin plant capable of handling 20 tons ore per hour. It is broken by a crusher on the main floor, raised by

[1]*Eng. and Min. Journ.*, 1906, LXXXI, 1005; *Oest. Zeit. f. B. u. H.*, 1906, LIV, 631.
[2]*Can. Min. Rev.* 1906, XXVII, 110.

a bucket-elevator to the top of the building, fed into the No. 1 Vezin sampler, discarding four-fifths of the ore; the remaining one-fifth goes through a mixer and feeder into a pair of Gates rolls crushing down to 1.5 in. size, is elevated again to the top of the building and dropped into the No. 2 Vezin sampler, retaining 0.2 as sample to be reduced to 0.5 in. size by rolls. A third elevator raises the reduced sample to the No. 3 Vezin sampler, when the reserved portion (1/125 of the original ore) passes through a third set of rolls crushing it to 0.25 in. size. This finishes the mechanical sampling; the further reduction is effected by hand.

In order to be of the right size for the Huntington-Heberlein process, by means of which the lime-roasting is done, the ore has to be reduced in size to pass a 6-mesh screen. The crushing part of the sampling mill is used for this purpose, one impact-screen being interposed before the Vezin sampler No. 1 to reject any over-size, and another after the last set of rolls to remove particles larger than 6-mesh. The reduced ore passes into a feeder which distributes it evenly in bedding bins. From these it is removed in vertical sections into buckets to be hitched to an endless-chain conveyor delivering into a second series of bins. When required for treatment the ore is fed upon a belt delivering into a mechanical mixer, which discharges into the supply-bins for the Huntington-Heberlein preliminary roaster and converter. The circular roasting furnace is 26 ft. in diameter and has a capacity of 55 tons ore in 24 hours. The roasted ore is moistened, raised by means of an elevator to above the Huntington-Heberlein converters and fed into them. A converter has a capacity of 12–15 tons ore, which it treats in 12 hours. Raw ore is often mixed in with the ore rough-roasted in the mechanical furnace. The converter-charge, when blown, is dumped upon the floor below and broken to convenient size by a drop-weight. The blast-furnace department is of the usual pattern except that the Harris fore-hearth, discussed later in this review, is in use. By using the Huntington-Heberlein lime-roasting process, the treatment charge for ore has been reduced from $15.00 to $12.00 per ton.

This report may be supplemented by a few notes taken from a paper by H. Harris.[1] The moistening of the roasted ore is accomplished by letting it drop against a spray of water during its passage to the bin. There are six hemispherical converters 9 ft. in diameter; they stand 17 ft. above the dumping level, and are supplied with air by a No. 7 Connersville blower. A fan 7 ft. in diameter draws the gases off from the hoods of the converters; the time of blowing is 8–12 hours.

A brief editorial[2] furnishes the text for $6\frac{3}{4}$x$4\frac{1}{4}$ in. photographs illustrating the operations of the Savelsberg process: the blowing of a charge

[1] *Brit. Col. Min. Record*, through *Electrochem. and Met. Ind.*, 1906, IV, 410.
[2] *Eng. and Min. Journ.*, 1906, LXXXI, 1136.

with converter under stationary hood; the carriage with converter removed from underneath the hood, the converter having been tilted to show the blown charge; the same with blown charge dropped onto an upright bar which has caused the spherical cake to be more or less broken up.

An editorial[1] reviews the decision of the German patent office which allows the Carmichael-Bradford patent against the appeal of the representatives of the Huntington-Heberlein process, in that it holds that the preliminary roasting is a fundamental part of the Huntington-Heberlein while Carmichael-Bradford dispense with it, use dehydrated gypsum and blow direct. This is of importance in view of the fact that the Carmichael-Bradford process is to be introduced at the works of the United States Smelting Company near Salt Lake City, Utah, while the American Smelting and Refining Company owns the Huntington-Heberlein patents for the United States. In alluding to the chemical reactions taking place in the Carmichael-Bradford process, the editorial expresses the opinion that the desulphurization of both gypsum and galena is due to some chemical reaction between the two. Exception is had to this by R. Anstee[2] who leans toward the probable fact that calcium sulphate is decomposed directly; whether by heat alone or by fluxing is not known.

Smelting in British Columbia.—In the "Report of the Commission Appointed to Investigate the Zinc Resources of British Columbia and the Conditions Affecting their Exploitation,"[3] W. R. Ingalls has chapters on "Lead Smelting in British Columbia" (pp. 64–72) and on the "Smelting Rates on Silver-Lead Ore" (pp. 72–74). In 1905 three smelting plants of the Province were in operation, while the one at Pilot bay, on Kootenay lake, was out of blast. The Canadian Smelting Works at Trail, recently transferred to the Consolidated Mining and Smelting Company of Canada, is the most important. The ores are sampled in a Vezin sampling mill. Until 1905 high-grade galena was smelted raw, and sulphide ores and matte were roasted in hand reverberatory furnaces (15x68 ft.) and in Brückner cylinders. In November, 1905, a Huntington-Heberlein plant was installed to treat the high-grade galena ores. It consists of two Heberlein roasting furnaces, 26 ft. in diameter, and six converters, each of nine tons capacity. The smelting is carried on in two blast furnaces, 44x144 in. at the tuyeres, which have closed tops. They put through 125–150 tons charge in 24 hours, or 2.84–3.40 tons per sq. ft. of hearth area; the ore forms 73 per cent. of the charge, which contains 32 per cent. lead; the slags assay about 10 per cent. ZnO. The electrolytic refining plant working the Betts process has been discussed last year[4].

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[1]*Eng. and Min. Journ.*, 1906, LXXXIII, 883.
[2]Ibid. 1906, LXXXII, 984.
[3]Department of the Interior, Ottawa, 1906, pp. 393.
[4]*The Mineral Industry*, 1905, XIV, 421.

The Hall smelting works, at Nelson, are smaller than those at Trail. The Huntington-Heberlein process carried on here is described under "Lime-Roasting." There are two blast furnaces (42x96 and 44x144 in. at the tuyeres) treating in 24 hours 60 and 110 tons charge respectively, of which ore forms about two-thirds.

The Sullivan Smelting Works at Marysville, East Kootenay, were completed in 1905. The ore is brought from the Sullivan mine by a wire tramway to the railroad, emptied into hopper-bottom cars, run to Marysville and discharged into self-emptying bins; coke, coal and limestone are also stored here. The ore is drawn from the bins upon a belt conveyor which takes it to the crushing and mechanical sampling mill, where it is reduced to pass a 4-mesh screen (0.18-in. holes). Two men operate the mill, which handles 100 tons in 10 hours. From the storage bins of the mill the ore is shoveled upon a belt conveyor, transported to the mixing floor and then to the hoppers of the roaster of the Huntington-Heberlein plant. The charge is usually made up of 100 parts sulphide ore (Pb, 27, Zn, 12, Fe, 18 per cent.) 10 parts silicious ore (SiO_2, 85 per cent.) and seven parts limestone (SiO_2, 3 per cent., no MgO). There are two Heberlein roasting furnaces, 26 ft. in diameter; a furnace has one fire-box with open ash-pit; directly opposite to it is the frame with its fixed rabbles; on either side of the central ore-feed is a pipe to take off the gases. The hearth, making 4 r.p.m., is of steel plate supported by I-beams; it is covered with a working bottom of coke. It takes 1.5 h.p. to run a furnace. The ore, when discharged, is a dull red; the bulk of it is sprinkled with water and collected in two hoppers, the rest drops into a third hopper without being moistened and serves to kindle the charge in the converter. A furnace treats 45 tons charge per day and consumes 400 lb. coal per ton; two furnaces are attended by one man on an eight-hour shift; the ore fed assays 25 per cent. sulphur; the roasted product, 10–12 per cent. sulphur.

The roasted and moistened ore is trammed to a platform elevator and discharged into the hoppers of the six converters arranged in two parallel rows of three each with operating platform between them. The converters are supported by trunnions about 12 ft. above the ground and rotated by a worm and handwheel. They are covered with conical hoods provided with slots for charging and tamping. A converter is 9 ft. in diameter, about 4 ft. deep, has a 6-in. blast-pipe, a 72-in. grate (0.75 in. thick with 0.75 in. holes, 2 in. between centers); it takes a charge of 8–12 tons and blows it in 12–18, average 16, hours. Wood, formerly used in starting, has been replaced by sawdust; this is followed by hot roasted ore spread to a depth of 4–6 in.; the blast is started with a pressure of 2 oz., moistened charge is introduced to fill one-third of the space between the grating and the top, and tamped down at the sides

where the blast has a tendency to break through; any blow-holes that form are poked down; after about an hour more charge is given so as to fill the pot two-thirds, and after two or three hours the rest is added, filling the converter. The pressure of the blast is gradually increased. reaching 12–16 oz. at the end of the blow; the pot is revolved and the blown cake dropped upon an inverted slag-pot; it retains 3–5 per cent. sulphur equally divided between sulphide and sulphate. The rough-broken ore is sprayed with water and further reduced in size by hand, shoveled into cars and conveyed up an incline to the feed-floor of the blast furnace.

There are two blast furnaces, 17 ft. high and 42x100 in. and 42x128 in. respectively at the tuyeres, of which there are seven to a side. Two Connersville blowers, each of 68 cu.ft. capacity per revolution, furnish a blast-pressure of 18 oz. A furnace puts through in 24 hours 120 tons of ore-charge and 30 tons slag, or 3.2 tons material per sq.ft. of hearth area, with 11 per cent. coke. In November, 1905, the sulphide ore treated averaged SiO_2, 7, Fe, 18, Zn, 12, CaO, 3, Pb, 27 per cent., Ag, 11 oz.; the slag was calculated to give SiO_2, 20, Fe, 26, CaO, 10, MgO, 4, Al_2O_3, 4, Zn, 15 per cent.; it assayed 1 per cent. lead and less than 1 oz. silver. No matte is produced, but a considerable amount of drossy material, consisting mainly of lead sulphide, which fills the crucible and chokes up the well. The smoke from the furnaces passes into a concrete dust chamber ending in a steel stack 8 ft. in diameter and 150 ft. high. The power of the plant, 150 h.p., is developed by two 6-ft. Pelton water-wheels; an extra wheel is held in reserve; the head of the water-power is 175 ft. There is an auxiliary steam plant for the blowers.

Lead Smelting at Freiberg, Saxony.—Kochinke publishes[1] a general plan of the Halsbrücken smelting and refining works near Freiberg, Saxony, which supplements a previous paper[2] giving the plan of the Mulden smelting and refining works and a detailed tree of the complex processes carried on in the two plants which treat the leading non-ferrous metals.

Lead Smelting in Austria.—The report[3] of the commission appointed by the Austrian government to investigate the causes of lead-poisoning in the smelting of lead ores and the manufacture of lead products, and the means of overcoming them, has been issued in four parts, embodying descriptions of plants and testimony of experts called in to suggest improvements. From this report the leading data relating to smelting and refining are given below.

The Pribram silver-lead works, Bohemia, produced in 1903, 3014 tons

[1] *Jahrbuch für das Berg-und Hüttenwesen im Königreiche Sachsen*, 1906, A, p. 158.
[2] Ibid., 1902, A, p. 96. *The Mineral Industry*, 1902, II, 438.
[3] "Bleivergiftungen in Hüttenmännischen und Gewerblichen Betrieben, Ursachenund Bekämpfung," five parts, Hölder, Vienna, 1905-7.

lead, 1015 tons litharge and 1,373,152 oz. silver, and employed 499 men. The average analysis of the ores worked in 1902 gave PbS, 22.5; Ag_2S, 0.2; CuS, 0.13; As_2S_3, 0.5; Sb_2S_3, 1.46; Zn, 6.75; FeS_2, 6.33; FeAsS, 0.83; (Fe, Mn, Ca, Mg) CO_3 over 20; Al_2O_3, 4.1; SiO_2, 33.4 per cent. The ore is roasted in 13 single-hearth hand reverberatory furnaces 8.2 ft. wide; 10 of them are 70.5 ft. long and three 73.5 ft. There are nine working doors on a side. The fire-bridge is air-cooled and the air thus warmed is turned into the hearth near the third door. The waste gases return underneath the hearth before they enter the dust chambers. In order to prevent dust and gases from entering the furnace-room while charging ore, the charging-door in the roof is automatically closed[1]. The slag-roasted ore is drawn through an opening in the hearth into a car, the vaulted chamber being closed by a door; thus the fumes are prevented from entering the building. Matte and other sulphurous intermediary products are roasted in heaps and stalls[2].

The roasted ore is smelted with intermediary products from the refinery in eight circular blast furnaces of which six have eight, one twelve and one five tuyeres. Usually five furnaces are in blast. All are fed from the top through a Pfort curtain, the lead is drawn off through a siphon-tap, hoods are provided to carry by natural draft the fumes arising from the lead-well and the slag-tap. In 1902 there were smelted 38,025 tons of charge, 26,051 tons of slag and 13,238 tons of intermediary products; there were produced 11,371 tons of base bullion (Pb, 98; Ag, 0.397 per cent.), 2185 tons of matte (Pb, 14; Ag, 0.118 per cent.), 14 tons of nickeliferous speiss, 186 tons dross, 32 tons brasque and 345 tons flue-dust. Slag with 2 per cent. lead and less goes to the dump. The base bullion is liquated in reverberatory furnaces taking charges of 2204 lb. before it is desilverized in the Steam-Pattinson plant. There are formed 4–5 per cent. dross (Pb, 70; Cu, 4 per cent.) which removes 80–90 per cent. of the copper content. The Pattinson plant with three crystallizing kettles furnishes desilverized lead with traces of silver and cupelling lead with 0.8–1.0 per cent. silver. The desilverized lead is freed from tin and antimony by heating for 24 hours in two reverberatory furnaces of 26.4 tons capacity. The softened lead is tapped, allowed to solidify, melted down in cast-iron kettles of 4.4 tons capacity, and then molded by hand. Cupellation is carried on in six German furnaces of 26.4 tons capacity. The resulting silver is fined in two small reverberatory furnaces. Antimony-skimmings are smelted for hard-lead with Sb, 20 per cent. Cakes of litharge low in silver, after falling to pieces, are charged into housed, revolving screens, furnishing flaky litharge for the market.

The condensation plant consists of seven dust-chambers 114.8 ft.

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[1] *The Mineral Industry*, 1905, XIV, 402.
[2] Hofman, "Metallurgy of Lead," p. 361.

long by 37.4 ft. wide, each having staggered partition walls, forming four minor chambers 110.5 ft. long, 6.5 ft. wide and 11.5 ft. high. The gases from the different departments pass into a main flue with a suction fan (there is a reserve fan) drawing the gases and forcing them through the dust chambers into the stack. In 1903 there were collected from the roasting furnaces 129 tons of dust (Pb, 30; As, 2; S, 11 per cent.); from the blast furnaces 277 tons (Pb, 47.7; S, 5.1 per cent.); from the Steam-Pattinson plant 76 tons; from the cupelling furnaces 53 tons (Pb, 61; S, 4.5 per cent.); from the last dust chambers, 373 tons (Pb, 42; S, 8.8 per cent.).

While during the preceding 10 years 2.9 per cent. of the men suffered from lead-poisoning, in 1903 this figure was reduced to 1.3 per cent.

The Lead Works of Raibl, Carinthia, produced in 1903 only 505 tons of lead. The galena concentrate contains Pb, 65 and Zn, 7 per cent. There are five Carinthian reverberatory smelting furnaces with hearths 11.5 ft. long and 4.9 ft. wide. They are fired with lignite, forced draft being employed. The charge weighs 925 lb. The stiffening ingredient is heavy spar. The waste slag assays 2 per cent. lead. The yield in lead is 94.8 per cent. One man tends a furnace in a 12-hour shift.

The lead works of Gailitz, Carinthia, produced 2175 tons of lead in 1903. The ore assays 75 per cent. lead. The plant has six Rossie ore-hearths 1.96 ft. square and 0.98 ft. deep, built around a central chimney. Three are worked at a time. There are also six hand reverberatory roasting furnaces for roasting and sintering slimes and flue-dust previous to working in the ore-hearth; also a blast furnace, resembling the one at Pribram, used on and off to smelt gray slag and roasted slimes. The fumes from the hearth furnaces are drawn by means of a fan through wet condensation chambers before they pass off into the stack. From the furnaces they enter a cast-iron pipe, 3.28 ft. in diameter, pass up and down through two brick chambers, 3.9 ft. square and 16.4 ft. high, having water-sprays, travel through four upright U-shaped clay pipes sprayed internally, traverse another couple of brick chambers similar to the first, pass through a fan into a flue 6.56 ft. square and 5.25 ft. long, and leave by a chimney 118 ft. high. The acid condensation water flows slowly through settling tanks, is then neutralized with lime and goes to waste. Filtering gases from the hearth furnaces showed that 1 cu.m. contained 6.33 grams solid matter with 3.1 grams lead, while the gases at the bottom of the stack gave 1.86 grams solid matter with 1.23 grams lead. These data show that it is easy to settle dust, but difficult to condense fumes. The direct yield in lead is 86.3 per cent. The gray slag, 10 per cent. of the weight of the ore, contains 25 per cent. lead, equaling 3.4 per cent. of the lead in the ore; the remaining 10.3 per cent. is volatilized, but about one-half is recovered by condensation. Five men tend a hearth in a

12-hour shift. Of the 49 men at work in 1903, 18 or 36.7 per cent. were affected by lead fumes.

The lead works of Scheriau, Carinthia, produced 4584 tons of lead in 1903. The galena is partly coarse, partly a fine concentrate. The ore-hearth smelting is carried on in two furnace-rooms, each of which contains three ore-hearths built around a central chimney; the hearths are separated from one another by brick walls. Fine concentrates and flue-dust are slag-roasted in a hand reverberatory furnace, 98.4 ft. long; gray slag, slag-roasted fines and other lead-bearing products are smelted in an eight-tuyere blast-furnace. Wet condensation is used for collecting fumes. Induced draft is furnished by a fan which, making 720 r.p.m., takes 35.3 cu.ft. of gas per revolution. Three samples of air taken near the face of the workman at the ore-hearth gave in 250 liters 0.2–0.3 mg. of lead oxide. The gases from the two sets of ore-hearths zigzag through five and seven, respectively, condensation chambers, 5.9 ft. square and 18.7 ft. high. Wires, 7 mm. in diameter, suspended in the chambers were effective in collecting dust, but soon interfered with the draft. The gases from the two ore-hearth pipes and the flue from the roasting furnace enter a sheet-lead flue, 8.5 ft. square and 180.4 ft. long, and then pass into the wet-condensation plant which consists of 52 chambers, 15.7 ft. square and 17.7 ft. high, internally sprayed from the top; they zigzag through these, are cooled down to 20 deg. C., their content of sulphurous acid is reduced from 13.8 to 2.6 gram per cu.m. Between the suction fan and the stack (98.4 ft. high) is a condensation chamber, 1484 sq.ft. and 10.5 ft. high, with seven partitions, which has saved much uncondensed lead-bearing material. The acid water from the condensation plant flows through clarifying tanks and then goes to waste without having been neutralized. Of the 61 men employed in 1903, 18, or 30 per cent.. were affected by lead fumes.

The lead works of Kreuth, Carinthia, are in operation only six months in the year; in 1903 they produced with two ore-hearths 666 tons lead; the fumes are not condensed.

The lead works of Windisch-Bleiberg, Carinthia, use the Carinthian reverberatory smelting furnace. They produced 342 tons lead in 1903 from ores containing 65 per cent. lead and 7 per cent. silica; the charges are of the old standard weight, viz, 440 ℔. It is of interest to note that the official limit of 5 per cent. silica in the ore to be suited for reverberatory smelting work is here exceeded.

The lead works of Eisenkappel, Carinthia, produce annually 143 tons of lead from ores assaying 76.7 per cent. There are two Carinthian reverberatory smelting furnaces used alternately; they take 660-℔. charges.

The silver-lead works of Littai, Carinthia, produced 2231 tons of lead in 1903 (which amount includes 16 tons hard lead and 77 tons lead charged

with 3 per cent. tin) and 11,114 oz. silver. They are custom works and obtain their ores from Carinthia, Galicia, Bohemia, Bosnia, Greece and Tunis. Up to 1902 the ores were smelted in the Silesian reverberatory furnace for base bullion and residue with 30–50 per cent. lead to be smelted in a circular blast furnace. Since then the Huntington-Heberlein process has replaced the reverberatory work. The ore is mixed with 6–15 per cent. lime, the amount varying with the percentage of sulphur, and is roasted in a hand reverberatory furnace at a temperature not exceeding 700 deg. C., being turned over and moved toward the bridge every 45 minutes. Every four hours roasted ore is drawn off into a converter brought to the discharging door, rammed down and then blown with a blast pressure of 2–3 cm. mercury. The blown ore is smelted in a blast furnace having a siphon-tap. The base bullion is desilverized by the Parkes process. The desilverized lead is refined with steam. The zinc-crusts, liquated in a kettle, are distilled in crucible-shaped retorts; the retort-bullion, containing 2–3 per cent. silver, is cupelled. Of the men employed, 30 per cent. were affected by lead fumes in 1903.

A description of the zinc works of Cilli, Syria, concludes the first part of the report, in which more stress is laid upon the local arrangements to carry off fumes and make them harmless than upon metallurgical details. Thus there are given many general plans of work and pictures of apparatus, but hardly any drawings of furnaces. For the depositions of experts in regard to the desirable sanitary improvements the reader must be referred to the original.

Lead Smelting on the West Coast of Tasmania.—The lead smelting industry of Tasmania, according to F. Kapp[1], is confined to the plant of the Tasmania Smelting Company, near Zeehan, which began operations in 1898. The ores are sulphides rich in zinc; in 1905 there were treated by this plant 39,863 tons of lead ore, averaging 22.6 per cent. lead and 34.4 oz. Ag per ton, and 22,310 tons of metal-bearing fluxes, making a total of 62,173 tons of material. The ores are sampled by a Bridgman mechanical sampler; they then pass into the bins (5000 tons capacity) of the Huntington-Heberlein lime-roasting plant which contains 12 converters. These deliver their gases into a roomy dust chamber ending in a stack 8 ft. inside diameter and 125 ft. 10 in. high. The blast furnace materials are assembled in bins holding 4000 tons raw and lime-roasted ore, 2000 tons of flux and 1000 tons of coke. There are three water-jacket blast furnaces 3 ft. 6 in. by 10 ft. at tuyeres and 21 ft. high, rated at 80 tons of ore; two furnaces are usually in blast. The blast furnaces gases travel through a dust chamber 200 ft. long before they enter the stack, which is 125 ft. high. The dump is 60 ft. high. The base bullion is shipped to Europe. The power-plant consists of four multi-

[1]Annual Report, Zeehan School of Mines and Metallurgy, 1905, pp. 89–95.

tubular boilers of 125 h.p. each, three being in use at a time; two 125 h.p. Reynolds-Corliss engines for driving two No. 7½ Root blowers (guaranteed to furnish blast at 5 lb. pressure) and for providing power for the shops; one 75 h.p. engine of same make for the sampling plant; and one 50 h.p. engine for the crushing and drying plant. Two Worthington pumps served by two special boilers furnish the necessary water. A haulage line connects the different levels of the plant.

The Blast Furnace.—I. John[1] remodeled the mechanical charging apparatus at the smelting plant of the Cia. Metalurgica de Torreon, at Torreon, Mexico. The eight furnaces had brick hoods and steel down-takes. The original plan was to hoist the charge on an incline (43.5 per cent. grade) in buckets of 55 cu.ft. capacity to the feed-floor, to release the grip of a bucket at the end of a furnace, drop the contents into a car, raise the feed-door, push in the car and release the drop-bottom. For the new plan the tops of the furnaces have been raised 6 ft. and the down-takes placed beneath the feed-floor. The charges are dumped into an incline-car having the same length of the furnace. It weighs four tons, takes five tons of charge and, when hauled up the incline, drops the charge into a feed-car, returns to the pit to be reloaded. Loading takes three minutes; ascent, discharge and descent take two minutes. The feed-car is run by means of a friction drum and tail-rope over the tops of the furnaces and discharged; the speed is sufficiently great for the car to discharge its contents and return to the receiving station in advance of the incline-car. The present arrangement has been in operation for two years and is giving complete satisfaction.

An editorial[2] gives an illustrated description of the double-discharge ore- and coke-bins of the smelting works of the Old Dominion Mining and Smelting Company, Globe, Ariz., in which each discharge-gate has its own suspended weighing-scales. The electrically operated charge-cars are run underneath the gates and receive weighed amounts of ore, flux and fuel.

H. Harris[3] constructed the fore-hearth shown in Figs. 2 and 3, which is very similar to the Orford fore-hearth of Howe. In the figures, C is the tapping jacket with tap-hole F; MU is an L-shaped cast-iron box with slag-spout S, matte-spout P, opening N in a water-cooled partition, and tap-hole W. The fore-hearth resting on a carriage is backed up against the tap-hole, the joint being made tight by tamping fire-clay in the groove L. Matte and slag, as well as any overflowing lead, are discharged continuously into the division M, the slag rises to the top and overflows through spout O, while the heavier matte remaining near the bottom runs off through spout P. Any lead failing to be removed

[1] *Eng. and Min. Journ.*, 1906, LXXXI, 126.
[2] Ibid. 1906, LXXXI, 1043; *Mining Reporter*, 1906, LIII, 337.
[3] Ibid. 1906, LXXXI, 178; *Can. Min. Rev.*, 1906, XXVI, 141; *Journ. Can. Min. Inst.*, March, 1906.

by the lead-well and collecting in the fore-hearth is tapped at intervals through *W*. The main advantage of this fore-hearth over the Orford

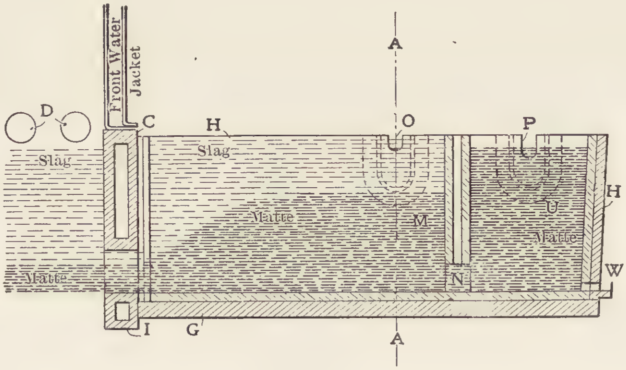

FIG. 2

type is that the matte does not come into contact with air and escapes the slight oxidation which otherwise can not be avoided. The result is a better separation and a cleaner slag.

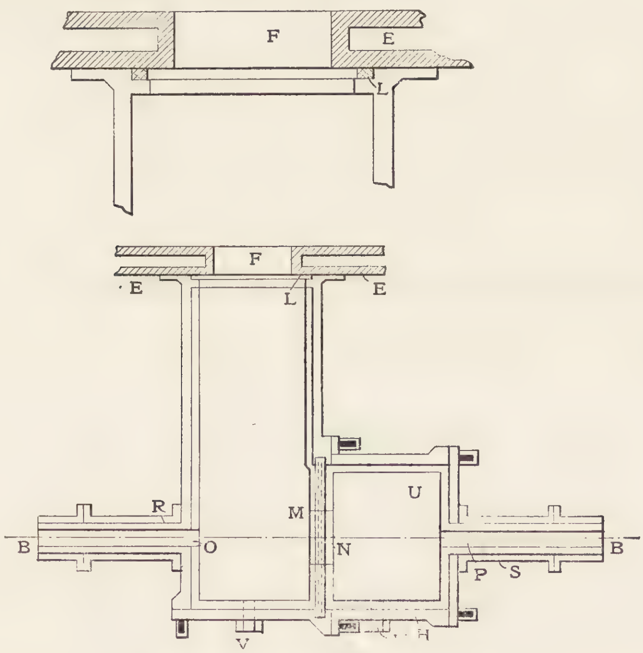

FIG. 3.—SIPHON SPOUT AND MATTE TRAP.

C. M. Allen[1] patented a fore-hearth for separating matte and slag.

[1]U. S. patent No. 810,364, January 16, 1906.

The leading point is that here also the matte is kept from contact with air to produce a better separation from the slag.

C. M. Hawkes patented[1] a method of fastening the stay-bolts in steel water-jackets; they are welded to the inner sheet of the jacket, pass through the outer and are riveted to this.

The Pich process[2] for brazing cast-iron would seem to be well adapted for closing up a crack in a cast-iron water-jacket. It consists in preparing a finely divided mixture of copper oxide and calcined borax, moistening it with water, coating the crack with the mixture, heating the joint and applying ordinary brazing solder. The carbon of the cast-iron reduces the copper oxide to metallic copper, part of which becomes alloyed with the iron and makes the whole receptive for the solder. It should be added that it is advisable to bore holes[3], to be closed by driving in copper pins, at either end of the crack in order to prevent this from extending.

H. W. Hixon gives[4] an illustrated description of the slag-granulating device which he introduced at the smelting works of the Mond Nickel Company, Victoria Mines, Ont. It has been working satisfactorily since it was put into operation five years ago.

The Bennet-Jones slag-car,[5] referred to previously in these reviews,[6] is used at the works of the Tacoma Smelting Company. The standard size weighs 1600 ℔. and has a capacity of about 1600 ℔. slag; the hight from top of rail to top of bowl is 37 in., the extreme width of the bowl is 3 ft. and the extreme length 4 ft. In discharging, a solid cake of slag is thrown 2 ft. beyond the ties; by using rails as skids it can be slid a distance of 18–20 ft.; liquid slag is delivered 18–20 in. from the edge of the track.

The Colorado Iron Works,[7] Denver, Colo., has brought out a single-bowl slag-truck. The bowl is designed for lead blast-furnace use; its capacity is 8 cu. ft. It has a special locking device to hold it in an upright position and is dumped by means of a hand lever.

The same company has put on the market another single-bowl slag-truck.[8] The bowl is conical and has a capacity of 35.1 cu. ft. The hight from top of rail to top of bowl is 4 ft. 7 in. The bowl is supported by a steel band riveted to trunnions, has a screw-dump and a worm-releasing attachment, so that the bowl when released may resume its upright position.

Another recent slag-car is that built by the Power and Mining Machin-

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[1] U. S. patent No. 815,013, March 13, 1906.
[2] *Eng. and Min. Journ.*, 1906, LXXXI, 228.
[3] Hofman, "Metallurgy of Lead," p. 245.
[4] *Eng. and Min. Journ.*, 1906, LXXXII, 553.
[5] Ibid. 505; *Iron Trade Review*, July 19, 1906.
[6] *The Mineral Industry*, 1904, XIII, 300.
[7] *Mining Reporter*. 1906, LIV 168.
[8] Ibid.1906, LIV,240.

ery Company,[1] Cudahy, Wis., for the Garfield plant of the American Smelting and Refining Company. The car is 15 ft. long and 6.5 ft. high. Car and bowl weigh 27,000 lb. The bowl, which holds 10 tons of slag, is made in five sections, the bottom being a single casting. The sections are bolted together and the sides tied by key-bolts to the steel supporting ring which is riveted to pinion-toothed trunnions rolling on racket-tracks; the rolling is effected by compressed air.

Condensation of Flue-Dust.—Armored concrete for the erection of flues and dust chambers has in recent years not given complete satisfaction on account of its cracking. The result is a tendency to give up entirely this valuable building material. This seems to be going too far, as the several unfortunate experiences may lie not in the principle, but in the detail of construction. For this reason an editorial[2] giving with illustrations the different forms of metal most widely used for the reinforcement of concrete is timely.

A note[3] states that in a dust-settling chamber the velocity of the gases should be reduced to 4 ft. per sec., and the gases remain 15 sec. or more in the chamber to allow the dust to settle.

The filtering of silver-lead fumes[4] which had fallen into disfavor, as it saved undesirable volatile compounds which otherwise would pass off into the air, is being taken up again by the American Smelting and Refining Company. The installation at East Helena, Mont., out of commission for several years, is being employed again; also at the works of the United States Smelting Company, near Salt Lake City, a bag-house is being erected.

K. Friedrich[5] has written a valuable paper on the condensation of flue-dust and fume in which he covers the leading principles, reviews the methods that have so far found application in metallurgical works, and proposes as a new method the use of Siemens regenerators. For these he makes the necessary calculations and furnishes preliminary plans for horizontal as well as vertical chambers to be attached to hand reverberatory roasting furnaces. The construction of such regenerators has to be different from those of Siemens, as their main object is to cool low-temperature sulphurous gases down to atmospheric temperature and to collect flue-dust by exposing to them large surfaces. In concentrating dilute sulphuric acid, a loss in acid by evaporation occurs only when the acid has reached a strength of 60–62 deg. B., and the boiling-point of such acid lies at about 200 deg. C. The hottest part of the condensation chamber may therefore not exceed that temperature, as otherwise, when the regenerator is put on air in the usual way, part of the condensed acid

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[1] *Eng. and Min. Journ.*, 1906, LXXXI, 847; *Iron Age*, 1906, LXXVII, 1389.
[2] *Iron Age*, 1906, LXXVII, 13.
[3] *Eng. and Min. Journ.*, 1906, LXXXII, 1226.
[4] Ibid.
[5] *Metallurgie*, 1906, III, 747, 774, 805.

will be again evaporated; further, air blown through the chamber will whirl up again settled dust. For these reasons the author divides his condensation plant into two parts. In the first, the gases are cooled from 300 to 200 deg. C. It is filled with rhomboidal clay cross-bars, or Freudenberg plates, or their clay tiles, spaced in such a way as to give free passage to the gases (similar to brick hot-blast stoves) and operated in a manner resembling that of Siemens regenerators, thus utilizing the heat stored in the filling material. In the second, the gases are cooled from 200 deg. C. to atmospheric temperature. It is filled with acid-proof brick in a manner similar to that of the hot chamber, and operated on the counter-current recuperative method, the warmed air passing seriatim through a number of chambers off into the open, one of which is always cut out to be cleaned by spraying. In the plans for the three hand reverberatory furnaces, each has its own hot-chamber, while the gases from the hot chambers go into a single cool-chamber. For details the reader must be referred to the original, which is well worth a careful study.

Smelter-Smoke in Utah.—The Federal Court[1] has handed down a decision (Nov. 5) granting an injunction against the American Smelting and Refining Company, the United States Smelting, Refining and Mining Company, the Utah Consolidated Company, and the Bingham Consolidated Company, forbidding them from smelting ores containing over 10 per cent. sulphur. The enforcement of this injunction has been postponed six months by appeal of the smelters. The investigation of sulphurous gases and the rendering them harmless are questions that have occupied European metallurgists for years; whether their methods can be profitably transplanted remains to be seen.

Working of Flue-Dust.—While copper smelters have devised means of working off their flue-dust by smelting in a reverberatory slag-settling furnace, lead smelters have to briquet on account of the ease with which lead oxide combines with silica, and the volatility of lead when exposed to flame. The briquetting of iron ores and the testing of briquets has been reviewed in an extended paper by Wedding,[2] covering the present practice and the German patent records. While dealing with finely divided iron ores, the treatment of the subject contains much that is of value to the lead smelter.

The late H. F. Brown[3] patented a process and furnace by means of which he aimed to reduce finely divided oxide ore by passing it in a shaft furnace through a heated reducing atmosphere to obtain metal in a slag-covered bath, at the same time scorifying the non-reduced part.

New Smelting Process.—A. G. Betts patented a process[4] for smelting

[1] *Eng. and Min. Journ.*, 1906, LXXXII, 885, 1077.
[2] *Stahl und Eisen*, 1906, XXVI, 2, 76.
[3] U. S. patents Nos. 817, 414 and 817,415, April 10, 1906.
[4] U. S. patent No. 821,330, May 22, 1906.

lead ore in which he proposes to fuse galena with siderite or limonite, sodium carbonate (in part replaceable by caustic soda), and coal or coke in a reverberatory or a blast furnace.

Electrolytic Processes.—A. G. Betts proposes[1] an electrolytic process for smelting lead ores which is based upon experiments carried out by himself and W. Valentine. Their first plan was to have a bath of lead in a cast-iron kettle as cathode, fused lead chloride as electrolyte, and graphite bars as anode; to feed galena into the electrolyte which readily dissolves it below a red heat; and to pass a current through the cell which would reduce the lead and cause the sulphur to pass off as a vapor (440 deg. C.) at the anode. The theoretical voltage of decomposition being 0.4 volt, there would be required 1–1.25 volts to operate regularly, which corresponds to 14 h.p. days per ton of lead sulphide; all the lead was to be recovered and the 250 ℔. of sulphur liberated were to cover the cost of power. After working for some time the pots became clogged on account of the reduction of some iron, which formed a scum instead of having a decomposing effect upon lead chloride ($Fe + PbCl_2 = FeCl_2 + Pb$) and being converted into ferrous chloride as was expected.

The first process having proved itself to be a failure, it is proposed to matte the lead ore, probably in an electric furnace, and to electrolyze the matte in a fused bath of sodium chloride. It is estimated that to smelt electrically 100 tons of ore per day there will be necessary 1880 h.p. for which the cost, taking e.g. the Niagara figure of $20 per year, is $1.04 per ton. The additional charges for labor, repairs, fluxes, power, smelting slag are estimated at $1.40, making a grand total of $2.44. With a high cost for electric power, it will probably be advantageous to heat the electrolyzing vat with carbonaceous fuel. The vat is a covered iron tank built into and lined with brick-work, it is provided with a lead well, a tap-hole for matte and an outlet-pipe for sulphur fumes. The anodes are graphite, the cast-iron tank serves as cathode. In operating, the pot containing fused pure sodium chloride (preferable to mixed chlorides) is charged with melted or solid matte, the surface of which acts as cathode; graphite bars form the anode. When the current is started, chlorine is liberated at the anode with the formation of sodium sulphide and liberation of lead. Soon chloride of sulphur forms at the anode and later sulphur alone, as the sodium sulphide combining with ferrous sulphide to a matte accumulates in the bath with a consequent drop of the voltage. An addition at the start of 22.6 ℔. sodium sulphide per ton of matte stops the first liberation of chlorine. The products are: condensed sulphur; matte tapped at intervals to be leached with water to recover sodium sulphide, then smelted with slags in the blast furnace; and metallic lead. It is estimated that with a potential of 2.4 volts there will be an

[1] *Electrochem. and Met. Ind.*, 1906, IV., 169; U. S. patent No. 816,764, April 13, 1906.

efficiency of 90 per cent. and a saving of 96 per cent. of the lead in the matte. The electrolysis of one ton matte will require 23 h.p. days, which at $20 per year cost $1.27; the other costs (labor, repairs, sodium sulphide, sodium chloride, roasting tapped matte) are estimated at $1.71, making a total of $2.98, corresponding to $2.68 per ton of ore (1 ton ore = 0.9 ton matte). Adding the estimated cost of electric smelting of ore at $2.44, makes a grand total of $5.12 from which has to be deducted $2.15 (215 lb. sulphur @ 1c.), leaving a net total of $2.97.

C. P. Townsend patented[1] a process for reducing galena electrolytically in a fused electrolyte consisting of alkali or alkali-earth haloid. The ore floats upon the cathode bath of lead and beneath the electrolyte on account of the great differences in the specific gravities; the anodes dipping into the electrolyte are of carbon or graphite. The rest of the process is similiar to that of Betts.

F. T. Snyder patented[2] an electric smelting process for mixed zinc-lead sulphides to which the reader may be referred.

Hydrometallurgy.

C. B. Jackes[3] proposes to treat lead ores by a wet process, using nitric acid as solvent and following analytical lines to obtain white lead and lead pigments including litharge as chief products, and as by-products iron pigments, niter cake, nitrogen oxides and sulphur.

Lead Refining.

Parkes Process.—A. J. Aubrey[4] in a general paper on the refractory uses of bauxite refers to the use of bauxite brick as a lining for the softening and refining reverberatory furnaces of a zinc desilverizing plant. Here it has been found to resist the corrosive action of the skimmings five or six times as long as does fire-brick.

C. A. L. W. Witter patented[5] a process for separating tin from tin-lead alloys, which consists in subjecting the alloy to an oxidizing fusion when tin will be oxidized before lead. The tin dross consisting of stannous and stannic oxides with more or less lead oxide is drawn off and reduced to an alloy rich in tin. This well-known mode of procedure has been in operation at the Freiberg (Saxony) silver-lead works for many years.[6]

A. Raht[7] patented the idea of adding a small quantity of sal ammoniac to the metal bath after the zinc has been stirred in in order to facilitate the removal of the zinc crust. The result of the addition is a drier and thereby a richer crust than is usually the case. One pound sal ammoniac

[1] U. S. patent No. 815,881, March 20, 1906.
[2] U. S. patent No. 834,644, Oct. 30, 1906.
[3] *Journ.* Can. Min. Inst., 1905, VIII, 244.
[4] *Eng. and Min. Journ.*, 1906, LXXXI, 217.
[5] U. S. patent No. 801,820, Oct. 10, 1905
[6] Hofman, "Metallurgy of Lead," pp. 435, 498.
[7] U S. patent No. 826,114, July 17, 1906.

to 10 tons base bullion is said to be sufficient. It will be remembered that Roessler and Edelmann[1] used this means of freeing zinc crust from oxidized material in their well-known experiments forming the basis of the Roessler-Edelmann modification of the Parkes process.[2]

Electrolytic Refining of Lead.—D. Tommasi[3] has issued a pamphlet, "Electrolytic Processes for the Extraction, Separation and Refining of Metals," in which he discusses his apparatus for the electrolytic refining of lead. This consists of a metallic disk half immersed in the electrolyte (a double acetate of lead and potassium) revolving between two base bullion anodes. The spongy cathode deposit is scraped off mechanically when it has become sufficiently thick, washed, compressed and melted down under charcoal; the wash-water is boiled down to 30 deg. B. and returned to the electrolyte; the anode mud is worked up by some means or other. Detailed calculations aim to show the profits that are expected from the process when applied on a large scale. The process has been repeatedly referred to in these reviews.[4]

A new Betts plant of 2500 tons monthly capacity has been erected by the United States Metals Refining Company, near Chicago. It went into operation in December, 1906. The problem of handling the anode mud has not yet been satisfactorily settled.

A. G. Betts patented[5] a modification of the bus-bar, used in his process for the electrolytic refining of lead, consisting of a trough-shaped cavity containing a number of wells filled with a liquid to make electric connection.

Cupellation.—It is not often that a complete analysis of fine silver is made, as it is usually sufficient to determine the fineness. The following analysis[6] of an average bar of Government silver (assay silver) is of special interest: Ag, 99.929, Cu, 0.056, Pb, 0.003, Au, 0.007, As, 0.001, Sb, 0.002, Fe, 0.001, Zn, trace, Fe, 0.001.

Present Position of Lead Smelting in Germany.

By F. T. Havard.

The débacle in the lead-smelting industry of Germany has been long expected. That it is not already history is due to these principal causes:

(1) The steady rise in the price of metals. Thus stocks have appreciated, custom smelteries have worked in a rising market, and mining and smelting companies like Mechernich, whose properties are almost worked out, have been reluctant to shut down while their metals fetched prices which permitted them to raise low-grade ore.

[1] *B. u. H. Ztg.*, 1890, XLIX, 245.
[2] Hofman, "Metallurgy of Lead," p. 450.
[3] No. 4 Avenue d'Italie, Paris, France.
[4] *The Mineral Industry*, 1896, V, 412; 1897, VI, 459; 1899, VIII, 410.
[5] U. S. patent, No. 827,702, Aug. 17, 1906.
[6] *Brass World*, 1906, II, 389.

(2) The reduction in the costs of working due to the introduction and successful application of the Huntington-Heberlein process. Many older companies took the opportunity while installing this process to remodel or improve their plants, and in all cases the introduction was followed by a reduction of costs, particularly in the roasting and smelting departments.

(3) The extraordinary advance in the price of antimony has caused a rise in the price of hard lead, while payment for antimony in ores has remained normal. The consequent profit made on hard lead has helped all works; particularly, however, those smaller works which make more or less a specialty of the manufacture of alloys.

It cannot, however, be more than a short time before the majority of the smelteries will be obliged to shut down, unless some radical change takes place in the handling of the ore market. Most of the works live on foreign ore, a great part of which is in the form of concentrates from Broken Hill, N. S. W. This foreign ore, amounting annually to about 100,000 tons of lead concentrates and 20,000 tons of mixed silver-lead and dry silver ores, is imported through two or three strongly financed houses, of which the Metallgesellschaft of Frankfurt am Main and Beer, Sondheimer & Co., of the same city, a more recent but extremely versatile and active firm, are the most powerful.

The Metallgesellschaft satisfies the requirements of its own works first, provides ore for the smeltery at Braubach, and then for those works which have installed the Huntington-Heberlein process the patent rights of which belong to this company. Beer, Sondheimer & Co. sell to all smelters, but are especially interested in the zinc and lead works at Overpelt, Belgium.

The returning charges allowed by these firms on ores sold to the independent smelteries are extremely low, amounting in the case of lead concentrates to 38 to 40 shillings c.i.f. Antwerp or Hamburg (equivalent to $7 or $8 f.o.b. works), with full pay for all lead, gold and silver. Should copper be present, full payment of all metal found by electrolytic assay is demanded and the returning charge is raised by only a few shillings. For dry silver and gold ores the competition is almost equally keen, and the margin for profit on these is only a little more than on the concentrates.

It is possible that this deplorable position of the smelters is not as much the result of past competition among the private works as of the suicidal tactics adopted by the directors of some of the royal smelteries, of buying lead concentrates at any price in order to keep their bins full. All efforts to form a union of smelteries to combat the demands of the dealers have been frustrated by the action of the fiscal works. Appended is a list of smelteries with their silver and lead output for the last three years. At the present time Freiberg and Mechernich are actually considering the closing down of their smelteries. The works at Call and those at Stol-

berg (both Rhein-Nassau and Stolberg) are making a plucky fight, and by careful management and nearly complete condensation of metal fume they probably show a profit on the year's run. Call is, it is true, further assisted by its stock of slags and tailings from ancient Roman workings on which it draws industriously. Ems uses its own ore and will live as long as its mines are worked. Braubach is a well managed plant of fair equipment, with strong financial backing, and will probably work prosperously under the patronage of the Metallgesellschaft when its less fortunate rivals are idle.

LEAD AND LITHARGE PRODUCTION OF GERMANY.
(In metric tons).

A. Smelteries of Rhineland-Westphalia.	1903.		1904.		1905.	
	Lead.	Litharge.	Lead.	Litharge.	Lead.	Litharge.
Stolberger Gesellschaft	16,090	109	16,570	100	16,517	101
Rheinisch-Nassauische Gesellschaft	11,908	....	12,101	....	11,650	....
Mechernicher Bergwerksverein	14,654	....	10,459	....	11,250	....
A. Poensgen & Sohne, Hütte zu Call	9,851	....	10,372	....	12,235	....
Gesellschaft des Emser Blei- und Silberwerke	4,259	....	3,787	....	4,258	....
Blei- und Silberhütte Braubach	20,147	....	20,290	....	20,894	....
Total	76,909	109	73,579	100	76,804	101
B. Smelteries in the rest of Germany.						
Victoriahütte bei Niederfischbach	......	....	......	....	......	....
Rothenbacher Hütte im Siegerland	17	265	24	262	24	288
Walther-Croneckhutte bei Rosdzin (Giesche's Erben)	6,719	1,092	8,367	1,134	8,945	1,238
Konigl. Hüttenamt Friedrichshütte bei Tarnowitz	35,005	1,244	31,017	1,025	41,610	649
Oberbergamt Clausthal, Oberharz	10,292	....	10,392	....	8,767	....
" " Unterharz	4,977	....	5,071	....	5,142	....
Konigl. Oberhüttenamt Freiberg	7,364	228	5,452	138	5,146	99
Anhaltische Blei- und Silberwerke (Anhalt)	2,417	1,416	1,661	1,473	1,970	1,212
Norddeutsche Affinerie zu Hamburg	......	74	8	202	9	302
Mansfelder Gewerkschaft (Silberhaltiges Blei)	291	....	340	....	268	....
Total	67,082	4,319	62,332	4,234	71,881	3,788
Total of A and B	143,991	44,428	135,911	4,334	148,685	3,889

SILVER PRODUCTION OF GERMANY.

	1901. kg.	1902. kg.	1903. kg.	1904. kg.	1905. kg.
Stolberger Gesellschaft	32,225	46,312	31,936	40,068	42,593
Rheinisch-Nassauische Gesellschaft	14,314	14,825	14,325	15,491	15,557
Mechernicher Bergwerksverein	18,214	15,034	6,978	3,239	5,498
A. Poensgen & Sohne, Bleihütte zu Call	8,364	9,956	19,285	16,630	13,742
Rothenbacher Metallhütte im Siegerland	673	504	465	626	590
Victoriahütte bei Niederfischbach	1,323	463	......	......	......
Gesellsch. des Emser Blei- u. Silberwerke	6,236	4,739	2,324	2,622	4,540
Blei- und Silberhütte Braubach	23,591	29,536	28,155	28,400	34,620
Walther-Croneckhütte bei Rosdzin (Giesche's Erben)	1,131	1,228	916	1,545	1,205
Konigl. Hüttenamt Friedrichshütte bei Tarnowitz	7,162	8,981	9,909	·12,563	11,270
Mansfelder Gewerkschaft	99,132	98,446	97,358	100,233	101,270
Oberbergamt Clausthal, Oberharz	27,474	37,224	37,431	36,554	28,468
" " Unterharz	11,289	11,522	11,628	10,425	10,275
Konigl. Oberhüttenamt Freiberg	76,473	91,713	73,656	74,414	70,372
Anhaltische Blei- u. Silberwerke (Anhalt)	7,582	7,434	4,600	4,639	7,497
Norddeutsche Affinerie zu Hamburg	80,610	69,202	74,613	84,686	92,274
Total	415,793	447,119	413,579	432,135	439,791

Of the Silesian plants, the Friedrichshütte, at Tarnowitz, a royal Prussian plant, smelts about 70,000 tons of ore per annum from its own mines. Almost the whole plant has been in late years rebuilt, a costly and complete flue-dust condensation system having been constructed, and much time and money devoted to perfecting the hygienic and sanitary conditions and minimizing lead poisoning. Briefly, the condensation of fume is effected by driving and sucking the hot furnace gases through a system of water-sprayed towers, spraying the fume before and after its passage through the various blowers, and settling and collecting the dust in spitzkasten. The wash water is neutralized by pumping it through towers into which lime water is injected. Thus almost all the sulphur that comes into the works finds its way to the dump in the form of sulphite and sulphate of lime, at a cost, however, which must seriously affect the rentability of the process. The neutralization of the acid water, which formerly ran off to the nearest stream, was insisted upon by the local forest authorities. The lead smeltery attached to the zinc works of Giesche's Erben, viz, the Walther Croneckhütte, at Rosdzin, smelts both ores from its own mines and custom ores and competes actively in the litharge and massicot markets.

Of the Upper Harz works, Clausthal smelts about 14,000 tons of local ores. The resulting bullion is refined and desilverized at Lautenthal. Altenau treats principally lead-copper matte from the Harz and the Rhenish smelteries, performing for these works much the same functions that the Philadelphia plant at Pueblo exercises for the American Smelting and refining Company. Altenau makes its copper regulus of 90 per cent. copper content in the old-fashioned *brillenöfen* and separates the precious metals from the copper in an electrolytic parting plant. The Silberhütte of the Anhalt Lead and Silver Works belies its name more or less by working as a chemical factory. The copper works of Oker and Mansfeld make some lead, the former about 5000 tons; the latter about 500 tons per annum. The Rammelsberg ores are treated at Oker, which also works up copper-lead mattes by the same process as that used in the Philadelphia plant, namely the Swansea process. Freiberg makes only 5000 tons of lead per annum in both its smelteries (Halsbrückehütte and Muldenhütte) but ranks third in silver production.

Freiberg, Braubach, and Anhalt use the Pilz furnace; Tarnowitz, Walther Croneckhütte and Clausthal the round furnace with side-feed and stack; the Rhenish works the oblong furnace, known as the "Raschette-Ofen," with tuyeres (generally 12) at the back, long side only, and with water-jackets on the two short sides.

Another essential difference between American and German practice is to be found in the fact that the Germans generally run a charge richer in lead than the Americans, who treat more dry ores and regard lead concentrates almost in the light of a collector for the precious metals.

One American blast furnace has been blown in at the progressive zinc and lead works at Overpelt, but I believe none has been introduced into Germany. Braubach is said to be quite satisfied with the behavior of its Pilz furnaces and probably as long as it works with a charge containing over 30 per cent. lead, will not introduce the American type of furnace, and consequently not learn the advantages which it offers in the treatment of most kinds of charge, but especially where a big tonnage of ore, with a comparatively small resulting lead fall, is to be treated.

Every German works possesses a refinery and markets its own lead. Some works part their doré bullion; others sell it, as such, to the Scheideanstalt of Frankfurt am Main, which charges at the rate of 90 pfennigs per kilogram of bullion for parting. The process more generally used is the electrolytic; viz, the Moebius or a similar process. Freiberg has recently abandoned parting with sulphuric acid in favor of the electrolytic process.

Freiberg still uses the Pattinson process in addition to the Parkes, with the object of concentrating the bismuth content of the work-lead in the richer silver-lead alloy, which is treated directly in a German cupelling hearth, where litharge, bismuth-litharge, and silver result in the order named. The silver-lead alloy from the Pattinson process is treated in the Parkes pots. The zinc is distilled from the rich scum either in direct- or gas-fired Stolberg retorts, or in graphite crucibles.

Although most of the German smelteries have increased rather than diminished their tonnage in late years, yet the margin of profit per ton is so much less today on account of the reduction in smelting charges that many would have been obliged to shut down if they had not succeeded in reducing materially their smelting costs, and in working with but small metal loss. In this latter particular they can probably show better figures than American works because the lead percentage in the charge is greater, the furnaces are under less pressure and temperature, and the flue and chamber system is either very complete or is aided by methods of deposition and collection such as that of Tarnowitz. The slags dumped are certainly not so clean as the American slags, but on the other hand less is made in proportion to the amount of lead put through the furnace.

Working costs vary in the several works, as those who compare the equipments of Freiberg and Braubach will readily understand; in the more modern class of smelteries, however, costs are not very different from those of American works, viz, 9 to 15 marks against \$2.25 to \$3.25 for smelting, while in refining the laurel is probably with the Americans. Labor in Germany is cheaper (3 marks per shift against \$2.25 to \$2.50) but it is much less efficient. The American works have a better and cheaper system of sampling, more skill in handling large quantities of ore, matte, slag and other materials; they excel in equipment, mechanical and labor-saving, such as feed-cars, settlers, sampling and crushing mills, conveying

and elevating apparatus and smelter bins, in organization, and in the intelligence and endurance of foremen and furnace men.

It is true that the problems facing the metallurgists in America and their brethren in Germany are of a different character. The business of the German is treating small quantities with minimum loss; that of the American is handling large quantities of ore, smelting it rapidly, and getting the finished products away from his plant quickly, avoiding any drag on his routine work that might be caused by the working up of by-products into any form other than that of metallic lead and silver, hard lead, or high-grade matte. The German pays more attention to the treatment of his by-products, which he prepares cheaply for a ready and profitable market. In the handling and separation of slag and matte the Germans have much to learn from the American system of deep settlers with slag overflow and matte tap, and the accompanying excellent dumping-bowl of slag cars. It is still not unusual to see the yard of a fair-sized German smeltery filled with slag cones, from the heads of which workmen laboriously knock off the matte which had run from the furnace with the slag directly into the hand-moved slag pot standing under the slag spout. Nor do the majority of German metallurgists concentrate their lead-copper matte as quickly and cheaply as their American brethren; probably because they roast the low-grade stuff insufficiently in heaps or kilns, and run the furnace charge with so much surplus iron that a considerable concentration in one operation is scarcely possible.

In conclusion, though the technology of lead smelting is more advanced today in America, we must not, however, forget that these small German works have played a röle in the history of lead smelting and refining quite out of proportion to their tonnage and output; wherefore in their present struggle against the forces opposed to them we will wish them "*guten Mut*" and a hearty "*Glueck auf.*" *Bene non vale dicemus.*

LIMESTONE.

The production of limestone in the United States is enormous. Immense quantities are used for building material, both as cut and rough stone, and also in the aggregate of concrete. Large quantities also are crushed for road and railway material. It would be extremely difficult to collect statistics of the production and consumption of limestone for these purposes. We are interested only in the consumption for metallurgical purposes, and to a less extent in the consumption for the manufacture of lime, which is in a way a metallurgical process.

Flux.—The chief metallurgical uses for limestone are in iron, copper and lead smelting. We are able to make a rough computation of the consumption for these purposes on the basis of the ore smelted and metal produced, especially in the case of the iron industry.

The production of pig iron in the United States in 1905 was 22,993,000 long tons; in 1906 it was 25,307,200 tons. In the production of 2240 lb. of pig iron there is required an average of 1150 lb. of limestone. This indicates a consumption of 13,220,000 short tons of limestone for iron flux in 1905, and 14,552,000 tons in 1906. The average value of the limestone used for this purpose may be reckoned at 45c. per ton.

In connection with lead and copper smelting, the bases for estimates are less well established. The bulk of the non-argentiferous lead is smelted from ore having a calcareous gangue and requiring no additional lime flux. The total production of desilverized lead of domestic origin in 1905 was 214,121 short tons; in 1906 it was 227,529 tons. It may be assumed that an average of 10 tons of ore had to be smelted to obtain one ton of lead, which would indicate a total of 2,141,210 tons of ore smelted in 1905 and 2,275,290 tons in 1906. We believe that these figures approximately represent the magnitude of the silver-lead smelting industry in the United States, being too low, rather than too high. Also, a certain quantity of ore of foreign origin is omitted. In American lead-smelting practice at the present time, the average requirement for lime flux is about 16⅔ per cent. of the tonnage of ore smelted. This indicates a consumption of about 357,000 tons of limestone for lead smelting flux in 1905 and 379,300 tons in 1906. The average value of the limestone used for this purpose may be reckoned at 90c. per ton.

With respect to copper smelting, we estimate that 652,000,000 lb. in 1905 and 693,549,000 lb. in 1906 of copper derived from ore outside of Lake Superior was obtained from ore yielding an average of 100 lb. of

copper per ton, indicating respectively about 6,520,000 and 6,935,490 tons of ore smelted. Reckoning the consumption of 8 per cent. of this tonnage for lime flux, we arrive at 521,000 and 554,839 tons respectively of the latter, the average value of which may also be reckoned at 90c. per ton.

The above estimates, which are admittedly rough, are summarized in the following table:

Items.	1905			1906		
Consumed for iron flux	13,220,000 tons	$0.45	$5,949,000	14,552,000 tons	$0.45	$6,548,400
Consumed for lead flux	357,000 tons	0.90	321,300	379,300 tons	0.90	341,370
Consumed for copper flux	521,000 tons	0.90	468,900	554,839 tons	0.90	449,355
Total	14,098,000 tons		$6,739,200	15,486,139 tons		$7,339,125

Lime.—The quantity of burned lime produced in the United States in 1904 was reported by the U. S. Geological Survey as 2,707,809 short tons, valued at $9,951,456. In 1905 the total tonnage was 2,984,100 and the total value $10,941,680. The State of largest production is Pennsylvania, which is followed in the order mentioned by Ohio, Wisconsin, Maine, Missouri, Maryland, Illinois, and New York, each of which produces upward of 100,000 tons per annum.

The cost of manufacturing lime is from $1.20 to $2.90 per 2000 ℔. of product. This corresponds to 4.2 to 10.15c. per bushel of 70 ℔. The cost per 2000 ℔. is divided as follows: Interest on cost of plant and quarry, 5 to 20c.; taxes, minor supplies, etc., 10 to 25c.; cost of quarrying two tons of limestone, 50 to 90c.; cost of fuel for burning, 30 to 75c.; cost of labor (exclusive of quarrymen), 25 to 80c. The minimum estimate represents what might be attained by a good modern plant, run steadily, and under exceptionally favorable conditions as regards quarrying, fuel and labor. The maximum estimate could easily be exceeded in a small or unsteadily operated plant.

LITHIA.

Three different minerals supply the lithia of commerce, viz., amblygonite, lepidolite and spodumene. The only places in the United States where amblygonite and lepidolite are known to occur in workable quantities are at Pala, San Diego county, California, and in the same district near Banner and Julian. Spodumene, the silicate of lithium, is abundant at the Etta mine and elsewhere in Pennington county, South Dakota.

Lithium is never produced as a metal, except as a curiosity, but it is used in the form of carbonate in the manufacture of various lithia waters and lithia tablets, and to a smaller extent as a component of certain preparations for producing colored fire. With only these limited uses of lithium, its production can never be a very important industry, but during 1902 and 1903 this fact was not appreciated, and a very large amount of lithium minerals was produced under the mistaken idea that they were of considerable value. It naturally happened that a large reserve stock accumulated, and in 1904 and 1905 the production fell to almost nothing. In 1905 only necessary assessment work was done at Pala, and the Dakota mines were idle. The prospects for renewed production are small until the accumulated stock shall have been reduced. The remarkable slump in demand for lithia minerals may easily be seen by inspecting the following table of imports to the United States, the increase in stocks in 1904 and 1905 being nicely reflected by the drop in the imports.

STATISTICS OF LITHIUM ORE AND SALTS IN THE UNITED STATES. (*a*)
(In tons of 2000 lb.)

Year.	Production. (*b*)		Imports. (*c*)		Year.	Production. (*b*)		Imports. (*c*)	
	Tons.	Value.	Pounds.	Value.		Tons.	Value.	Pounds.	Value.
1902......	1,245	$25,750	21,216	$22,951	1904......	577	5,155	19	$48
1903......	1,115	23,425	5,596	3,669	1905......	21	252	*Nil.*	....

(*a*) Statistics of the U. S. Geological Survey. (*b*) Ore. (*c*) Lithia salts.

In Vol. VIII of the MINERAL INDUSTRY a six-page article by H. A. Frasch describes the occurrence of lithium minerals in the United States and outlines several processes used in the manufacture of carbonate and phosphate of lithia from lepidolite and spodumene.

MAGNESITE AND MAGNESIUM.

California is the only State in the Union thus far producing magnesite in commercial quantities, but the output does not exceed 4000 tons per annum, that being about the limit of the local consumption. Thus far it has been found impossible to ship the mineral East on account of the high freight rates, the European product being furnished cheaper than the Californian. While there are several dozen known magnesite deposits, thus far very few of them have become productive, and these few are near railroad lines. A wagon haul of any distance is against mining this substance when other deposits have the advantage of being near railroads.

For these reasons the deposits near Porterville, in Tulare county, continue to furnish most of the magnesite used. Both raw and calcined magnesite are furnished from this point. The deposits on Red mountain, in Stanislaus and Santa Clara counties, are being worked, and the product is shipped to factories in Oakland harbor, where some is made into building material and magnesite brick. Carbon dioxide gas is also manufactured at this plant. In Contra Costa county, the Magnesia Product Company, at Nobel, is making magnesia alba, which is sold to the powder companies as an absorbent for high-grade explosives. Some is also made into paint.

STATISTICS OF MAGNESITE IN THE UNITED STATES.
(Tons of 2000 lb.)

Year.	Production (a)		Imports.		Consumption.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1897	1,143	$13,671	(b)	....	....	....
1898	1,263	19,075	16,039	$134,130	17,302	$153,205
1899	1,280	18,480	20,807	(e) 174,779	22,087	193,259
1900	2,252	19,333	28,821	(e) 216,158	31,073	235,491
1901	4,726	43.057	33,461	(e) 250,958	38,187	294,015
1902	2,830	20,655	49,786	373,928	52,616	394,583
1903	1.361	20,515	54,776	461,399	56,137	481,914
1904	2,850	9,298	38,704	286,828	41,554	296,126
1905	3,933	16,221	74,374	638,619	78,307	415,347
1906	....	....	90,396	863,492	....	....

(a) Reported by the State Mining Bureau of California. (b) Not reported. (e) Estimated.

The Napa and Sonoma county deposits continue unproductive owing to lack of demand at remunerative prices. It would seem as if this mineral, which in a well calcined state is fire-proof, would be largely utilized in San Francisco, where there is a demand for a fire-proof struc-

tural material. No attempt to furnish it on a commercial scale has been made, or at least it has not come into general use. As to make it fire-proof all the carbon dioxide must be eliminated, the process is a somewhat expensive one and cheaper materials are more commonly used. More or less work was done during 1906 in new or old claims, but for some reason no new producers have come to the front. Unless some new uses are found, or there is a material reduction in east bound freights, it is hardly probable that the magnesite industry of California will expand to any great extent.

Value.—At the present time crude magnesite is worth about $7 per ton landed at New York and magnesite deposits to be of economic value must be situated near enough to consuming centers so that mining and transportation charges will not be greater than the value of foreign magnesite landed at New York. The magnesite industry of the East is entirely in the hands of The Harbison-Walker Refractories Company of Pittsburg, Penn. This company is engaged in the manufacture of refractories and uses at present imported material exclusively.

The Producing Deposits of Magnesite in California.[1]

By Frank L. Hess.

Livermore Deposits.—Near Livermore, 48 miles southeast of San Francisco, are a number of magnesite deposits; the only one being worked is located 32 miles southeast of Livermore and belongs to the American Magnesite Company. An aerial tram 2500 ft. long, with a capacity of 100 tons per 10 hours, delivers the magnesite to bunkers, from which it is loaded into iron wagons for hauling to Livermore. The wagons carry 17½ tons each, and two are hauled by an oil-burning traction-engine. The magnesite is shipped to Oakland, where the company's factories for brick, carbon dioxide, and other products are situated.

The magnesite veins stand out prominently and as they are almost dazzlingly white, they can be seen from the higher hills, miles away. One of the veins shows an outcrop 40 ft. broad and traceable through a distance of several hundred feet. Others show smaller outcrops, but the total amount exposed is large.

These deposits were not worked until 1905, owing to their distance from a railroad. The first magnesite was shipped in November, 1905. The vein has been developed through a length of between 250 and 300 ft. Mining is carried on by means of an open cut. With the exception of magnesite shipped to the Western Carbonic Acid Gas Company, of San Francisco, for making gas, all of the magnesite mined here is burned before being shipped. Crude oil is used for fuel; the magnesite is raised to a white heat and kept there for from 20 to 25 min.

[1]Abstract of an article in *Eng. Mag.*, August, 1906.

when it is withdrawn from the kiln. Shipments are made from Hilo, a town on the line of the Southern Pacific railroad.

Pope Valley.—The Walters magnesite mine is located on the east side of Pope Valley, 22 miles northeast of Rutherford. The distance of the mine from the railroad makes hauling expensive, and the mine has never been worked on a large scale and has made no production for several years. A proposed electric road from San Francisco to Lake county, if built, will pass within four miles or less of the mine, in which case it will be in an excellent position to ship magnesite.

The deposits are in three principal groups, two of which lie on the east side of the ravine and the other on the west. In the main group on the east side are three large veins of magnesite which can be definitely traced for distances of about 140, 250, and 230 ft. respectively. At their northern end the western and eastern veins are but 30 ft. apart, with the middle vein probably converging with the eastern one. The dip of the western vein is shown by a shallow shaft to be 50 deg. easterly. The veins stand up boldly and can be seen from any part of the valley not hidden by hills. Longitudinal faults occur in both of the outer veins. Between the large ones are many smaller veins having a general parallelism to the main bodies. At its widest exposure the western one is about 10 ft. thick, of which about 5 ft. on the foot-wall is solid white magnesite, while the upper 5 ft. (on the hanging wall side) contains many inclusions of serpentine. The structure of the east vein is similar, and in places the magnesite may be seen grading into the country rock. It is about 12 ft. wide where exposed in a shallow cross-cut. In the middle vein a width of 18 in. to 5 ft. of clear white magnesite is exposed.

Porterville Deposits.—The Willamette Pulp and Paper Company is working a large group of magnesite veins four miles northeast of Porterville, in the outer range of foot-hills of the Sierra Nevada mountains. The veins occur in a brown serpentine.

ANALYSES OF MAGNESITE AND MAGNESIA.

	1.	2.	3.	4.	5.	6.	7.	8.
Silica (SiO_2)	0.10	0.50	1.10	3.30	2.48	a1.81	a6.68	a0.90
Iron Oxide (Fe_2O_3)	.25	.30	.47	7.25	6.02			
Alumina (Al_2O_3)	......	......	.76	.84	1.18	.08	15.10	.49
Lime (CaO)	.62	.70	......	2.48	4.34	Trace	......	1.49
Magnesia (MgO)	47.10	46.90	97.35	84.72	84.56	46.55	37.23	44.39
Carbon dioxide (CO_2)	51.98	51.60	......	......	......	51.25	40.98	50.06
Water & undetermined	......	......	.32	1.40	1.33	.32	......	2.57
	100.05	100.00	100.00	99.99	99.91	100.00	99.99	99.90

(*a*) Insoluble.

1, 2. Alameda claim, American Magnesite Company, 32 miles southeast of Livermore, Cal.
3, 5. Same, calcined.
6. Chiles Valley.
7. Same, poor; not shipped.
8. Porterville;
Analyst, Nos. 1-5, unknown; 6-8, Abbot A. Hanks, 531 California street, San Francisco, Cal., October 1, 1903.

Magnesite in Foreign Countries.

Magnesite is produced in Greece, Austria and to a small extent in India. Germany does not produce the mineral magnesite, but exports considerable quantities of the chloride and the sulphate of magnesium. India has deposits of magnesite in the Chalk Hills in the Salem district of the Madras Presidency, and in 1905 there were produced 2063 tons, an increase of 748 tons over 1904. Russia formerly imported all its magnesite, but by the imposition of a customs duty and a special railroad tariff, the native product of the Urals is enabled to compete.

South Africa.—Large deposits of magnesite of excellent quality have been discovered in South Africa, and active production is well under way. The mines are situated 300 miles from Johannesburg and 87 from Delagoa Bay. The property is being mined by the Magnesite Mines of South Africa, Ltd., whose engineer, J. B. Garbe, estimates that at the point now being worked there are about one million tons of the mineral. Kilns to treat 40 tons of magnesite per day have been erected. The milling plant, which is driven by a suction gas plant, is capable of dealing with about 100 tons per day, and further capital is being raised to erect the additional kilns needed to work the plant to its full capacity.[1]

Austria-Hungary.—Magnesite was discovered in the province of Styria in the early eighties by Carl Späte r, of Coblenz, Germany, who had acquired some property near Veitsch. The first magnesite was produced in small quantities in 1882, but its production on a large scale began in 1890, when shipments were first made to the United States. In 1897 the Veitsch property was organized into a stock company with 2,000,000 crowns ($400,000) preferred shares and 6,000,000 crowns ($1,200,000) common shares, the preferred shares to pay 7½ per cent. interest, guaranteed. This dividend was soon raised to 10 per cent., and has been paid on preferred and common shares alike for several years past. Carl Späte r retains the controlling interest in the company and a Vienna bank owns a large part of the remainder of the shares.

Magnesite mines were operated in competition with the Veitsch mines at Eichberg, lower Austria, for a number of years but were acquired by the Veitsch company in March, 1905. There are also magnesite mines in Hungary, which operate under a cartel agreement with the Veitsch company. The Veitsch company is acquiring other mines in Styria and is desirous of maintaining a monopoly of the business in Austria. At present, however, there are two or three independent concerns opening up magnesite properties in Austria and in Hungary, and there is some prospect of a lively competition after the furnaces are started during 1907.

At present the Veitscher Magnesitwerke Actiengesellschaft is the principal company operating magnesite mines and is enjoying extraor-

[1] *Eng. and Min. Journ.*, May 12, 1906.

dinary prosperity, its shares, which have a par value of 400 crowns (crown =20.3c.), now selling as high as 1140 crowns. This great rise in the company's shares occurred mainly in 1906, and is due to the continually increasing sales of magnesite to the United States.

A report from Consul-General W. A. Rublee at Vienna states that the export of calcined magnesite from Austria to the United States, amounting during 1906 to nearly 53,000 tons net weight from the Vienna consular district alone, has attracted general attention to this industry, and new magnesite mines are being located in several parts of Austria.

THE PRINCIPAL SUPPLIES OF MAGNESITE.
(In metric tons.)

Year.	Austria-Hungary. (a)	Germany. (d)	Greece. (e)	India. (e)	United States. (e)
1897	(c)	53,086	11,311	(f)	1,038
1898	(c)	50,114	14,829	(f)	1,146
1899	(c)	60,910	17,184	(f)	1,161
1900	(c)	67,988	17,277	(f)	2,043
1901	(b)40,236	67,732	13,410	(f)	4,286
1902	(b)53,467	58,947	27,103	3,597	2,567
1903	(b)69,058	60,834	25,657	838	1,234
1904	(b)53,781	65,142	35,989	1,193	2,585
1905	(b)92,359	58,578	....	2,096	3,567

(a) Exports. (b) Calcined magnesite. (c) Previous to 1901 magnesite was included with other minerals not elsewhere specified. (d) Chloride and sulphate of magnesium. (e) Crude magnesite. (f) Not reported.

Technology and Uses.

Magnesite is a valuable source of carbon dioxide, and is far superior to limestone for this purpose. Limestone yields 34 per cent. of CO_2 upon calcining, while magnesite gives 52 per cent. The heat required to drive off CO_2 from magnesite is less than that needed in the case of limestone, and the residual magnesia is more valuable than the lime which results from calcining limestone. The burned magnesia is used for making refractory brick for use with basic smelting charges, as an adulterant of paint and medicinally. Magnesite which has been converted into the sulphate is largely used in the digestion of wood pulp in paper making. The light carbonate (*magnesia alba levis*) is employed for medicinal and toilet purposes, and with the admixture of asbestos ranging from 15 to 40 per cent. of the whole weight, is used for pipe covering and boiler lagging. Sulphate of magnesium, or epsom salt, is largely used in warp sizing or weighting in cotton mills and smaller amounts are used in medicine. Chloride of magnesia is used in cement making and the hydrate in sugar refining.

Messrs. Goodwin and Mailey[1] have investigated the manufacture and properties of fused magnesia. They produce this substance from a convenient apparatus which consists of an electrically heated graphite tube surrounded by a powdered coke jacket into which the pure magnesium

[1] *Trans.* Am. Electroham. Sec., May, 1906.

oxide is fed. After heating in the electric furnace, blocks of the fused oxide were obtained from the interior, of which rods of the pure product were cut. The surface of the fused oxide is like that of glazed porcelain, but the crystalline structure becomes more apparent when the cooling is slow. The melting point is approximately 1920 deg. C., and the specific gravity is 3.493 at 20 deg. C. The hardness of this magnesia approximates to that of feldspar. Below 1100 deg. C. fused magnesia conducts the electric current less than porcelain, but above this temperature its conductivity is greater.

The following chemical properties of fused magnesia are interesting. It possesses in a remarkable degree the ability to withstand chemical action of many neutral salts at high temperatures, and is, therefore, well adapted for use as vessels and apparatus for containing such salts when subjected to high temperatures. Silver, sodium and potassium nitrates, sodium and potassium chlorides, bromides and sulphates, zinc chloride and barium nitrate showed no action on a polished sample of the fused oxide, when the latter was heated for an hour or more in the fused salt. Barium chloride had a very slight action; sodium carbonate, potassium sodium carbonate, potassium hydrate, and cryolite attacked the fused oxide energetically. Dilute hydrochloric, nitric and sulphuric acids attack the fused oxide in the cold slowly. Concentrated acids are less active than the dilute acids.

Manufacture of Liquid Carbon Dioxide.

The Western Carbonic Acid Gas Company, of San Francisco, with works across the bay at Sedan, or Emeryville, was the only manufacturer of gas from magnesite in 1905, although one of the subsidiary companies of the American Magnesite Company was erecting a plant at East Oakland (Clinton).

The magnesite is burned in a kiln with one-tenth its weight of coke, and the gas is pumped into scrubbers, of which there are three, filled with broken limestone to counteract any sulphuric acid formed, and washed with sea water. The gas then passes to an absorption tower where it comes in contact with a sprayed solution of potassium carbonate, by which it is absorbed. The "loaded solution" is then pumped into boilers where it is raised to a temperature just below the boiling point of water. The solution gives up its gas and is pumped back to the absorption tower while the gas is pumped through cleansing tanks to a gasometer. It is then compressed in a three-step compressor to the liquid state and is shipped in steel cylinders holding 25 to 60 ℔. In this process the weight of gas obtained is about 5 per cent. of the weight of the magnesite used. The gas is shipped all over the Pacific Coast and the southwestern States. It is used in refrigeration and in making soda water and other carbonated beverages.

This same company is preparing to make light "magnesia alba" by the Pattinson process, to be used as an absorbent for nitroglycerine in the manufacture of dynamite. It is said to make a superior absorbent for the best grades of dynamite.

Magnesium.

Magnesium is prepared by the electrolysis of a chloride solution. The production of the metal is made chiefly in Germany, but small amounts are made in England and the United States. No statistics of the production are available.

This metal is white in color with a silvery luster, being much whiter than aluminum and also lighter than that metal. Its low specific gravity renders it very useful in making light alloys of aluminum and magnesium, which can be made lighter than aluminum alone and as strong as brass.

Uses.—The chief uses of magnesium are as an illuminant in photography, and as a deoxidizer of metals. In the latter field it is becoming especially valuable on account of its extreme affinity for oxygen, and is used to a considerable extent in copper and brass castings to insure soundness, which it does by reducing the harmful oxides to the metallic condition.

Magnesium appears on the market in the form of sheets, ribbons, powder and in long sticks, 0.5 in. square. For the purpose of alloying with copper, magnesium in sticks is the only form applicable, but a more convenient way is to make a rich alloy of copper and magnesium, and then to add the necessary amount of this alloy. To make an alloy of copper and magnesium, melt 45 ℔. of Lake copper under a good layer of charcoal, so that it will not oxidize any more than is necessary, and then introduce 5 ℔. of stick magnesium, one stick at a time. The sticks have to be held under the copper, in order to insure proper mixing, since magnesium is extremely light. The melt should then be stirred with a graphite rod, and poured into small ingots. The amount of magnesium required to insure a sound copper casting is only about 0.1 per cent. It is therefore much easier to use a high alloy of the two metals rather than the pure magnesium.

Bibliography.

In Vol. II of The Mineral Industry, the occurrence, production, preparation and uses of magnesite were quite fully treated. In Vol. XI there is a description of the process of manufacturing Epsom salt from magnesite, and a short account of some deposits of epsomite in Wyoming. Aside from these two instances, only the usual statistics of production and statements of mining conditions are to be found in the articles on magnesite.

Following is a list of some of the recent articles and patents which relate to magnesite and magnesium:

"The Magnesite Mines of South Africa." (*Engineer, London,* Sept. 14, 1906.) Describes the occurrence of magnesite in South Africa and the present state of its development.

C. Vautin, London. The use of magnesium or aluminum silicides for reducing metals from their oxides, and the application of the process for welding iron and steel. Brit. pat. 10,881 of 1905.

Magnesium Cement and Process of Manufacturing Same.—Willi Jeroch, Berlin, Germany. U. S. pat. 833,930.

MANGANESE.

BY EDWARD K. JUDD.

The sources of the manganiferous materials consumed in the United States may be enumerated as follows: 1. Manganese ores, i. e., containing around 50 per cent. metallic manganese, mined in the Shenandoah valley and in Campbell county, Va.; at Batesville, Ark.; at Cartersville and Cave Springs, Ga.; and at several points within a short radius of San Francisco, Cal. 2. Manganiferous iron ores, the great majority of which carry less than 10 per cent. manganese, obtained at certain of the Lake Superior iron mines; at Batesville, Ark., and at Leadville, Colo. 3. Manganiferous silver ores, poor in silver but averaging nearly 20 per cent. manganese, and valuable as blast furnace flux, mined at Leadville, Colo. 4. Manganiferous zinc ore recovered by the New Jersey Zinc Company in the smelting of its franklinite ore from Franklin Furnace, N. J. 5. Imported ores carrying above 50 per cent. manganese, and coming mainly from Brazil, British East Indies, Russia and Cuba. 6. Imported manganese-iron alloys, coming from Europe.

The ores enumerated under 1 and 5 are used largely in the preparation of ferro-manganese for the steel industry; the chemical trade and other miscellaneous users of manganese oxide also consume large quantities of these rich ores, but depend almost wholly upon imports. The ores under captions 2 and 4 are made into spiegeleisen. Those under 3 are utilized by silver-lead smelters as an excellent flux, incidentally recovering whatever small silver value they may contain.

The mining of true manganese ores in the United States cannot be said to be in a flourishing condition. This is partly due to lack of suitable ores, for the deposits in the Shenandoah valley of Virginia, although not wholly exhausted, are pockety and of indefinite duration, but the discovery of manganese minerals in various parts of the country is of almost daily occurrence. The real explanation is probably to be found in the superiority of the foreign ores and in the cheapness with which they can be brought to consuming centers in this country. The Brazilian ores, in particular, excel all others in freedom from silica and phosphorus; the Russian ores are not much better than ours in respect to these highly deleterious impurities, but run uniformly higher in manganese. On the other hand, the Virginia ores, while up to the standard in manganese, are penalized for their silica, and the utility of the Arkansas ores is restricted by their high phosphorus. Although the producing districts of Russia,

India and Brazil are uniformly at long distances from the sea, whereby heavy costs for land transport are entailed, the mining labor is inexpensive and, once arrived at a port, the ores are shipped as ballast at very low rates, and enter the United States free of duty. As the heaviest consumption of such ores in this country centers in the South and East, the total cost of the foreign ores laid down at the works is low enough not to stimulate active competition from the producers of domestic ores.

PRODUCTION OF MANGANESE ORES IN THE UNITED STATES. (*a*)
(Tons of 2240 lb.)

Year.	Manganese Ores.				Manganiferous Iron Ores.				Man.Zinc Ores.	Total Production.	
	California.	Georgia.	Virginia.	Other States.	Arkansas.	Colorado.	Lake Superior.	Va. & N. Car.	New Jersey.	Long Tons.	Value.
1896...	318	2,538	1,588	*Nil.*	3,038	9,072	110,317	....	35,655	162,526	$339,083
1897...	450	962	2,408	190	4,430	18,600	80,260	....	50,000	(*b*)158,600	328,176
1898...	393	2,477	3,307	1,250	2,775	17,792	112,318	....	47,470	187,782	416,627
1899...	263	1,623	3,626	105	855	29,161	53,702	....	53,921	143,256	306,476
1900...	131	3,447	7,881	312	*Nil.*	43,393	75,360	*Nil.*	87,110	217,546	1,172,447
1901...	610	4,074	4,275	3,036	*Nil.*	62,385	512,084	20	52,311	638,795	1,644,117
1902...	846	3,500	3,041	90	*Nil.*	13,275	884,939	3,000	65,246	973,937	2,145,783
1903...	16	500	1,801	508	*Nil.*	14,856	566,835	2,802	73,264	660,582	1,670,349
1904...	60	*Nil.*	3,054	32	600	17,074	365,572	*Nil.*	68,189	454,581	789,132
1905...	1	150	3,947	(*e*)20	3,321	45,837	720,098	*Nil.*	90,289	863,663	1,681,472

(*a*) Statistics of 1900 and subsequent years are by the Geological Survey. (*b*) Includes 1300 tons of manganiferous iron ore from Vermont. (*e*) Estimated.

Manganese Mining in the United States.

Arkansas.—The manganese deposits at Batesville in north-central Arkansas are peculiar in that the ore is braunite, Mn_2O_3, whereas elsewhere pyrolusite and psilomelane are the prevailing manganese minerals. The ore occurs as nodules in a clay which has presumably resulted from the decomposition of a limestone. The State possesses other deposits, not of commercial size; its combined output has never reached 7000 tons in one year, and of late mining has practically ceased. The old dumps yield a spasmodic output of hand-picked ore, which is used in the iron industry in spite of its unfortunate high phosphorus contents.

California.—The principal productive locality of California is near Livermore, Alameda county. A certain amount of exploitation also has occurred at the following localities: Near Elsinore, Riverside county; near Cloverdale, Sonoma county; and in San Luis Obispo and Santa Clara counties. The most recent development has taken place on Red Mountain, between Potter and Redwood valleys, in Mendocino county. The California Ore Company has secured these deposits, and is opening them actively; an aerial tram will be employed to carry the ore down into Redwood valley.

Colorado.—Certain of the Leadville mines produce a limonite carrying

silver and manganese. The silver is rarely worth the cost of extraction, but the iron and the manganese render the ore a valuable adjunct in the silver-lead smelting industry of the State, by reason of the fluid slags which these oxides induce. About 163,760 tons of such ore were utilized for this purpose in 1906. Occasionally the percentage of manganese reaches as high as 30 or 40, and such an ore is consigned to steel works, particularly to the Illinois Steel Company, at Chicago, and the Colorado Fuel and Iron Company at Pueblo. Of such ore, 36,000 tons were shipped during 1906. Discoveries of manganese ore have been reported from the neighborhood of Gore cañon, in the district traversed by the new Moffat railroad.

Georgia.—This State has never produced as much as 10,000 tons of manganese ore in one year; its deposits, however, are worth consideration owing to their proximity to the iron and steel centers of the South. At Cartersville the ore is found in the form of nodules disseminated through clay; at Cave Spring it occurs in a bedded deposit with chert. At the former locality a certain amount of renewed activity took place during 1906.

Lake Superior.—The red hematites of the Lake Superior ranges not infrequently carry appreciable and important amounts of manganese oxide. The Gogebic range in Michigan-Wisconsin appears to be the best provided with ore of this sort. Only the ores richest in manganese are used for making spiegeleisen; the others are treated like the ordinary ores of the district. In the following table are grouped a number of mines from each range whose ores show the most noteworthy proportion of manganese; the percentages are based on ore dried at 212 deg. F., and represent the averages of shipments in 1905:

MANGANESE IN THE LAKE SUPERIOR HEMATITES.

Mine.	Range.	Mn %.	Mine.	Range.	Mn %.	Mine.	Range.	Mn %
Bonnie	Gogebic	6.01	Lake	Marquette	0.55	Group No. 3	Mesabi	3.54
Cary Empire	"	2.37	Princeton	"	0.52	Higgins (Basic)	"	1.37
Newport	"	5.81	Salisbury No. 2	"	0.41	Hobart	"	1.06
Ottawa	"	3.05	Ajax	Menominee	0.48	Kinney	"	1.25
Ottawa Mang.	"	6.25	Bristol	"	0.82	Leonard	"	0.78
Rand	"	2.72	Chapin	"	0.40	Mayas	"	8.00
Wisconsin	"	7.13	Davidson	"	0.38	Mohawk	"	1.36
Cambridge	Marquette	0.52	Hemlock	"	0.33	Pillsbury Mang.	"	1.79
Clinton	"	0.30	Manganate	"	3.05	Chandler	Vermilion	0.13
Jackson, South	"	2.72	Manganate No. 2	"	4.91	Jura	"	0.13

New Jersey.—Nearly every mineral found in the New Jersey Zinc Company's mines at Franklin Furnace carries more or less manganese, but the franklinite, having the composition $(FeMnZn)(FeMn)_2O_4$ and containing, when pure, 35.7 per cent. manganese, is the one in which it principally occurs. After extracting the zinc for the manufacture of oxide,

the residue of the ore is smelted into spiegeleisen in blast furnaces, of which there are two at Palmerton, Pa., one at Bethlehem, Pa., and one at Newark, N. J. The spiegeleisen made in these furnaces averages 20 per cent. manganese, and is guaranteed to contain not more than 0.08 per cent. phosphorus; in respect to its phosphorus contents, this spiegel is superior to that made at many of the steel works from true manganese ores. It is not attempted to enrich the spiegel beyond 20 per cent. manganese, there being no pecuniary advantages in so doing, and iron ore is usually added to a particularly rich residue to maintain the average grade of the spiegel. The zinc in the residue introduces no difficulties beyond the necessity for a more extensive system of catching the flue dust, composed largely of zinc oxide, which tends to choke the regenerating plant. A new development in 1906 was the installation of firebrick regenerating stoves. It had been supposed that only iron stoves would withstand the action of the zinc oxide, but, following a European model, a firebrick stove was set up in connection with one of the furnaces at Palmerton, and it has given complete satisfaction.

During 1906 a little over 100,000 tons of residue were produced, but not all of it was smelted, owing to the temporary dismantling of the Palmerton blast furnaces. The New Jersey Zinc Company's spiegeleisen furnaces have a combined output of 3000 to 4000 tons per month, and are now working at full capacity.

Virginia.—This State has long afforded the principal supply of domestic manganese ore. Shipments in 1906 were greatly curtailed by the temporary closure of the Crimora mines, but the output of 1907 is expected to resume normal size since a number of new deposits were developed to the shipping stage during 1906, and a few old mines, active shippers in former years, were reopened.

There are two principal producing regions: The Shenandoah valley, in the northwest, is the more important, and Campbell county, near the center of the State, is the other. In the Shenandoah valley the operators, not all of whom are producers at this moment, are: Crimora Manganese Company, Crimora; Old Dominion Manganese Company, Crimora; Manganese Corporation of Virginia, Vesuvius; Kendall & Flick, Lyndhurst; Huston Manganese Company, Huston Station; Steele Ore Company (leasing its property to the Raymond Mining Company), Stuarts Draft; H. P. Binswanger Company, Woodstock. In Campbell county, the largest operator is the Piedmont Manganese Company, of Lynchburg. Recent development of manganese orebodies has occurred also at Evington, Campbell county, and near Cedar Springs, Smythe county.

The Crimora Manganese Company suspended operations in October, 1906, pending changes of management, and additions and alterations in its plant. A description of this company's property and its method of

working will be found on a following page. Its neighbor, the Old Dominion Manganese Company, whose ore-bearing ground is part of the same basin as that worked by the Crimora company, has utilized a development shaft sunk by the latter concern, and has drifted into the basin. It will be obliged to extract the ore by the same inefficient method as that pursued by the first exploiters of the basin, namely drifting through the watery clay, under the protection of heavy timbering, and removing whatever nodules of the manganese oxide happen to be encountered. The Manganese Corporation of Virginia has been developing some old mines at Vesuvius, which in the past produced small tonnages. An extensive body of ore has been blocked out and preparations for handling it were completed before the end of 1906. During the closing days of the year a trial run of the plant was made, and henceforth the company expects to produce regularly 25 or 30 tons per day. The Raymond Mining Company, on the advice of its engineers, relinquished its lease and ceased operations at Stuart's Draft; development of the property was at once resumed by its owner, the Steele Ore Company. The ore nodules in this deposit are scattered through a clay bed 25 ft. thick, but the ore requires concentrating, and, as shipped, averages only 40 per cent. manganese, besides 5 per cent. iron and 9 per cent. silica. The H. P. Binswanger Company, of New York, is reopening the Fort Powell manganese tract, between Woodstock and Seven Fountains, and is reported to have discovered a quantity of ore, averaging 47 per cent. metallic manganese and 0.25 per cent. phosphorus. The recently incorporated Piedmont Manganese Company has acquired the old manganese mines in the extreme northeast corner of Campbell county, two miles east of Mt. Athos, worked some years ago by the Lernel Mining and Manufacturing Company, and developments for large shipments of ore are in progress.

The Crimora Manganese Mine.

The Crimora mine, lying at the west base of the Blue Ridge in the Shenandoah valley of Virginia, has long been the largest and most regular producer of manganese ore in the United States. Just now operations are suspended for a few months, pending changes in management and equipment. The mines are owned by the Crimora Manganese Company, of New York, and are in charge of Frank W. Wood, at the mine.

Crimora is on the Shenandoah division of the Norfolk & Western Railway, six miles north of Basic, Va. The mine is two miles east of the station but is connected with it by a standard-gage spur.

The Potsdam quartzite, the base of the district, forms a bowl, perfectly inclosed on all sides, except where, at the north, a stream has eroded a narrow gorge. This drains only a small part of the bowl, and the remainder has no natural drainage whatever. Within this bowl the decomposi-

tion of overlying strata has left a deposit of stiff, red clay, 212 ft. deep at the center, covered by a layer of gravel drift with an average depth of 15 ft. In this clay manganese oxides are found in rounded concretions, and irregular pockets, seams and stringers, whose position follows absolutely no apparent rule. They are found at all depths, and in all shapes and sizes, from that of a pea to masses of several tons weight. All the ore nodules, however, combine in offering indubitable evidence to their origin as hydrolytic precipitates; the rounded nodules almost invariably present a concentric structure, layers of psilomelane alternating with layers of pyrolusite, while the stalactitic forms of the more irregular masses are too familiar to admit of uncertainty. An occasional geode is found in which crystalized pyrolusite forms a black velvet lining to a shell of psilomelane. Wad manganese prevails in certain portions of the deposit. Some of the adjacent country rocks, on the mountain side, have been demonstrated to carry minute proportions of manganese, and these have no doubt provided the source of the ore now concentrated in the basin.

The magnanese-bearing portion of the bowl has been determined, by drill borings, to cover an area of a little over 100 acres, of which the Crimora Manganese Company owns 73 and its neighbor, the Old Dominion Manganese Company, the remainder.

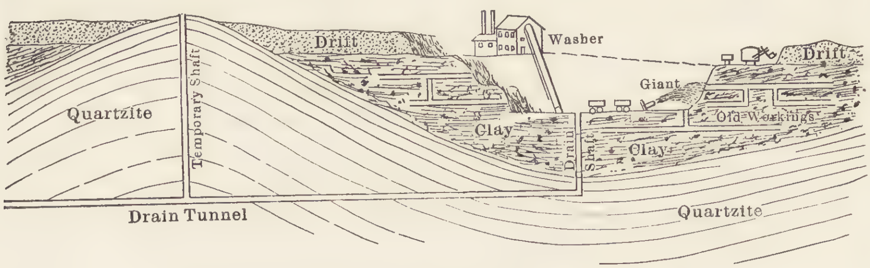
In the early days, when the ores first attracted the attention of the Carnegie steel interests, the clay was thoroughly honeycombed with shafts and drifts which required timbering of the most substantial nature. The lowest workings of that epoch reached a depth of about 100 ft. below the original surface, when the copious influx of water, naturally gravitating to this spot and finding no outlet, forced abandonment of the project. Although the old operators removed the choicest ore bodies, the inefficiency of their methods is shown by the great quantity of valuable ore that is now recovered, by open working, from the close neighborhood of the old openings, and even in among the very timbers of the disused drifts.

The present system of mining may have been suggested by that so successfully practiced among the largest gold gravel operators of the Sierras, but it is probably unique among manganese miners. The first step was to drive a drainage tunnel 5800 ft. long through the quartzite rim, tapping the bottom of the basin at its lowest point, and discharging into one of the small streams that flow into the south fork of the Shenandoah. A shaft 202 ft. deep then connected its inner end with the surface. In the neighboring mountains an extensive system of reservoirs and flumes was then constructed, ending in a smaller masonry tank above the mine, from which a steel pipe, crossing on a trestle the deep gulch above referred to, leads the water to a nozzle in the working place. The tank is now 224 ft. higher than the nozzle, and the discharge is 640 gal. per minute.

The wash from the clay banks thus attacked is allowed to settle in the

bed of the working place, so that only the clay is carried off in the overflow down the drainage shaft. The sand and gravel, together with the big fragments of ore, are then collected in cars traveling on movable tracks, and carried to the washing plant close to the edge of the pit. The area now under work covers 5½ acres, and a depth of 56 feet below its present level will have to be gained before the untouched ore will become available. The entire output now comes from ground that was overlooked by the earlier workers. An ordinary section of clay bank, in which no especially heavy lumps are encountered, will carry around 10 per cent., by weight, of manganese oxides. Wad, which is rather abundant in some spots, cannot be saved in this way, and is lost. In any case it would require calcining to bring it up to shipping grade.

A steam shovel is used for removing the gravel overburden. Trains of dumping cars are brought around to it, on track laid close to the edge of the pit, loaded, and drawn off to a waste dump at some distance from the mine, at a total cost of about 6c. per cu. yd. Waste from the washery is disposed of in the same way.



METHOD OF WORKING THE CRIMORA MANGANESE DEPOSIT.

The washery has a double-tracked incline extending from its top floor to the bottom of the pit, and carrying a pair of self-dumping skips. The loaded cars from the pit, whose contents still retain a lot of adherent clay, dump into a sump which is kept half full of water so as to cause the material to flow easily. The mixture is then drawn off through gates into the skips and hoisted to the top of the washery. Here it passes into an inclined double logwasher fed abundantly with water, where most of the clay is washed off and discharged. The lumps then go through a crusher and into another, but horizontal, logwasher, where the last clay is removed.

The discharge from this washer goes into a trommel with ⅝-in. holes; the oversize passes to picking belts where waste is discarded and the ore becomes ready for market. The undersize goes to another trommel, which makes four sizes, and each size falls into a separate jig. The McClannahan jig has proved admirably suited for this work and makes a clean separation into heads and tails with no middling product. The tails,

carrying 3 or 4 per cent. of manganese, are sold to foundries and basic steel makers. The waste from the picking belts sometimes carries as much as 20 per cent. manganese. A great deal of this waste is in the form of a quartzite conglomerate, manganese oxide being the cementing agent, and, if crushed again, might yield some good ore; the management, however, is storing this material until it can find a market, it being worth more than jig tailings.

No attempt is made to reduce the size of the ore, the management preferring to sell its product in lump form to the steel industry; the Carnegie Steel Company's plants at Pittsburg have been taking the entire output. It is probable that the small additional outlay for grinding equipment would prove a good investment, since the other consuming industries, the glass, the paint, and the storage battery manufacturers, are accustomed to pay about double the price allowed by the steel makers.

The lump ore shipped from the mine averages around 48 per cent. metallic manganese, and occasional lumps reach nearly to 60 per cent. Phosphorus rarely exceeds 0.10 per cent., but silica is rather high, ranging from 2 or 3 up to 15 per cent. The jig concentrates range in silica 2 or 3 per cent. higher than the lump ore produced at the same time.

Imports and Consumption.

By inspection of the accompanying table it would appear that domestic production of manganiferous materials largely exceeds imports; the imported ores, however, carry uniformly more than 50 per cent. manganese,

CONSUMPTION OF MANGANESE ORE IN THE UNITED STATES.
(Tons of 2240 lb.)

Year.	Imports.		Consumption.		Production of Man. Silver Ores. (b)	
	Long Tons.	Value.	Long Tons,	Value.	Long Tons.	Value.
1897	119,961	$1,023,824	278,561	$1,352,000	149,562	$424,151
1898	114,885	831,967	302,667	1,248,594	99,651	295,412
1899	188,349	1,584,528	331,605	1,891,004	79,855	266,343
1900	256,252	2,042,361	473,798	3,214,808	188,509	897,068
1901	165,722	1,486,573	804,568	3,130,690	228,187	865,959
1902	235,576	1,931,282	1,209,513	4,077,065	174,132	908,098
1903	146,056	1,278,108	806,638	2,948,457	179,205	649,727
1904	108,519	901,592	563,100	1,690,724	105,278	348,132
1905	257,033	1,952,407	1,120,696	3,633,879	127,170	445,095
1906	221,260	1,696,643	....	....	163,760	573,160

(b) Mined in Colorado and used as flux in silver-lead smelting; not included in the statistics of consumption.

owing to the classification imposed by the tariff, while the domestic products included in the first table range down to below 1 per cent. manganese. If imported and domestic ores be reduced to a common denominator, say 50 per cent. manganese, the domestic output is seen to provide less

than half the consumption. Ore containing over 50 per cent. manganese, and not more than 10 per cent. iron, enters the United States duty free; an ore outside these limits is classified as iron ore, and taxed 40c. per ton.

Brazil has been the largest single contributor of imported manganese ore. It is noteworthy that, beginning in 1905, the British East Indies came into decided prominence as a source of imports. The advancing importance of the Indian ores is a corollary to the internal disorders of Russia, whereby shipments from the Black Sea have been curtailed. Baltimore, Philadelphia and New York receive over three-quarters of the total imports of manganese ores, most of the remainder entering at Mobile.

Value of Manganese Ore.

The Carnegie Steel Company, which practically controls the market for manganese ores in this country, pays the prices quoted below for manganese-iron ores delivered at the Lucy furnaces, Pittsburg, Pa., the Edgar Thompson furnaces, Bessemer, Pa., or the South works of the Illinois Steel Company, South Chicago, Ill. The prices are per long ton and per unit of metallic manganese: Above 49 per cent. $0.30; 46 to 49 per cent. $0.29; 43 to 46 per cent. $0.28; 40 to 43 per cent. $0.27. Iron is paid for at the rate of 6c. per unit.

The above quotations are based upon ore containing not more than 8 per cent. silica and not more than 0.25 per cent. phosphorus, and are subject to the following deductions: For each 1 per cent. silica in excess of 8 per cent., 15c. per ton is deducted, fractions in proportion; for each 0.02 per cent., or fraction thereof, of phosphorus in excess of 0.25 per cent., 2c. per unit of manganese is deducted. Ore containing less than 40 per cent. manganese or more than 12 per cent. silica or more than 0.27 per cent. phosphorus is subject to refusal or acceptance at the buyer's option. Settlements are based on analysis of sample dried at 212 deg. F., and the percentage of water found by this drying is deducted from the gross weight of the shipment.

As a concrete example of the operation of this schedule, suppose an ore to analyze: Mn, 45; Fe, 15; SiO_2, 9; P, 0.26 per cent. The 0.01 per cent. excess of phosphorus over the base has the result of reducing the payment per unit of manganese from 28c. (to which ore of this grade in manganese is entitled) to 26c., and the total payment from manganese is thus 45x0.26 = $11.70. Adding to this the allowance for iron, 15x0.06 = $0.90, gives a total of $12.60 per ton. The silica is 1 per cent. excessive, so that this total must be reduced by 15c., giving $12.45 as the price per long ton of this ore. If the ore had not more than 8 per cent. silica nor 0.25 per cent. phosphorus, it would bring $13.50 per long ton.

Production and Imports of Iron-Manganese Alloys.

Statistics of production and imports of ferromanganese and spiegeleisen are accurately compiled by the American Iron and Steel Association, which are reported in the accompanying table.

UNITED STATES PRODUCTION AND IMPORTS OF IRON-MANGANESE ALLOYS.
(In tons of 2240 lb.)

	1903		1904		1905		1906	
	Production.	Imports.	Production.	Imports.	Production.	Imports.	Production.	Imports.
Ferromanganese....	35,961	41,518	58,022	21,813	66,179	52,841	55,520	84,359
Spiegeleisen.........	156,700	122,016	162,370	4,623	227,797	55,457	244,980	103,267
Totals............	192,661	163,534	220,392	26,436	293,976	108,298	300,500	187,626

Manganese Mining in Foreign Countries.

The outputs of the principal producers of manganese ores are given in the accompanying table; it will be noted that Brazil, Russia and India provide a great majority of the total production. The Russian deposits are estimated to be capable of supplying the entire world's consumption for a century. The mining of manganese ore in India and Brazil is relatively a recent departure, but in both countries the industry is rapidly expanding. As a whole, the methods of mining, dressing and transporting manganese ores are primitive, owing to the undeveloped character of the regions in which the ores are most plentiful.

Brazil.—The total exports of manganese ore from Brazil in 1906, contributed by six principal producers, amounted to 201,500 metric tons, with a value of $1,296,315. Manganese production at the present time is so costly that during 1906 there would have been few shipments of ore abroad but for contracts made previous to the present high course of international exchange. Definite figures as to the output of the several mines or the exact cost of production are not available. The actual expense at Bahia is about $6 gold per ton, and a price of $7 to $10 a ton in the United States is not very remunerative at present exchange.

The manganese industry is a new one and has recently taken great strides. As there is no metallurgical industry in the country all the ore is exported to Europe and North America. The principal exportation centers are in Minas-Geraes and Bahia, but the industry in this last State still awaits development. In Minas there are two districts. That of Burnier, located 500 km. from the sea by railway, contains manganese deposits included in the metamorphic rocks and associated with limestones and iron ores. The ore exported yields from 50 to 55 per cent. metallic manganese and contains from 10 to 20 per cent. water. It is com-

paratively free from phosphorus. The leading mine in this district is owned by Carlos Wigg, of Rio de Janeiro, a recent analysis of whose ore shows: Mn, 51.90; Fe, 4.31; SiO_2, 1.39; S, 0.20; P, 0.033; water, 16.33 per cent.

The Lafayette district has an ore of a different nature; it appears in fissure veins and has been formed by the decomposition and leaching of a rock containing manganiferous garnets. The ore exported gives from 49 to 51 per cent. metallic manganese, 3 per cent. silica and 0.08 phosphorus.

WORLD'S PRODUCTION OF MANGANESE ORE. (*a*)
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Bosnia (*b*)	Brazil. (*d*)	Canada.	Chile. (*d*)	Colombia.	Cuba.	France.	Germany.	Greece.	India.	Italy.
1896.	5,941	23,265	6,821	14,120	112	26,152	10,668	....	31,318	45,062	15,500	57,783	1,800
1897.	10,043	28,372	5,344	16,054	14	23,528	8,382	....	37,212	46,427	11,868	74,862	1,634
1898.	14,219	16,440	5,320	26,417	45	20,851	11,176	....	31,935	43,354	14,097	61,469	3,002
1899.	10,484	....	5,270	65,000	279	40,931	10,160	....	39,897	61,329	17,600	88,520	4,356
1900.	14,550	10,820	7,939	108,244	34	25,715	8,748	21,973	28,992	59,204	8,050	129,865	6,014
1901.	12,077	8,510	6,346	100,414	447	18,480	95	25,586	22,304	56,691	14,166	122,831	2,181
1902.	12,883	14,440	5,760	157,295	175	12,990	*Nil.*	40,048	12,536	49,812	14,962	160,311	2,477
1903.	11,489	6,100	4,537	161,926	135	17,110	(*c*)	21,070	1,583	47,994	275,232	174,563	1,930
1904.	15,460	485	1,114	208,260	123	(*c*)	(*c*)	33,152	11,254	52,886	239,635	152,707	2,836
1905.	19,496	(*c*)	(*c*)	224,377	22	(*c*)	(*c*)	(*c*)	6,751	51,463	228,182	258,046	5,384
1906.	(*c*)	(*c*)	(*c*)	201,500	84	(*c*)	(*c*)	(*c*)	(*c*)	52,485	(*c*)	(*c*)	(*c*)

Year.	Japan.	New Zealand.	Portugal.	Queensland.	Russia.	South Australia (*d*)	Spain.	Sweden.	United Kingdom	United States. (*e*)
1895........	17,141	213	1,240	361	203,081	49	10,162	3,117	1,293	173,237
1896........	17,967	66	1,494	305	208,025	*Nil.*	38,265	2,056	1,097	165,135
1897........	17,351	182	1,652	403	370,195	*Nil.*	100,566	2,749	609	161,138
1898........	11,517	220	907	68	329,546	*Nil.*	102,228	2,358	235	190,787
1899........	11,340	137	2,049	747	659,301	102	104,974	2,622	422	145,548
1900........	15,228	166	1,971	77	802,234	*Nil,*	112,897	2,651	1,384	221,714
1901........	16,298	208	904	221	522,395	134	60,325	2,271	1,673	649,016
1902........	10,866	*Nil.*	*Nil.*	4,674	536,518	18	46,069	2.850	1,299	989,519
1903........	5,616	71	30	1,341	*d*458,894	10	26,194	2,244	831	671,151
1904........	(*c*)	199	1,851	843	(*d*)358.443	(*c*)	18,732	2,297	8,880	461,854
1905........	(*c*)	55	(*c*)	1,541	(*d*)524,995	(*c*)	26,020	(*c*)	14,582	877,482
1906........	(*c*)	(*c*)	(*c*)	1,131	(*c*)	(*c*)	(*c*)	(*c*)	23,126	....

(*a*) From official statistics. (*b*) Includes Herzegovina. (*c*) Statistics not available. (*d*) Export returns. (*e*) Includes output of manganiferous iron ore.

In Burnier the manganese deposits are worked by underground operations; timbered levels are driven into the ore. In the Lafayette district the workings are open-cast. Morro da Mina at Lafayette and UzinaWigg at Burnier are the two most important manganese mining properties of the country, both worked by native capital. Besides these there are explorations in Piquiry, now exhausted, and at S. Gonçalo. Other installations are being made at Entre Rios, all in the district of Lafayette.

A large manganese ore deposit was prospected in Matto-Grosso, near the Bolivian frontier, by the same firm that explored the Piquiry mine. This deposit is situated some miles from the river Paraguay and offers favorable working and transportation conditions.

India.—The recent surprising growth of the manganese ore industry of India is due both to the inherent excellence of the ore and to the temporary curtailment of Russia's productivity. The main output now comes from two districts, Nagpur in the Central Provinces, and Vizianagram in Madras, but at numerous other points in the Empire, notably in Mysore, valuable deposits are rapidly being exploited. The entire output is now exported, although the establishment of manganese-consuming industries in India is expected in the near future. The growth of the output is shown in the accompanying table. Exports from the port of Bombay

PRODUCTION OF MANGANESE ORE IN INDIA.

District.	1903		1904		1905	
	Long Tons.	Metric Tons.	Long Tons.	Metric Tons.	Long Tons.	Metric Tons.
Central India.						
Jhabua State	6,800	6,909	11,564	11,749	30,251	30,735
Central Provinces						
Bálaghát district	7,898	8,024	10,323	10,489	159,950	162,329
Bhandára district	....	....	8,558	8,695		
Nagpur district	93,656	95,159	66,153	67,214		
Madras.						
Vizagapatam district	63,452	64,470	53,699	54,260	63,959	64,982
Total	171,806	174,562	150,297	152,707	253,896	258,046

during the first 11 months of 1906 were 310,446 long tons, against 185,520 tons during the corresponding period of 1905.

Nagpur.[1]—The ores of the central provinces, near Nagpur, to the north of the Bengal-Nagpur railway, were first alluded to over thirty years ago. Ramtek is one of the principal mining centers. The chief orebodies, averaging 40 ft. thick, dip south in the metamorphic rocks at an angle of 80 deg. Some are exposed in hillsides and are well adapted to the use of aerial trams. The ores of the Central Provinces, besides being much less phosphoric than at Vizagapatam, are more largely braunite, and while the psilomelane ores of Vizagapatam are often cavernous, porous, and friable, those of the Central Provinces are invariably compact, hard, and crystalline.

The most powerful concern in the Nagpur field is the Central Provinces Prospecting Syndicate, whose holdings include large deposits at Ramtek and Balaghat. In 1905 this concern produced approximately 100,000 tons. The ore, fairly uniform in grade, averages as much as 54 per cent. manganese, while silica runs up to 9 or 10 per cent. The Central India Mining Company, Ltd., has been producing since 1904 and in 1905 reported about 40,000 tons. The chief property is connected to the main line by a light railway. The Indian Manganese Company, owning a large ore-

[1]Ralph Stokes, *The Mining World*, June 16, 1906.

body at Kodegowan, among others, recorded an output of some 14,000 tons in 1905. Other firms are Jessop & Co. (actually shipping) and Burns & Co., whose concession, slightly prospected, may be handed over to an American company shortly.

The Central Provinces are severely handicapped by freight charges, although the railway rate is only 0.10 pie per maund (½c. per ton) per mile. But the companies face a charge of $2.30 per ton to Bombay, or $3.10 to Calcutta. Compared with this the expense of mining is small.

Vizianagram[1].—The Vizianagram manganese deposits are near the coast, less than 60 miles from Garividi station. The most productive orebody lies only 11 miles from Garividi, while the Kodur, second in importance, is within a stone's throw of the station. The deposits occur as lenticular masses of great width in the gneiss formation, which strikes roughly east and west, dipping south. Kodur and Garbham, the principal deposits, lie on different horizons, between which there appears to be fully two miles of formation. The Garbham is a most irregular body marked in the plain by a prominent elevation, and altering very considerably in various sections. On this account the excavation, which measures 1200 by 400 ft., presents an irregular appearance, rich portions having been followed down to greater depths than those characterized by large percentage of silica or iron. Parts of the orebody, comparatively poor in manganese, are highly jaspery; others, containing 20 per cent. of iron and perhaps 35 per cent. of manganese, are now being carefully mined to meet the increased demand. Only 25 to 35 per cent. of the ore extracted is shipped, the remainder being waste rock. Some 1500 laborers are employed in the Garbham, the majority being under contractors or "maistries." Their wages are 2 or 3 annas (4 to 6c.) per day and local labor supplies are ample.

The Kodur mine has a basement area of 80,000 sq. ft. and a depth of 60 to 70 ft. The average content of manganese is about 50 per cent., of silica 2 to 7 per cent., and phosphorus 0.25 per cent. The Vizianagram Mining Company has the exclusive right to work manganese on the Samastharam estate, belonging to the Rajah. Outside this area there are but few occurrences, the most noteworthy being at Bankuruvalsa, in Bobbili Taluk.

Mysore.—A new deposit of manganese which promises to be important is being opened in Mysore State. The properties belong to the Mysore Manganese Company, whose office is at Bangalore. They are situated 20 miles northwest of Shimoga station on the Southern Mahratta Railway, which station is 340 miles from the railway's terminal port at Mormugoa. The distance from the deposit to the railway is covered partly by ordinary road, but the four miles nearest the mine are in dense jungle, through which a rough road has been cut. The ground is now being surveyed for a branch line from the railway. Development has been conducted with

[1]Ralph Stokes, *The Mining World*, June 16, 1906.

unusual rapidity. It was only in August, 1905, that the deposit was dis covered. By the end of 1905 mining was in full swing and between March and August, 1906, 20,000 tons were shipped to the coast and 25,000 tons additional made ready for immediate shipment.

On the surface the ore occurs in nodules and boulders cemented together with a form of laterite clay. These vary in size from an inch in diameter to several tons in weight. This deposit extends over an area of several thousand square yards, as far as has yet been ascertained. At one point a well defined lode of high-grade ore has been opened. The ore is very compact, crystalline and hard, with a bright metallic fracture. Its analysis, after drying, gives 55.69 manganese; 2.98 iron; 0.53 silica; 0.015 phosphorus; with traces of sulphur and copper. Ore of this grade is now being shipped at the rate of 5000 tons a month. The surface deposit of nodules and boulders is also being exploited and three grades of ore are being obtained: No. 1 grade, 53.72 manganese and 5.55 iron; No. 2 grade, 43.29 manganese and 11.79 iron; No. 3 grade, 32.76 manganese and 14.28 iron. The silica and phosphorus are about 1 and 0.05 per cent. respectively, in all three grades, and are sufficiently low to bring the ore into favor.

Operations are handicapped by the long and difficult transit. The ore is carried in native carts to the railway at a rate of six rupees per ton, and the railway charge, including loading into the steamer, is another six rupees per ton. There are 500 carts and 1000 bullocks in use in hauling the ore to the railway. Labor is plentiful and the work appears congenial. There is in fact a superabundant population which in periods of scarcity becomes a burden to the State, so that this new source of employment is welcomed by all concerned.

Mining Methods.[1]—The Indian manganese deposits occur both on hillsides, as at Jhabua and at Kandri, and on level ground as at Kodur and Kodegaon, and the Madras localities. In either case, the simplest quarrying methods are employed, the advantages of the hilly location being freedom from drainage difficulties and the ability to profit by the use of aërial trams. In many instances valuable ore has been forfeited by a careless disposal of waste from earlier operations.

Nearly all work is done by hand. When in very hard, compact masses, the ore is first hand-drilled and then blasted. Otherwise it is simply pried out with crowbars, advantage being taken of the divisional planes of the orebody. The huge blocks thus detached are broken with heavy sledge hammers, and then usually carried down the hill, or out of the quarry on the heads of women and children. At quite a number of quarries light rails have been put down to facilitate the disposal of both manganese ore and waste. The ore is, if necessary, cleaned by women, children and old men, with small cobbing hammers, and finally piled into rectangu-

[1] *Trans.* Mining and Geological Institute of India.

lar stacks for measurement. Where a chemist is employed, the stacks are usually sampled and assayed separately, and the ore then carted or trammed to the railway station, where the products of different quarries are often mixed so as to yield a cargo of a certain standard. The working of Kandri—a hill 250 ft. high—may be praised as a model of how such a deposit should be developed. The ore deposit, which forms a sort of backbone to the hill, is attacked by a series of levels at different hights, exposing both the orebody and the country. As these are removed, the waste is conveyed by trucks on tram lines and dumped over the end of the level remote from the working face, so that the levels are constantly being extended away from the orebody, the rails keeping pace with this growth. By means of a system of two aërial ropeways, an inclined plane and some zig-zag tramways, the labor of transportation is greatly reduced.

The Bálághát deposit, also, is being well developed, and has the advantage of an inclined haulage plane by which the ore is let down in trucks some 200 ft. to a chute for delivering it into railway wagons on a small two-mile branch line connecting up with Bálághát station, B. N. R.

Russia.—The manganese industry of Russia suffered severely during 1906 from the internal disorders to which the Empire has been subject. The Russian deposits are probably the most extensive in the world, and an be relied upon, at the present rate of output, for another century. Their development has been slow, owing to the lack of suitable means for transport; the government has, within a few years, built railroads between the most important mining centers and the shipping ports, but even now the charge for railroad haulage constitutes over three-quarters of the total cost of ore delivered at the export harbors.

There are three principal productive localities, all in southeastern Russia, whose outputs in 1905 are estimated as follows: The Caucasian district, Kutais, 21,000,000 poods; the Nikopol district in the Donetz region, 10,000,000 poods; the southern Urals, 282,750 poods. In the Caucasus the industry is divided among a host of small proprietors who mine their ore in a crude fashion, cart it to Tschiatura, and ship it thence by rail to the export brokers at Batum and Poti. All attempts to organize the mining interests for their mutual benefit have thus far failed. An English company has recently leased 135,000 acres of ground at an annual rental of $12,875, and will mine ore during the current year, after having built a branch railroad from Kwirilly, on the Transcaucasian Railway. The ore from this district averages fully 50 per cent. manganese, and sold, during 1905, at $6.37 per ton at Poti.

The deposits in the Nikopol district are not far from the river Dneiper, so that small boats are used to carry the ore to the seaports, Kherson and Nicolaieff, where the export brokers are situated, at a cost of only 90c. per ton. That part of the output of this district which is consumed by

the Silesian iron and steel industry goes all the way by rail, without the intervention of export brokers. The four largest operators in the region are: A. M. Zawadski and the Société Métallurgique Nikopol-Marioupol, at Basaloduski; the Société Métallurgique Dnieprovienne, at Marganetz; and the Société Piroluside, at Nikopol. All the mines are on patented ground, and the royalty varies from 30 to 60c. per pood. French and German companies have recently obtained concessions in the district and within the current year two or three new mines will be opened.

The manganese resources of the Urals have scarcely been touched as yet, but they are supposed to be of vast importance. The latest official returns from this region give the production as follows: 1902, 375,580 poods; 1903, 187,116 poods; 1904, 167,742 poods; 1905, 282,750 poods. The leading operations are: the Nijni Tagilski, the Bogoslovski, the Tungatarovski, the Ushkoff, and the Ramieeff mines. The entire output of the district is consumed locally by metallurgical works whose activity is stimulated by heavy duties levied on imported ferromanganese and spiegeleisen.

New Localities.—In several parts of the world, not heretofore producers of manganese ore, the discovery of such minerals has been reported. A deposit near Djebel Batoum, in Tunis, has been the subject of a favorable report, but a railroad will be required for its exploitation. A deposit in the Duchy of Hesse, Germany, will probably be actively developed by a responsible company. In South Africa, exploration has uncovered deposits of manganese ore at Hout Bay and Constantia, which will be developed by organized capital.

MICA.

By John T. Glidden.

The domestic demand for mica cannot be met by the mines in the United States and the consumption is supplied by importing considerable quantities of mica from Canada and India, which countries are the chief producers, although there is a possibility that Africa may soon enter the ranks. The mica industry is heavily protected and imports are made in the face of a 20 per cent. *ad valorem* duty in addition to 6c. per ℔. on unmanufactured mica and 12c. per ℔ on cut or trimmed mica. Recent improvements in methods of making sheets of mica from scrap by a patented process of cementing have greatly decreased the demand for large sheets of natural mica.

Following are statistics of production and imports of mica for the last 10 years. These statistics are somewhat unsatisfactory owing to the fact that the mica mining industry is largely in the hands of many small producers whose reports are sometimes unreliable and who frequently refuse to make any reports whatever; all statistics of mica production are subject to these difficulties.

STATISTICS OF MICA IN THE UNITED STATES.
(In pounds and tons of 2000 lb.)

Year	Production. (a)			Imports.			
	Sheet. (c)	Scrap.		Unmanufactured.		Cut or Trimmed.	
	Pounds.	Tons.	Value	Pounds.	Value.	Pounds.	Value.
1897	118,852	2,882	$28,820	(b) 722,939	$161,334	226,771	$41,068
1898	110,918	3,529	39,837	877,930	115,930	78,567	34,152
1899	97,586	6,917	50,596	1,709,839	233,446	67,293	42,538
1900	127,241	5,417	42,889	1,892,000	290,872	64,391	28,688
1901	360,600	2,171	19,719	1,598,722	299,065	78,843	35,989
1902	373,266	1,400	35,006	2,149,557	419,362	102,299	46,970
1903	619,600	1,659	25,040	1,355,375	288,783	67,080	29,186
1904	668,358	1,096	10,854	1,085,343	241,051	61,986	22,663
1905	851,800	856	15,255	1,506,382	352,475	88,188	51,281
1906	1,423,100	1,489	22,742	2,984,719	983,981	82,019	58,627

(a) Statistics for 1901 to 1906 inclusive are those of the Geological Survey. (b) The classification required by the Dingley Act went into force July 24, 1897. The figure for unmanufactured includes the entire unclassified import of that year previous to July 24; the figure for cut or trimmed covers the import of that class during only the last five months of the year. (c) The value of sheet mica being so widely variable, and so little indicative of commercial results, and all previous statistics being of doubtful accuracy, they have been omitted from this table.

Mica Mining in the United States.

Alabama.—A discovery of mica is reported in eastern Alabama where the veins are said to be traceable along the outcrop for a long distance. Test pits and shafts sunk at various places disclose mica of good quality and cleavage, from which large sheets can be secured. The property is not exploited at present.

Colorado.—Deposits of mica occur in Boulder and Fremont counties and although no commercial production has as yet been made, preparations for work are already under way. The Cañon City Mica Mills and Mining Company is engaged with plans for a mill at Cañon City for treating mica. The Federal Mica Mining Company of Duluth, Minn., owns a deposit in Fremont county. Some good sized sheets of mica are found on the property.

Idaho.—Mica mining has been in progress in Idaho since 1895, but recently with increased importance. The Spokane Mica Company, of Spokane, Wash., operates a deposit 18 miles northeast from Troy, Latah county. A shaft is down 93 ft. and carload shipments are made weekly to Spokane, where the grinding plant is located.

New Hampshire.—Although New Hampshire is the oldest producing State the industry has declined considerably during the last 15 years and has been kept going only by the demand for scrap mica which cannot be imported on account of the high tariff. During 1906 there was a revival of interest due to the entrance of the General Electric Company into the producing field at Alexandria. Other shipping centers are Bristol, Canaan, Graf ton,Rumney and Warren.

South Dakota.—In the region around Custer City, and Deadwood, mica deposits are worked and during 1906 the scope of operations was considerably enlarged. The Westinghouse Electric and Manufacturing Company operates the New York Mica Mine in Custer county, also the White Star Mine. At the New York mine about 25 men are now employed. The mica is found along both walls of the vein, which is a well defined vertical.

Virginia.—Although numerous deposits of excellent merchantable mica are known in a number of counties in Virginia, some of which have been rather extensively worked in the past, very little activity was manifested in mica mining during 1906. The Pinchback mines, located about two miles northeast of Amelia courthouse, in Amelia county, were the only ones operating. New openings were made during the year on the property with encouraging results. All the products from these mines are marketed; they comprise mica, feldspar, and kaolin, the last of which results from the alteration of the feldspar of the pegmatite dike down to some depth below the surface.

Mica in Foreign Countries.

Africa.—Considerable deposits of mica have recently been discovered in the Vitimiribi mountains, in German East Africa. Specimens examined at Berlin have been pronounced equal to Indian mica. The yield of the mica deposits in the Oulongourou mountains, near Urogoro, has

shown a notable increase of late, the exports for this year being estimated at 100,000 marks.

India.—The total production of mica in India during the year 1905 amounted to 25,641 cwt., of the value of $421,975. In the same year the exports of Indian mica rose to 133,159 cwt., as against only 18,250 in 1904. The total value of the exports, $710,040 against $415,915, did not rise in 1906 in the same ratio, owing to the fact that in Bengal the mica now being mined is of very inferior quality. Present production is chiefly confined to Bengal and Madras, but 2760 cwt. were produced in the Ajmere district of Rajputana during the year. Mica of fair quality has also been reported to occur in the Kishengarh State of Rajputana, but does not appear to have been mined as yet. The prevailing prices of mica in Calcutta ranged during 1906 from $1.38 per pound for one shipment of clear ruby down to 13c. per pound for shipments of stained ruby.

In Bengal, the producing area is about 60 by 15 miles in extent and lies 200 miles northwest of Calcutta. The pegmatite veins, in which the mica occurs, are rarely of great extent and run along the bedding planes of the country roughly in an east and west direction. The erratic nature of many of the deposits has been largely responsible for the crude methods of mining which are still in vogue. The small syndicates do not risk too much in any one mine, preferring to rely upon the law of averages.

Nearly all the Bengal mica which is shipped directly to America passes through the hands of Messrs. Chrestien & Company. Mica won from private land is free from tax. Government land is leased to applicants at a reasonable rate, a dead rent of one rupee per acre being charged. A royalty of 5 per cent. on production is collected unless the rent happens to exceed this amount. This system is not a good one, inasmuch as the unscrupulous holder of government and private lands invariably manages to find all the mica on the latter and the government supervision is lost.

The statistics of mica during 1906 as compiled from the returns of the London warehouses are as follows: In cases of about 100 lb. each; arrivals, 19,075; deliveries, 15,680; stock, Dec. 31, 1906, 6798 cases.

Uses.

The electric industry requires the largest amount of mica, while glazing and decorative industries consume smaller amounts. The softness and insulating power of mica which is free from iron make it very valuable as an insulator between segments of commutators of motors and dynamos. Softness is essential in order that the mica may wear down flush with the copper. A patent was granted in the United States on Jan. 29 to Max Meirowsky, of Cologne-Ehrenfeld, Germany, whereby hard mica is calcined under pressure, being pressed between plates of suitable material which are placed in the calcining furnace. It is stated that the internal

structure of the mica is so altered by this treatment that it becomes softer than the softest natural mineral. For electrical purposes light color and transparency are not essential, but even small traces of magnetite in the sheets render them much more liable to penetration by the current. The considerable expense which formerly attended obtaining large sheets for big armatures is obviated by the use of "micanite," which is made by cementing together small pieces until the desired area and thickness of the sheet is obtained.

Transparent sheet mica is used in stove windows, for chimneys of incandescent lamps, phonograph diaphragms and automobile windows. Scrap mica is used as a basis for boiler lagging. Its insulating properties are not due so much to the non-conducting power of the mica as to the considerable amount of air space inclosed in the interstices of the scrap lagging.

Ground mica is used in increasing quantities in mica bronzes and paint, as an absorbent for explosives, as a lubricant and for decorative purposes, notably in giving luster to high grade wall papers. Owing to the difficulty and slowness of the operation, grinding raises the price of mica considerably above that for scrap, but the price is much lower than for the sheet product.

Scrap from various sources is first cut into small squares and fed through tubes by pneumatic blast into what is known as the "atomizer." This machine has two rotating shafts having closely arranged spiral beaters, fitting into one another and which break up the mica into fine fragments. From the "atomizer" the fine mica is sized by being blown into a room containing a series of bins. The finer particles are carried to the most remote bins while the coarse fragments settle near the end of the tubes. The usual practice is to size between 10 and 160 mesh but for special purposes 200-mesh product is sometimes used. Grinding is also done in old-fashioned buhr mills.

Bibliography.

The customary statistics of the production of mica in the United States, also notes on the progress of the mica industry in this country and abroad, are to be found in every volume of The Mineral Industry. In addition to these articles the following volumes contain matters of special interest. Vols. I and II have extended notes on the condition of the mica mining industry in the United States for the years 1892 and 1893. Vol. VI gives an account of mica mining in Canada and India and complete notes on the uses of mica. In Vol. VII there is an article by J. A. Holmes dealing with the geology of the mica deposits in the United States. In Vol. VIII an article by J. Obalski deals with mica mining in foreign countries and especially with the deposits in Norway. Vol. XIV outlines the industrial conditions in the United States, and discusses the development attained in 1905 in the various producing States.

MINERAL WOOL.

The mineral wool industry changes very little from year to year. The use of the material is confined almost entirely to technical trades in which its non-conducting property gives it value, and there are no more factories in the country today than there were a year ago. In 1906 the production was 5375 short tons valued at $55,550 as compared with 6164 tons, worth $69,560, in 1905. The following table gives statistics for the last 10 years except two years in which no figures of production were collected:

PRODUCTION OF MINERAL WOOL IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Amount.	Value.	Per Unit.	Year.	Amount.	Value.	Per Unit.
1897	5,617	$61,494	$10.95	1902	10,843	$105,814	$9.67
1898	6,560	70,314	10.72	1903	(a)	....	....
1899	7,448	85,899	11.53	1904	(a)	....	....
1900	6,002	60,320	10.05	1905	6,164	69,560	11.28
1901	6,272	68,992	11.00	1906	5,375	55,550	10.33

(a) No statistics collected.

The Columbia Mineral Wool Company with offices in Chicago operates a plant at South Milwaukee and is the largest producer in the West. In the East, the U. S. Mineral Wool Company of New York makes a considerable production.

The disrepute into which mineral wool fell on account of the fact that it contained sulphur, which in the presence of water produced corrosive action on the pipes that were packed in the wool, has been entirely overcome. The elimination of the sulphur is accomplished either by forming a slag from rocks free from sulphur, the product being known as rock wool, or by smelting chilled slag from blast furnaces. The second melting and subsequent blowing of air through the mass assists greatly in reducing the sulphur content of the resultant wool to an insignificant percentage.

In Vol. XI of The Mineral Industry a complete account of the history of the mineral wool industry is given with notes of various methods of manufacture, analysis of different mineral wools, and tabulations of conducting power.

MOLYBDENUM.

A small amount of molybdenum is consumed in the United States for making special steels to which molybdenum imparts great strength and toughness. The two minerals which supply this metal are molybdenite and wulfenite. Their production is erratic and of comparatively little importance from the mining standpoint. There are workable deposits of molybdenite in Arizona, California, Colorado, Idaho and Maine. During 1906 a discovery was reported in California, in the Pigeon Springs district, where an 8-ft. ledge of molybdenite-bearing quartz was opened. Assays showed 15 per cent. metal, and the claim is now being developed. In Chelan county, Wash., the Crown Point Mining Company is operating and a shipment of ore valued at $10,000 is reported. Another discovery is reported from the Southern Pacific mine in the Sierra Madre district, near Ogden, Utah. Among foreign countries, Australia, Canada and Norway are producers and export their ore principally to Great Britain and Germany.

Price.—Molybdenum ore, guaranteed 90 to 95 per cent. MoO_3, is quoted at $400 @ 450 per ton. The price is variable, and usually is a matter of negotiation. In the East the largest buyer is the Primos Chemical Company, of Primos, Penn. In San Francisco, De Golia & Atkins are the principal dealers.

Uses.—Molybdenum, either as a metal or as an alloy with iron, is added to special steels in order to give them certain self-hardening properties. The metal resembles tungsten in its action, but it is claimed that one part of molybdenum will secure the same properties in the steel as two parts of tungsten. It raises the elastic limit of steel, also the tensile strength, and does not increase the brittleness in any way near the same proportion. Particular uses of molybdenum steel, where these properties may be taken advantage of, are in large forgings, guns where great strength and resistance to the erosive action of powder fumes are needed, wire having great elongation combined with high tensile strength, armor piercing shells, motor-car steel and magnet steel.

In the case of high speed tool steel, the actual status of molybdenum as against tungsten seems to be somewhat uncertain, and it is a question whether the advantage is all with the molybdenum. F. W. Taylor stated[1] that 4 to 4.5 per cent. of molybdenum appeared to be a sufficient quantity for making a high speed tool, against 8 per cent. of tungsten

[1] *Trans.* Am. Soc. Mechan. Engineers, New York meeting, 1906.

used for the same purpose. To offset this point in favor of molybdenum, it appears that tools containing it are liable to be brittle, irregular in cutting speed or have a tendency to fire-crack. The cause of this irregularity in molybdenum tool steel is not definitely known. A marked characteristic of tungsten-chromium steels is that they require little or no skill on the part of the blacksmith in sharpening them, and it is impossible to injure their cutting properties even when they are heated to a point where the thin cutting edge is about to melt. It is suggested that the irregularity in molybdenum tools may be because they require a more accurate degree of high heat, and at the same time one which is considerably below their melting point. It is difficult to judge precise points by the eye at these high temperatures and passing the critical point in one tool and getting the correct heat in another may explain why two tools of the same chemical composition, and apparently treated alike, give such variation in cutting speed.

Bibliography.

Articles on molybdenum appear in Vols. VI to XIV, inclusive, of The Mineral Industry, except Vol. XII, and the usual reviews of the progress of molybdenum mining are given. Special features of these articles are as follows:—In Vol. VII, a procedure for determining the amount of available molybdenum in molybdenite, and also descriptions of approximate volumetric methods. In Vol. XI some notes on experiments to determine suitable methods of concentrating Canadian ores are given. Vol. XIII contains data on the reduction of molybdenite to the metal and a review of molybdenite mining in the United States and foreign countries.

Recent patents and articles dealing with this substance are as follows:

"Elektrisches Verschmelzen sulfidischer Erze und Hüttenprodukte unmittelbar auf Metall." C. Lehmer. (*Metallurgie*, Aug. 22 and Sept. 8, 1906; 10 pp.) Describes method of reducing molybdenum and molybdenum alloys with other metals.

"A Rapid Method of Determining Molybdenum." J. Darroch and C. A. Meiklejohn. (*Eng. and Min. Journ.*, Nov. 3, 1906.) Description of a rapid method for the determination of molybdenum used by the authors, and the standard solutions employed.

Process of producing molybdenum and its alloys. Frederick M. Becket, Niagara Falls, N. Y., assignor to Electro Metallurgical Company of West Virginia. U. S. pat. 835,052.

MONAZITE.

Monazite is valuable because of its thorium oxide which is present in the mineral not as an essential constituent, but rather as an accidental admixture. The thorium oxide is used in the manufacture of incandescent mantles for gas lighting, and the demand for it for this purpose is increasing. The present sources of supply of monazite are Brazil, the Carolinas in the United States and Ceylon, with small productions from Norway, Russia and Africa. In Ceylon the production of monazite appears to be giving way to thorianite, a newly discovered mineral which contains more thorium oxide than monazite. Methods of mining and concentrating and commercial conditions attending the monazite industry were so well treated in Vol. XIV that it is unnecessary to enlarge upon them here.

In the United States the recent investigations of the black sands of the Pacific coast by the U. S. Geological Survey have demonstrated the presence of considerable amounts of monazite in Western placer sands. The thorium content of these sands is largely uncertain at present, as no definite investigations upon that point have been undertaken.

It is stated that a deposit of monazite sand at Centerville, on the Snake river in Idaho, is to be worked in the spring of 1907 by W. H. Day. In the Carolinas no new developments took place except that there was a tendency to introduce more modern methods of concentrating the original material, which practice is resulting in a higher grade of concentrates, the latter now going from 80 to 90 per cent. monazite, against the former 50 to 75 per cent. This explains the great increase in value per pound in the statistics for 1906.

T. W. Miller, William Teague and D. E. Shade, of Joplin, Mo., who have been operating for monazite in Greenville county, South Carolina, are erecting four mills to increase their product. Until recently they had but one mill, with a capacity of 400 lb. a day. The four now being constructed will have a capacity of about 1000 lb. each per day. The operations are about seven miles from Piedmont, a station on the Columbia and Greenville line of the Southern Railway, and about 14 miles from Greenville.

The accompanying table indicates that up to 1903, the total monazite production of the United States was from North Carolina.

MONAZITE PRODUCTION IN THE UNITED STATES.
(In pounds.)

Year.	United States. (a)			North Carolina. (b)		
	Pounds.	Value.	Per Pound.	Pounds.	Value.	Per Pound.
1897	44,000	$ 1,980	$0.045	44,000	$ 1,980	$0.045
1898	250,776	13,542	0.054	250,776	13,542	0.054
1899	350,000	20,000	0.057	350,000	20,000	0.057
1900	908,000	48,805	0.054	908,000	48,805	0.054
1901	748,736	59,262	0.079	748,736	59,262	0.079
1902	802,000	64,160	0.080	802,000	64,160	0.080
1903	862,000	64,630	0.075	773,000	58,694	0.076
1904	745,999	85,038	0.114	685,999	79,438	0.116
1905	1,352,418	163,908	0.121	894,368	107,324	0.120
1906	846,175	152,312	0.180	697,275	125,510	0.180

(a) Statistics of the U. S. Geological Survey. (b) From "The Mineral Industry in North Carolina during 1905," North Carolina Geological Survey.

The production of monazite in Brazil in 1906 was 4351 metric tons, valued at $480,843, against 4437 tons, valued at $485,184, in 1905.

Industrial Conditions.

In January, 1906, the German "Thorium Convention" which controls the monazite and thorium nitrate industry, decided to kill all competition in thorium nitrate by suddenly dropping the selling price to $6.43 per kg., less 3 per cent. discount. Early in 1894 thorium nitrate was sold at $476 per kg., but dropped in price year by year, being maintained during the period from May, 1904, to January, 1906, however, at $12.61 per kg. The sudden drop spread consternation throughout the gas mantle manufacturing trade. The result has been disastrous to the small thorium works outside the ring with large stocks of high-priced material on hand. Some suspended operations, others went bankrupt, and the "Thorium Convention" was left completely master of the situation.

In February, 1907, the price of thorium nitrate increased 20 per cent. The cause of this sudden elevation in price was the reported decision of the Brazilian Government to levy a heavy export tax on monazite sand. English manufacturers of incandescent mantles, hitherto dependent upon German firms for their supplies, now have hopes that the discoveries of monazite and thorianite in Ceylon and the Transvaal will allow them to avail themselves of an opportunity to obtain an independent supply of the raw material.

The total European demand for monazite sand, including that exported from Europe to America, is stated to be 1200 to 1500 metric tons per year. Brazil alone can supply 10,000 tons easily, which exceeds all possible consumption. However, prospecting for new sources of supply is being actively carried on, and not without success. The result is that the monazites and business may cease to be remunerative; even now,

it is said, the stocks in Europe are sufficient to inundate the markets at any moment.

Bibliography.

The following articles dealing with monazite have appeared in volumes of The Mineral Industry: In Vol. VI an eight-page contribution by L. M. Dennis discusses the occurrences of monazite in the United States, with notes on the quality of the sand occurring in these deposits, and describes in detail various methods of manufacturing incandescent lamps from thorium solutions obtained from the monazite. Vol. VII reviews the status of monazite mining in the United States, with notes on its commercial value and the technology and gives several procedures for the chemical analysis of incandescent mantles. Vol. XI compares the ammonium oxalate, sodium hyposulphite and hydrogen peroxide methods for determining thorium in monazite sand, and describes a new method for the separation of thorium from cerium, lanthanum and didymium. Vol. XIII gives a new procedure for the analysis of monazite sand.

NICKEL AND COBALT.

By John T. Glidden.

The nickel and cobalt produced in the United States in 1906 was derived almost entirely from ore and matte imported from Canada and New Caledonia, the production from domestic ores being insignificant, but the latter statement probably will not remain true for 1907. The smelting of the imported ore and matte is concentrated in the hands of the International Nickel Company, operating the Orford smelter at Bayonne, N. J., and the American Nickel Works at Camden, N. J. Since the control of the nickel production is thus invested in one company, direct statistics of production are not available, but the output for 1906 is estimated at 7150 tons against 6500 tons in 1905.

UNITED STATES IMPORTS AND EXPORTS OF NICKEL AND COBALT.
(In pounds, and tons of 2240 lb.)

Year.	Imports.							Exports.	
	Nickel Ore and Matte.		Nickel Alloys. (a)		Nickel Mnfrs.	Cobalt Oxide.		Nickel. (b)	
	Long Tons.	Value.	Pounds.	Value.	Value.	Pounds.	Value.	Pounds.	Value.
1897..	12,420	$781,483	(c)	....	....	24,771	$34,773	4,255,558	$997,391
1898..	26,826	1,534,262	(c)	....	....	33,731	49,245	5,657,618	1,359,609
1899..	19,857	1,216,253	(c)	....	....	46,791	68,847	5,004,377	1,151,923
1900..	25,670	1,183,884	455,188	$139,786	....	54,073	88,651	5,869,906	1,382,727
1901..	52,111	1,637,166	635,697	209,956	$2,498	71,969	134,208	5,869,655	1,521,271
1902..	14,817	1,156,372	752,630	251,149	30,128	79,984	151,115	3,228,607	925,579
1903..	15,936	1,285,935	521,344	170,670	37,284	73,350	145,264	2,414,499	703,550
1904..	8,548	915,470	589,555	203,071	2,950	42,352	86,925	7,519,206	2,130,933
1905..	13,451	1,626,920	941,966	331,920	3,291	70,048	139,377	9,550,918	2,894,700
1906..	15,156	1,816,631	210,000	77,373	8,963	41,084	83,167	10,620,410	3,493,643

(a) Includes nickel, nickel oxide, and alloy of any kind in which nickel is the material of chief value, in ingots, bars and sheets. (b) Comprises domestic nickel, nickel oxide and matte. (c) Not separately enumerated; included in "Nickel Ore and Matte."

Market.—The nickel market during 1906 was without special interest. The price rose a few cents per pound in the fall in sympathy with the larger increases in the values of other metals. Following is the course of average prices by months in cents per pound at New York for large lots: January to May inclusive, 43½c.; June, 43c.; July and August, 42½c.; September, 45½c.: October to January, 47½c. Average price for the year, 44.46c. Small lots were 5c. per lb. higher.

On January 8, 1907, the United States Circuit Court, southern district of New York, affirmed the decision of the Board of General Appraisers classifying nickel anodes entered for import as manufactures of nickel and assessing a duty of 45 per cent. upon them in accordance with para-

graph 193 of the present tariff act. The contention of the importers that the duty should have been 6c. per lb. as provided for by paragraph 185 was denied, as the court held that nickel anodes are not "nickel in pigs, ingots, bars or sheets" within the meaning of the act.

Nickel and Cobalt in the United States.

Alaska.—Prospecting for nickel ore has been carried on in Alaska along the shores of Prince William Sound. The country rock of the region is diorite, in which there is a zone impregnated with pyrrhotite, which also occurs in a few pegmatite veins. So far it has not been definitely determined just how much nickel there is in the pyrrhotite and no development has taken place.

North Carolina.—At Webster, in Jackson county, there is a nickel property owned by W. S. Adams and associates. The United States Chrome & Nickel Company, which was organized to exploit certain nickel and cobalt deposits in this State decided to abandon its claim after doing a small amount of development work.

Missouri.—In this State the Mine La Motte, in Madison county produced for many years matte containing nickel and cobalt and for a time was the only producer of domestic nickel in the United States. During 1906 it made but a very small output of nickel matte which was not worked up.

At Fredericktown, about six miles southeast from Mine la Motte, the North American Lead Company has an extensive deposit of mixed sulphides of copper and iron which contains nickel and cobalt, and which was neglected for a long time. The concentrated ore averages 6 per cent. copper and 3 per cent. nickel plus cobalt. During 1906 the company constructed a special plant for the smelting of the ore, consisting of a smelter and an electrolytic refining department. Early in 1907 the furnace began the production of 5000 lb. of blister copper daily that awaits the opening of the electrolytic department which will probably be completed in June, 1907.

Oregon.—Deposits of nickel ore in this State have been known for more than 25 years in the region around Piney Mountain, Douglas county. These deposits are similar in mineral content, in modes of occurrence, and in their association to the deposits of New Caledonia, whose mines are the second largest producers in the world. It was hoped, therefore, that development would prove the presence of nickel mineral in sufficient quantity for economic production, but this hope is not realized as yet.

Cobalt ore appears to be of more importance than nickel ore in this State. The Standard Consolidated Mining Company in the Quartzburg district of Grant county is operating a 100-ton mill on ore which carries cobalt, gold and copper. It is stated that T. A. Edison has arranged to

purchase two cars of cobalt concentrate per month hereafter, and one such carload has already been shipped in 1907. The occurrence of cobalt at this mine is unique and promises to be of considerable commercial importance.

Nickel and Cobalt Mining in Foreign Countries.

Canada.—The nickel mines of the Sudbury district were more active in 1906 than during any previous year. High-water mark in the yield of nickel was reached in 1905, when the production was 9503 tons; for 1906 the output was 10,750 tons. The chief producers are the Canadian Copper Company and the Mond Nickel Company. The production of copper, which comes mainly from the nickel-copper ores of Sudbury, was about 5200 tons in 1906, as against 4525 tons in 1905.

PRODUCTION, EXPORTS AND IMPORTS OF NICKEL IN CANADA. (*a*)

Year	Production.		Exports.		Imports.
	Pounds. (*b*)	Value. (*c*)	Pounds. (*d*)	Value. (*e*)	
1900	7,080,227	$3,327,707	13,493,239	$1,040,498	$6,988
1901	9,189,047	4,594,523	9,537,558	958,365	12,029
1902	10,693,410	5,025,903	3,883,264	834,513	15,448
1903	12,505,510	5,002,204	9,032,554	878,159	26,177
1904	10,547,883	4,219,153	14,229,973	1,237,307	16,330
1905	18,876,315	7,550,526	11,970,557	1,185,056	19,076
1906	21,490,955	8,948,834	23,959,841	2,166,936	(*f*)15,808

(*a*) Statistics for production and imports cover calendar years, and are taken from the Annual Reports of the Geological Survey of Canada. Figures for exports cover the fiscal years ending June 30, and are taken from the Statistical Year Book. (*b*) Pounds metallic nickel contained in copper and nickel matte exported. (*c*) On the basis of refined nickel at New York, from the *Engineering and Mining Journal* average annual quotations. (*d*) Pounds of nickel contained in ore, matte or speiss. (*e*) Spot value, to the producer, of the exported material; the variety of stages at which the material is shipped, as well as the different periods of time covered, lead to the apparent discrepancy in value when it is known that practically the entire production is exported. (*f*) Anodes only.

ONTARIO NICKEL STATISTICS.
(In tons of 2000 lb.)

Schedule.	1900	1901	1902	1903	1904	1905	1906
Ore raised	216,695	326,945	269,538	152,940	203,388	277,766	343,814
Ore smelted	211,969	270,380	233,388	220,937	102,844	251,421	340,059
Per cent. nickel	1.67	....	2.55	3.16	4.58	(*b*)	(*b*)
Per cent. copper	1.59	....	1.78	1.81	2.41	(*b*)	(*b*)
Ordinary matte	23,336	29,588	24,691	30,416	19,123	17,388	20,364
Bessemerized	12	15,546	13,332	14,419	6,926		
Nickel content	3,540	4,441	5,945	6,998	4,743	9,438	10,745
Copper content	3,364	4,197	4,066	4,005	2,163	4,386	5,265
Value of nickel (*a*)	$756,626	$1,859,970	$2,210,961	$2,499,068	$1,516,747	$4,019,814	$4,629,011
Value of copper (*a*)	319,681	589,080	616,763	583,646	297,126		
Wages paid	728,946	1,045,889	835,050	746,147	570,901	(*b*)	(*b*)
Men employed	1,144	2,284	1,445	1,277	1,063	(*b*)	(*b*)

Note.—The quantities reported in 1901, 1902 and 1903 under "bessemerized matte" include both bessemerized matte and high-grade matte, the former being the product of the Mond Nickel Company's works and the latter of the Ontario Smelting Works, which re-treat the low-grade matte produced by the Canadian Copper Company. (*a*) Value based on nickel and copper in matte and not on refined metals. (*b*) Not available.

The Mond Nickel Company, at Victoria Mines, is diverting the waterway at the smelter and cutting a channel through the rock there so that the

water-bed will be removed from the Canadian Pacific Railway track. This company declared a dividend of 18 per cent. on the last year's operation.

The Crean Hill nickel-copper mine, owned by the Canadian Copper Company, of Copper Cliff, is now being operated and ore is being daily hauled over the company's new track to Victoria mine, thence to the company's smelter at Copper Cliff, over the Canadian Pacific.

The ores from the Cobalt district are extremely varied, both in the metal contents and in the character of the gangue. Below are some typical analyses of ores from different parts of the district.[1]

COMPOSITION OF ORES OF COBALT DISTRICT.

Ag. Oz. per Ton.	SiO_2 Per Cent.	Fe Per Cent.	CaO Per Cent.	As Per Cent.	Ni & Co Per Cent.
300	4.0	4.0	5.0	55.0	14.1
350	53.0	12.4	2.2	5.7	4.1
400	46.1	5.8	4.0	6.2	....
3,000	10.0	6.0	5.0	41.0	....
3,000	28.3	....	21.7	2.4	....
5,600	8.3	....	10.3	27.8	....

According to the report of the Nipissing Mines Company, giving the first official statement made by any company operating in the Cobalt area, the production during a period of 15 months since the opening of the property and ending Jan. 13, 1906, was as follows: Ore shipped, net weight, 1,728,695 ℔.; ore shipped, dry weight, 1,677,446 ℔. Products returned: silver, 1,360,492 oz. ($865,553); nickel, 35,444 ℔. ($836); cobalt, 145,046 ℔. ($107,700); arsenic 78,526 ℔. ($381); total value, $974,470. The shipments of ore thus far made show an average assay of 2300 oz. in silver, 8 per cent. cobalt and 5 per cent. nickel. This ore has been produced from veins averaging between 2 and 9 in. in width. Several of the workings have reached a depth of over 50 ft., and retain their value and width at the bottom. On an adjoining property a shaft has been sunk on ore to a depth of over 200 ft., showing a permanent vein and continued values. Forty-eight veins have been discovered to date on the Nipissing property, although only a portion of them have been opened.

The report of the Cobalt Silver Queen Company for the year ended March 31, 1907, states that 442,141 ℔. of ore were shipped and 109,273 ℔. remain on hand. The shaft is down 145 ft. and laterals extend east and west at depths of 75 and 125 ft. respectively. Nearly 900 ft. of underground work brought out ore worth $700,000 and $120,000 was paid in dividends. Recently the Orford Copper Company has entered the market as a purchaser of cobalt ore and the Silver Queen expects to derive revenue from the cobalt contents of the ore. The ore is graded into A 1, No. 1

[1]As reported by F. F. Colcord.

and No. 2, the last averaging from 250 to 500 oz. of silver per ton. The vein continues to maintain its width and value; no estimate of the total extent has been made.

A report issued by Dr. Eugene Haanel, Dominion Superintendent of Mines, on the "Present and Prospective Output of the Cobalt District," embodies the results of a recent investigation. The ore from the Cobalt district has been shipped to the American Smelting and Refining Company, which has works at Perth Amboy, N. J.; the Balbach Smelting and Refining Company, at Newark, N. J., and the Orford Copper Company, at Copper Cliff, Ont. The former two have no process for saving the cobalt, nickel or arsenic contents of the ore. The American Smelting and Refining Company charges $10 per ton dry weight for treatment, and pays for 94 per cent. of the silver contents, at New York quotations, as given on date of assay. The Balbach Smelting and Refining Company receives silver ore if not too high in arsenic, the charges for smelting being arranged on a sliding scale, graded from $6 per ton on ore running 400 oz. per ton, to $2 per ton on ore above 700 oz. to 800 oz. per ton. On higher grade than this there is no smelting charge. The company pays for 93 per cent. of the silver contents at New York market price. Both companies accept the sampling of Ledoux & Co. as a basis of settlement.

The Orford Copper Company pays both for silver and cobalt, but not for nickel or arsenic contents. The smelting is done at Copper Cliff, Ont., resulting in a partial silver recovery together with a high silver-bearing speiss which is shipped to Camden, N. J., for further treatment. The payment for silver is on a graduated scale from 94 per cent. of the New York official rate on the silver content of ore carrying 4000 oz. per ton or over, to 80 per cent. on ore of between 150 and 300 oz. to the ton. Cobalt is paid for on a similar plan, the rate being $30 per ton of ore containing 12 per cent. cobalt and over; $20 per ton of ore containing 8 per cent. cobalt and over; and $10 per ton of ore containing 6 per cent. cobalt and over.

(By Thomas W. Gibson.)—In the Province of Ontario the chief interest centered in the silver ores of the Cobalt camp. The early promise of these narrow, but wonderfully rich veins has so far been more than realized. Nothing was shipped from Cobalt in 1903; in 1904 the output was 206,875 oz. of silver; in 1905 it rose to 2,451,356; in 1906 the product was 5,357,830 oz. The prospects for 1907 are that the production will be still greater.

The first shipments in 1904 were native silver nuggets and plates found in the upper zones of the deposits; the 158 tons sent out averaged 1309 oz. per ton; in 1905 the consignments were 2144 tons and the average contents 1143 oz., while in 1906 the shipments amounted to about 5000 tons, containing an average of about 1100 oz. of silver to the ton. Some of the consignments were of phenomenal richness; one carload from the O'Brien

mine in 1905 brought about $65,000, while one from the La Rose mine, in 1896, yielded a return of $110,000. Ores carrying up to 200 or 300 oz. of silver per ton, which in other camps would be reckoned of high quality, are in Cobalt spoken of as "low grade."

The producing mines are as follows: Nipissing, La Rose, Coniagas, Jacobs, Trethewey, Drummond, Foster, University, McKinley-Darragh-Savage, Buffalo, Cobalt Silver Queen, O'Brien, Violet, Right-of-Way, Lawson, Hargrave, Watts and Allen, and Green-Meehan. In addition to these, several other properties are about to enter the productive stage. The rich territory is fairly well distributed. Most of the mining companies hold one, two or not more than three 40-acre claims, so that there is no great likelihood of any one concern's obtaining absolute control. One or two of the companies which were early on the ground are, however, an exception to this rule, the principal one being the Nipissing Mining Company, whose holdings aggregate 846 acres.

The prospectors seem to have pretty well defined the boundaries of the productive area of Cobalt, and have shown it to be limited to certain portions of the eastern half of Coleman township and portions of the adjoining townships of Bucke and Lorrain. Outside of this territory, comprising perhaps 10 or 12 square miles, cobalt bloom and smaltite are found at many widely separated points, but silver is low or absent altogether. The Cobalt orebodies carry considerable cobalt, nickel and arsenic. The last two constituents are at present valueless to the mine owner, and he gets nothing for cobalt unless it is in excess of 6 per cent. Ore buyers settle on the basis of 93 per cent. of the value of the silver, if in excess of 2000 oz. per ton. A sliding scale governs payments for ore of lower grade. Up to the present, most of the ore has been shipped to Newark and Perth Amboy for treatment, but several reduction plants are in course of construction in Ontario, some of which will shortly be ready to begin operations. The Copper Cliff refinery has already treated considerable ore.

New Caledonia.—New Caledonia is a mountainous island 400 km. long by 40 km. wide, lying in the Pacific Ocean, about 1600 km. to the northeast of Australia. Its mountain chains are almost continuous, and reach an elevation of 1000 m. The three principal rock formations are Paleozoic, then a group of more recent sedimentary rocks, and finally an eruptive serpentinous rock which covers about two-fifths of the island. It is in this latter formation that the ore bodies are found.

The nickel deposits are scattered irregularly through the serpentine area, and yield an ore consisting apparently of garnierite, or nickel silicate, although it is actually a compound silicate of nickel and magnesia. It is found in masses, often widely extended, but always of shallow depth, which are, without exception, mined in open quarries. The average grade of the ore, as shipped, ranges between 6 and 8 per cent. nickel.

The cobalt deposits consist of irregular impregnations of cobalt oxide in ferruginous clays, derived by alteration of the serpentine rock. Nickel and manganese oxides are associated with it, forming a mixture with a bluish-black color, and an earthy appearance, going under the name of asbolite. The cobalt deposits are not extensive, and permit only small-scale working, which is undertaken by individual miners. Owing to the high price of cobalt these miners are able to make a living from an ore carrying between 4 and 6 per cent. cobalt. This is sold at Noumea for 250 fr. per ton. There are about 30 producing mines.[1]

Large contracts for New Caledonian nickel ore were made during 1906 for future delivery, at a somewhat higher price and under better conditions than were in force for then existing contracts. The increase appears to be necessary as many of the nickel mines near the coast are nearly exhausted and new deposits in the interior must be opened. The old price of 60 centimes per kilogram of metal contained in the ore is too disadvantageous except for very large contracts lasting for a long period. Such a contract for 60,000 tons of 7 per cent. ore was recently closed, delivery to occupy three years, at the above price. It is reported that M. Brini, who holds this contract and is a large operator, has acquired the Téné group of mines which belonged to the Prévost estate and that he is to open immediately the St. Vallecty mine which is situated near the source of the Téné river at an altitude of 380 m. A railroad 18 km. long will be constructed to transport the ore to Guarou, the shipping port.

SHIPMENTS OF NICKEL AND COBALT ORES FROM NEW CALEDONIA. (*a*)
(In metric tons.)

	1899	1900	1901	1902	1903	1904	1905	1906
Nickel ore	103,908	100,319	133,676	129,653	77,360	98,655	125,289	130,688
Cobalt ore	3,200	2,400	3,110	7,512	8,292	8,961	7,919	2,487

(*a*) Reported by *Le Bulletin du Commerce*, Noumea.

Progress in the Metallurgy of Nickel and Cobalt.

Electro-Metallurgical Treatment of Nickel-Copper Matte.—W. McA. Johnson proposes a new solution of the old problem of treating nickel-copper matte.[2] The matte may contain 39 per cent. of nickel, 39 per cent. of copper, 1 per cent. each of iron and cobalt, and 20 per cent. of sulphur, together with small proportions of platinum and palladium. This matte is crushed to 80-mesh, placed in leaching vats provided with agitators, and subjected therein to the action of a heated solution of hydrochloric acid of about 10 per cent. concentration, at a temperature approximating 100 deg. C. Means are provided for conducting away the sul-

[1]From a paper read by M. Dupay, before the Société l'Industrie Minérale, August, 1906.
[2]U. S. pat. 825,056, July 3, 1906.

phuretted hydrogen formed by reaction of the acid upon the sulphides of the matte. The solution thus obtained contains nickel, cobalt and iron as chlorides. It is freed from cobalt and iron by treatment with chlorine and sodium carbonate or by the action of hypo-chlorites. The cobalt is recovered from the precipitate as oxide (Co_3O_4). The residual nickel-chloride solution is concentrated, if necessary, brought to neutrality or to a faintly acid reaction, and electrolyzed with insoluble anodes, the solution being maintained during the electrolysis at a temperature of about 65 deg. C. Nickel is thereby precipitated in reguline form and the evolved chlorine is utilized for the manufacture of bleaching powder or otherwise. The residue from the acid treatment may contain 75 per cent. of copper, 5 per cent. of nickel and some 20 per cent. of sulphur. This residue is subjected to an oxidizing fusion in a reverberatory furnace, and thereby converted into blister copper, the nickel being slagged off and retreated in a small blast furnace. The blister copper is poured hot into refining-furnaces, and therein brought up to metal carrying 0.3 to 1 per cent. of nickel. This is then cast into anodes and electrolytically refined.

Bessemerizing with Blast Rich in Oxygen.—R. Hesse[1] gives the result of experiments made in Prof. Borcher's laboratory at Aachen, to convert nickel matte into nickel by blowing in air rich in oxygen. The contents of oxygen in the air were run up to 63 per cent. He found that even when the temperature is increased considerably, it is not possible to get metallic nickel by the action of NiO on NiS. The experiments have, however, shown that an action of NiO on NiS takes place, notwithstanding statements to the contrary. It begins only at temperatures exceeding 1400 deg. C., but it was not possible to obtain a quantitative result according to the chemical reaction $NiS + 2NiO = 3Ni + SO_2$. In regard to converting nickel matte by blast containing oxygen it was established by the experiments that it is not possible to perform that operation, on account of the oxidation of the nickel, which in the presence of silica is slagged off as nickel oxide.

Lime Roasting.—A. C. de Jongh[2] experimented with the roasting of nickeliferous matte by the Huntington-Heberlein process. He worked upon the theory that the true function of the lime in this particular instance is to accelerate the roasting and that its influence would be exerted in the first part of the process. In other words, he proposed to obtain a mechanical mixture of calcium sulphate and absolutely sweet-roasted matte. To test this theory the following experiment was performed: An artificial matte, consisting chiefly of NiS and Cu_2S, besides containing a small amount of iron, was finely pulverized (60-mesh) and subjected to roasting. For this purpose 100 grams of the substance were spread

[1]*Metallurgie*, Nos. 10 and 11, 1906 through *Electrochem, and Met. Ind.*, (Aug., 1906).
[2]*Eng. and Min. Journ.*, April 28, 1906.

out upon a Battersea roasting dish and placed in a glowing muffle, with free access of air. The material was continuously stirred, as far as feasible, with a small iron rod, until it became pasty. Then it was taken out, allowed to cool and pulverized again. A part was reserved and the remainder submitted to heat a second time, until no sulphurous fumes were perceptible. Analysis showed 1.54 per cent. of sulphur remaining, a rather fair figure under the unfavorable circumstances. Then the other part, or half-roasted matte, was mixed with pure, precipitated $CaCO_3$ and treated in the same manner. No sulphurous gas was observed to come off, but after leaching out the CaO and $CaSO_4$, which had been formed, the residue retained only 0.054 per cent. of S. Practically a completely sweet roast had been attained.

Smelting Cobalt Ores.—Hiram W. Hixon.[1] suggests a system of smelting ores from the Cobalt district based on well established principles, which is herewith submitted:

1. The ores should be sorted and the metallic silver separated for direct smelting by cupellation with lead or melted in crucibles and poured into bars to be sold to refiners.

2. The ore should be ground in a dust-proof ball mill to such a fineness that it can be accurately sampled.

3. The ground ore should be roasted in a suitable furnace to expel the arsenic and sulphur.

4. The gases from this roaster should pass through a long sheet-iron dust chamber in order to cool them, and then be passed through a bag house, where all the dust and volatilized arsenic will be recovered.

5. The roasted ore should then be briquetted with a strong bond of red clay and lime water and dried or frozen, after which it should be smelted in a blast furnace with about an equal weight of galena or lead concentrates containing about 50 to 60 per cent. lead. The necessary iron and lime would have to be added to the charge to produce a slag of the proper composition.

The sulphur for matte formation would be supplied by the galena and the arsenic remaining in the ore will form some speiss.

The base bullion resulting from this smelting could be sold to lead refiners, or refined to separate the silver.

The matte and speiss will contain all the nickel and cobalt that can be recovered from the first smelting and also some lead and silver. This matte and speiss should be accumulated until sufficient for a special run and then ground in the ball mill, roasted and smelted separately with more lead to free it from silver. The resulting bullion could be sold or refined.

The second matte should be run into a converter and converted to free

[1]Paper for discussion at the Toronto meeting, 1907, of the Canadian Mining Institute.

it from iron. In the process of converting, the lead and most of the cobalt will oxidize at the same time as the iron and go into the converter slag, except that portion of the lead which is volatilized along with the remaining arsenic. These will pass off with the flue gases and should for safety be passed through the bag house, as they are both poisonous.

The converter slag will, therefore, contain the greater portion of the cobalt as oxide, and in order to recover, it should be resmelted in a separate blast furnace with a portion of the briquetted flue dust from the roasting of the ore.

The flue dust highest in arsenic should be used, as the recovery of the cobalt would depend upon the formation of a cobalt speiss or arsenide of cobalt.

This would be separated from the slag in the same manner as the matte and its further treatment for cobalt would be by chemical methods after roasting to expel the arsenic, which should be recovered in the bag house in the same manner as first described.

The flue dust containing arsenic would have to be roasted separately to sublime the arsenic which should be recovered in a separate flue and bag house provided for that purpose.

The arsenic would be marketed as white arsenic.

By the scheme outlined, the silver would be produced either as silver or in base lead bullion.

The nickel would be produced as high-grade nickel matte containing some cobalt suitable for further refining by the Orford, Mond or other processes; the cobalt would be produced as cobalt speiss, and the arsenic as white arsenic.

As regards the percentage of metals that could be recovered by all these various smeltings and resmeltings, it is not possible to make any statement based on actual results.

The smelters who have bought these ores have not seen fit to give out any information, but the fact that they have allowed nothing for nickel, cobalt or arsenic is significant. To make even an attempt at a partial recovery of each of the metals is a long, tedious process and the most complicated from a metallurgical standpoint that has been presented for solution.

For the economical working of this plan the smelter should be located at North bay, where lead concentrates from the western mines can meet the ores. A working capital of at least $500,000 would be necessary and the plant would cost at least $200,000.

OCHER AND IRON OXIDE PIGMENTS.

The leading states in the production of these mineral pigments are Pennsylvania, Georgia and California, with smaller amounts coming from New York, Vermont, Iowa and Virginia. The Cartersville district in Georgia will doubtless become of increasing importance in this industry, on account of the favorable development of Clinton oolitic hematite beds there, from which excellent pigment can be made. In Tennessee the region around White Oak Mountain is a promising locality. The mines in this section are situated near Coltewah, 15 miles east of Chattanooga.

PRODUCTION OF OCHER, UMBER AND SIENNA IN THE UNITED STATES.(*a*)
(In tons of 2000 lb.)

Year	Ocher.		Umber.		Sienna.		Total.	
	Amount.	Value.	Amount.	Value.	Amount.	Value.	Amount.	Value.
1901	16,711	$177,799	759	$11,326	305	$9,304	17,775	$198,429
1902	16,565	145,708	480	11,230	189	4,316	17,234	161,254
1903	12,524	111,625	(*b*)666	15,367	....	....	13,190	126,992
1904	16,826	110,602	(*b*)522	12,960	....	....	17,348	123,562
1905	13,402	126,351	(*b*)689	17,004	....	....	14,091	143,355

(*a*) Statistics of the U. S. Geological Survey.

IMPORTS OF OCHER, UMBER AND SIENNA INTO THE UNITED STATES.

Year.	Ocher of All Kinds.						Umber. (*a*)		Sienna.			
	Dry.		Ground in Oil.		Total.				Dry		Ground in Oil.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1897	7,720,075	$59,272	20,123	$1,000	7,740,198	$60,272	1,447,889	$14,479	580,468	$12,340	7,058	$481
1898	*a*5,898,720	46,571	31,460	1,546	5,930,180	48,117	1,123,079	9,051	544,713	11,451	4,008	280
1899	9,765,616	72,825	14,881	756	9,780,497	73,581	1,739,036	13,336	758,691	14,242	6,484	492
1900	8,449,252	57,342	19,167	1,019	8,468,419	58,361	1,703,256	11,862	796,534	14,912	6,335	495
1901	8,546,691	83,196	16,738	918	8,563,429	84,114	1,465,431	12,510	1,106,553	18,294	13,861	1,004
1902	9,988,518	107,285	19,668	1,013	10,008,186	108,298	1,894,425	16,133	1,535,377	27,310	5,921	494
1903	9,839,989	99,269	20,335	1,178	9,860,334	100,447	2,169,570	18,272	1,873,532	28,447	1,387	123
1904	9,430,916	93,137	12,756	583	9,443,672	93,720	2,274,926	20,511	1,286,301	22,118	5,770	396
1905	10,616,496	91,673	15,985	880	10,632,481	92,553	2,587,284	21,224	1,737,908	26,097	2,886	227

(*a*) In 1898, 4,608 lb. ($323) ground in oil and 1,118,471 lb. ($8,728) crude, powdered, washed or pulverized; in 1899, 4,849 lb. ($300) ground in oil and 1,734,187 lb. ($13,036) dry, crude, powdered, washed or pulverized; in 1900, 11,653 lb. ($723) ground in oil, and 1,691,603 lb. ($11,139) dry, crude, powdered, washed or pulverized; in 1901, 3,184 lb. ($173) ground in oil and 1,562,248 lb. ($12,337) dry, crude, powdered, washed or pulverized; in 1902, 11,999 lb. ($689) ground in oil and 1,882,426 lb. ($15,444) dry, crude, powdered, washed or pulverized; in 1903, 9,656 lb. ground in oil and 2,159,914 lb. dry, crude, powdered, washed or pulverized; in 1904, 13,133 lb. ground in oil and 2,261,793 lb. dry, crude, powdered, washed or pulverized; in 1905, 6,783 lb. ground in oil and 2,580,501 lb. dry, crude, powdered, washed or pulverized.

Market and Prices.—The ocher market during 1906 was without interest, the quotations being $8.50@9 per short ton throughout the year for

American common, and $16 per ton for best. In regard to sienna and umber, prices during 1906 increased to a considerable extent, which is more particularly due to the ever increasing expense of output and manufacture. The production of sienna was remarkably inferior to that of the previous year, and the superior grades are becoming more and more scarce. This accounts for the higher prices now asked for them. Demand, on the other hand, has remained moderate. There are strong probabilities that new increases will take place during 1907, but they are not expected to be considerable, seeing that for this line of colors the demand would certainly cease if the prices asked were too high.

Bibliography.

Statistics of production of ocher and iron pigments with notes on industrial conditions may be found in Vols. VII-XI, inclusive, of The Mineral Industry, supplemented by the following special articles. In Vol. VII, notes upon the production of ocher, sienna and umber, in Italy, and a contribution dealing with the durability of iron pigments and methods of testing them. In Vol. X there is a brief description of the dry and the wet processes of manufacturing these pigments and Venetian red.

PETROLEUM.

By Harold C. George.

Probably the most remarkable feature of the petroleum industry in the United States is the continuous decline of the Appalachian Field, which produces practically all of the high-grade petroleum of the United States, and also the rapid decline of the Texas and Louisiana fields. The California fields showed a marked decline in 1906, over 5,000,000 bbl., but unlike the

PRODUCTION OF CRUDE PETROLEUM IN THE UNITED STATES.
(In barrels of 42 gal.)

Field.	1900	1901	1902	1903	1904	1905	1906
California (a)	4,250,000	8,786,330	13,973,500	24,337,828	28,476,025	35,671,000	30,538,000
Colorado	525,000	460,520	396,901	483,925	(b) 501,763	(e) 550,000	600,000
Gulf: Texas	800,000	4,393,660	18,083,658	17,955,572	21,672,111	30,354,263	12,666,000
Gulf: Louisiana			548,617	917,771	6,611,419	9,672,015	7,100,000
Illinois	......	......	......	......	......	......	4,900,000
Lima: Indiana	4,329,950	5,757,086	7,535,561	9,177,122	10,744,849	22,102,108	25,680,000
Lima: Ohio	16,407,704	16,176,293	15,877,730	14,893,853	13,350,060		
Mid-Continental (c)	65,000	179,150	359,123	1,157,110	5,617,527	12,000,000	21,929,905
Ken.-Tennessee	(f)	(f)	(f)	(f)	998,284	(e) 1,200,000	1,000,000
Pennsylvania (d)	35,540,965	33,618,180	32,018,787	29,897,815	(b)30,410,183	28,324,324	27,345,600
Wyoming	7,200	5,400	6,253	8,960	11,542	(e) 12,500	13,000
Others	30,000	2,585	957	3,000	2,572	(e) 3,000	4,000
Total	61,955,819	69,379,204	87,801,087	98,832,956	118,396,335	139,889,210	131,771,505

(a) Reported by the California Producers' Association. (b) Statistics of the U. S. Geological Survey. (c) Kansas, Indian Territory and Oklahoma. (d) Pennsylvania, New York, West Virginia, Eastern Ohio, and, until 1904, Kentucky and Tennessee. (e) Estimated. (f) Included in Pennsylvania.

EXPORTS OF MINERAL OILS FROM THE UNITED STATES. (In gallons.)
(1 1000 in quantities and values) (a)

Year.	Crude Petroleum.		Naphthas.		Illuminating.		Lubricating and Paraffin		Residuum.		Totals.	
1897	121,864	$5,044	13,704	$1,020	804,446	$46,876	52,659	$6,732	(b)12,247	335	1,004,920	60,007
1898	120,436	5,016	17,258	1,071	764,823	38,895	65,526	7,626	(b)30,436	815	998,479	53,423
1899	117,690	5,958	18,210	1,597	733,382	49,172	71,116	8,658	(b)21,609	658	962,007	66,043
1900	133,161	7,341	18,570	1,681	739,163	54,693	71,211	9,933	(b)19,750	845	986.855	74,493
1901	127,008	6,038	21,685	1,742	827,479	53,491	75,306	10.260	(b)27,596	1,255	1,079,059	72,786
1902	145,234	6,331	19,683	1,393	778,801	49,079	82,200	10,872	(b)38,316	922	1,064,234	68,597
1903	126,512	6,782	12,973	1,519	691,837	51 356	95,622	12,690	(b) 9,753	282	936,697	72,629
1904	111,176	6,351	24,989	2,322	761,358	58,384	89,738	12,389	34 904	1,174	1,022,165	80,620
1905	126,185	6,086	28,420	2,215	881,450	54,901	113,730	14,312	70,728	2,128	1,220,513	79,641
1906	148,045	7,731	27,545	2,488	878,284	54,858	151,269	18,690	64,645	1,971	1,269,788	85,738

(a) In addition to the above, the following quantities of paraffin and paraffin wax were exported; 1896, 112,517 lb. ($4,563); 1897, 136,069 lb. ($5,284); 1898, 166,317 lb. ($6,363); 1899, 181,861 lb. ($7,650); 1900, 157,108 lb. ($8,186); 1901, 151,695 lb. ($7,960); 1902, 175,268 lb. ($8,398); 1903. 204,120 lb. ($9 596); 1904, 174,582 lb. ($8,273); 1905. 160,836 lb. ($7,873); 1906, 173,504 lb. ($8,463) (b) Reported in barrels of 42 gallons.

Appalachian, Louisiana and Texas fields this decline was not due to a decrease in the available oil supply, but was caused by a curtailment of the supply to meet the demand. The Mid-Continental field showed a

remarkable production in 1906, almost 22,000,000 bbl., an increase of 10,000,000 bbl. over 1905. The production of petroleum in Illinois also increased very rapidly in 1906. In five months the production increased from 10,000 to 30,000 bbl. a day, and the total production for the year was 4,900,000 bbl.

PETROLEUM OUTPUT OF THE WORLD.
(In metric tons.)

	1902	1903	1904	1905	1906
United States	11,906,000	13,402,000	16,055,000	18,969,000	17,862,000
Russia	10,950,000	10,320,000	10,600,000	6,500,000	7,249,000
Sumatra, Java and Borneo	732,000	830,000	1,000,000	1,200,000	1.350,000
Galicia	576,000	713,800	827,000	800,000	760,000
Rumania	310,000	384,300	455,000	568,000	887,000
India	209,000	325,400	404,000	456,000	560,000
Other countries	270,000	250,000	250,000	350,000	(*b*) 350,000
Total	24,953,000	26,225,500	29,591,000	28,843,000	29,018,000

(*a*) In the above table the statistics for the United States are computed from the production reported in barrels as given in the first table of this article. As a gallon of the crude petroleum found in the United States varies in weight from 6.41 to 7.83 lb., the oil in a barrel varies from 269.22 to 328.86 lb. The arithmetical mean of these figures is 299.04 lb., which figure has been used as a factor in converting the output stated in barrels into metric tons. This is not strictly correct, because in the period from 1900 to 1906 the proportion of petroleum production of various gravities has altered materially, especially because of the largely increased production in California, Texas and Louisiana. However, the method adopted is as near an aproximation as can be made at this time. The other figures in the table are assumed as reported by the "Petroleum Review," except in the case of 1906, for which year the figures for the Dutch East Indies Galicia, Rumania and India are as reported by the "Moniteur de Petrole Roumaine."

(*b*) In order to arrive at an approximate total, the production has been estimated, taking the same figures as for 1905.

The growth of the petroleum industry in Canada, Mexico, Trinidad, Peru, New Zealand, China, Japan and Africa, and the numerous discoveries of petroleum indications in many other countries, make the future of the industry assured for years to come.

Review of the Domestic Oil Fields.

The Appalachian Oil Fields.

Neither the increased demand for high grade petroleum nor the advance in price over 1905 seems to have been a sufficient stimulus to maintain production in the Appalachian fields during 1906 at the level of the previous year. The Appalachian oil-fields, which produce the high grade petroleum of the United States, include the oil-field of western New York, all the fields of Pennsylvania in the various sands, the field in southeastern Ohio in the Berea grit, and the fields of West Virginia, Tennessee and Kentucky.

The gradual decrease in the production of the fields producing high grade petroleum is attributed not so much to the natural decline in the production of the old wells as to the failure of West Virginia and southeastern Ohio to produce wells of the gusher type. Few large wells have been struck. The oil territory of western New York and northern Pennsylvania seems to produce each year fewer wells and fewer attractions in the petroleum industry.

Western New York and Pennsylvania.—The old wells in the Allegheny field in New York, and in the fields of Bradford and Warren counties in Pennsylvania, average about 0.25 bbl. per day, while the new wells in the same fields average only about 3 bbl. per day. This indicates that these fields are practically things of the past. Very little undrilled territory is left, and operations are confined mostly to improvements of methods. Many of the old wells are being abandoned and the material secured from them is being used to equip new wells drilled between the locations of the old ones. By this method of procedure many prospects which were about to be abandoned have been put on a paying basis.

In the Venango-Clarion county pools of Pennsylvania, which include the new developments in Jefferson and Mercer counties, there is still much territory to be tested. Mercer county has afforded some good wells regularly; and Jefferson county, which has not been thoroughly developed, may produce some new pools in the near future.

In Venango county, Pennsylvania, in the old Bullion district made famous by the "Big Injun" strike of 30 years ago, one of the largest gushers the State has seen in years was struck in October. This well started off at 1500 bbl. per day. The pool in Butler county which created so much excitement in the fall of 1905 experienced a decline as rapid as its growth. No more large wells have been struck, and the gushers of 1905 have almost ceased to be producers. The new wells in Butler and Armstrong counties average about 5 bbls. per day. The Burgettstown field in Washington county, Pennsylvania, produced some excellent wells in 1906. Six were completed in March which averaged 45 bbl. apiece, and 20 in April which averaged 30 bbl.

The fields of Beaver and Allegheny counties of Pennsylvania produced nothing but small wells and dusters during 1906. It has been thoroughly tested and its limits defined; the territory offers nothing to encourage renewed activity. The one field which, in its producing status, is the exception in Pennsylvania oil territory is the new one in East Finley township, Washington county. All the other fields of the State show a steady decline without any promise of renewed activities. Nearly one-quarter of the wells completed in Pennsylvania in 1906 were dusters, and the average production of the others was very low.

West Virginia and Southeastern Ohio.—The oil-field of southeastern Ohio embraces Washington, Harrison, Monroe, Morgan, Mercer, Belmont, Jefferson, and Noble counties. Some very good wells have been drilled in Washington county. Several of the largest wells have been found in the Maxon sand. In this region the "Big Injun" sand has previously been regarded as the principal producing formation; and the tendency is to regard the recent large wells in the Maxon sand as "freaks" and of local occurrence only. Operations were active in 1906 in Harrison and

Jefferson counties; and both furnished some new producing territory. The greatest activity was exhibited in the Berea grit. The Woodfield district in Monroe county was the most active in southeastern Ohio. In addition to operations under way and within defined limits, there is much work in advance of the old developments. In southeastern Ohio the old districts have been fully developed, and except in a few localities, there is no longer room for additional wells.

The wells in West Virginia furnish 45 per cent. of the total production of the fields producing high grade petroleum. This gusher territory has been gradually declining. With the single exception of a new pool in the Green district in Wetzel county, no territory in West Virginia furnished inducements for renewed activities during the winter of 1906-1907. When operations are resumed in the spring of 1907 there will probably be a repetition of the work of 1906. Drilling in old territory and an occasional test, looking for extensions in some of the old fields, will then, as now, be the rule. The wells completed during the latter part of 1906 were light producers, as a rule, located within definite limits. Several good strikes were made in the Point Pleasant district in Tyler county. There was considerable interest aroused in the new developments on State Run Road in Wetzel county. There is room for a pool of some dimensions in this locality; and the territory has not been condemned by previous tests.

In Roane county the extension of the Rowell's Run district (Calhoun county) attracted some notice, as most of the wells drilled are test wells of more or less importance. In Ritchie county the salt-sand district received much attention. The developments extend over a large area; and a large percentage of the work is of the experimental kind. In the southeastern counties, (Cabell, Lincoln, Putnam, and Kanawa) West Virginia, some new work was started during the fall of 1906; but in all save Cabell county it was purely experimental. In general it can be said of operations in West Virginia that there has been a general decline, especially in the deep-sand district. Except for the new work projected by the various gas companies and the test wells drilled, things have been very quiet. The territory that awaits development is not good for anything better than light pumpers; and the expense of drilling and equipping small wells is too great to induce investment. The work under way consists of drilling odd wells in the old fields, which, if they prove remunerative, can be operated in connection with the old producers.

Tennessee and Kentucky.—There were practically no petroleum developments in Tennessee during 1906. Kentucky operators continue to manifest interest in the theory of a connecting link between their developments and the new field in southern Illinois, but nothing has yet been furnished to support it.

Nine distinct fields were developed in Kentucky in 1906. These include

Wayne, Knox, Whitney, Cumberland, Wolfe, Esbill, Bath, Rowan and Floyd counties, but operations were confined mostly to Wayne and Wolfe counties, and the new wells completed from month to month help to sustain a total production of about 3000 bbl. per day. The production of high grade oil in the Kentucky-Tennessee fields during 1906 was about 1,000,000 bbl. The output in 1905 aggregated 1,200,000 bbl., but conditions were then more favorable for development work than during 1906.

The year 1906 was marked by the lowest prices paid for crude oil in the history of the field, and the decline in production naturally followed. The average price paid for the Kentucky-Tennessee high grade oil was about 86c. per bbl. Kentucky's inferior grade oil commanded about 56c. per bbl.

The total production of the Appalachian field in 1906 was 27,345,600 bbl., as compared with 28,324,324 bbl. in 1905.

MONTHLY AND YEARLY AVERAGE PRICE OF PIPE-LINE CERTIFICATES PER BARREL OF CRUDE PETROLEUM AT THE WELLS IN THE APPALACHIAN FIELD.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Average
1899.....	\$1.17	\$1.15	\$1.13	\$1.13	\$1.13	\$1.13¼	\$1.22½	\$1.27½	\$1.44½	\$1.50⅞	\$1.57⅜	\$1.65⅛	\$1.29⅜
1900.....	1.66⅔	1.68	1.68	1.55	1.39⅜	1.25½	1.25⅝	1.25½	1.23	1.10⅜	1.06¾	1.08¾	1.35¼
1901.....	1.69½	1.25	1.29	1.20½	1.07⅜	1.05	1.13⅜	1.25	1.25⅜	1.30	1.30	1.21	1.21
1902.....	1.15	1.15	1.15	1.17½	1.20	1.20	1.22	1.22	1.22	1.28	1.38	1.49	1.24
1903.....	1.52	1.50	1.50	1.51½	1.51½	1.50	1.53	1.56	1.56	1.69½	1.79½	1.88½	1.59
1904.....	1.85	1.82½	1.73	1.65	1.59½	1.57	1.53½	1.50	1.54½	1.56	1.58	1.55	1.62⅜
1905.....	1.43½	1.39	1.37½	1.32½	1.28	1.27	1.27	1.27	1.39	1.58½	1.59½	1.58	1.31⅜
1906.....	1.58	1.58	1.58	1.60	1.64	1.64	1.63½	1.58¾	1.58	1.58	1.58	1.58	1.5969

The Lima Field.

The Lima field includes the oil-pools in northwestern Ohio and northeastern Indiana in the Trenton formation. The existing conditions in these two States are practically the same as in the Appalachian field. The producing areas have been defined and operations are confined to the drilling of odd wells and the discovery of small local pools. But there seems to be a larger number of wells abandoned than drilled. The casing, tubing and other material from these abandoned wells is taken to the new field in Illinois where there is a great demand for it. The output of the Lima field in 1906 was 25,680,000 bbl. as compared with 22,102,000 bbl. in 1905. The average value of the oil for 1906 was about 90c. per bbl.

The Texas and Louisiana Fields.

Texas.—There was a heavy decline in the output of the fuel oil fields of southern Texas and Louisiana in 1906, while the districts in northeastern Texas showed a marked gain, amounting to over 100 per cent. The production of the higher grade in the Corsicana oil district showed but a slight gain, amounting to less than 25,000 bbl.

Following are the figures of the production of the Gulf coast oil fields for 1906 and 1905, as reported by the *Oil Investors' Journal*, of Beaumont, Texas:

PRODUCTION OF GULF COAST FIELDS.
(In barrels.)

	1906	1905		1906	1905
Spindletop, Texas	1,075,755	1,652,780	Corsicana	336,387	311,544
Sour Lake	2,143,723	3,362,153	Powell	675,842	132,866
Saratoga	2,170,153	3,125,028	Henrietta	111,072	101,661
Batson	2,388,288	3,774,841	South Bosque	1,000	300
Humble	3,570,844	15,594,310	Jennings, La	7,051,688	8,891,416
Dayton	92,460	60,294	Welsh	35,000	10,000
Matagorda	1,000	46,471	Anse La Butte	15,000	9,000
Hoskins Mound	100,000	......	Caddo	4,650	......

The total production in 1906 was 19,772,862 bbl., against 37,072,674 in 1905, showing a decline of 17,299,812 bbl. or nearly 48 per cent. The principal decline was in the Humble district in Texas and the Jennings district in Louisiana. No new fields of any importance were discovered in 1906 and nearly all the old wells are down to a pumping basis.

Southeast Texas, embracing the Texas Gulf coast districts, produced 11,542,000 bbl. in 1906, in comparison with 27,616,000 in 1905, a decrease of 16,074,000. The average price for southeast Texas crude in 1906 was 58c. per bbl., against 38c. per bbl. in 1905. The total production of southeastern Texas since the discovery of oil at Spindletop in January, 1901, has been as follows: Spindletop, 37,801,000 bbl.; Sour Lake, 20,-841,000; Saratoga, 6,034,000; Batson, 17,072,000; Humble, 19,165,000; Dayton, 153,000; Matagorda, 200,000; Hoskins Mound, 100,000; total, 101,366,000.

The total production by years since 1901, as published by the *Oil Investors' Journal*, has been as follows: 1901–5,185,883 bbl.; 1902–17,-897,146; 1903–17,453,582; 1904–21,672,111; 1905–27,615,877; 1906–11,542,223; total, 101,366,822.

For northern Texas, which includes the Corsicana and Powell districts, the total yield for the past 11 years has been 6,277,000 bbl. The production in 1905 was 546,000 bbl. and in 1906 it was 1,124,000 bbl.

The total production of Texas since it came into prominence as an oil producing State has been about 108,185,000 bbl.

Louisiana.—This State has produced about 24,100,000 bbl. in the last five years. The production in 1905 was 9,672,015 bbl. and in 1906 it was 7,100,000, a decrease of 2,572,015. In regard to the number of wells in the Southwest their average production, etc., the *Oil Investors' Journal* says:

"On Dec. 31, 1906, there were 875 wells in operation in the Gulf Coast region of Texas and Louisiana. Of this number 780 were credited to

southeast Texas and Louisiana. The output of the Gulf Coast region in December was 1,278,121 bbl., southeast Texas being credited with 860,174 bbl., and Louisiana with 417,947 bbl. In north Texas 891 wells were operated on Dec. 31, making the total number in operation in Texas 1671 and in both Texas and Louisiana 1766. The average daily output of all southeast Texas districts at the close of the year was 27,747 bbl., and the average daily output per well 35.6 bbl. The average daily output of the Louisiana districts in December was 13,841 bbl., and the average daily output per well was 141.9 bbl. The average daily output of the entire Gulf Coast region in December was 41,288 bbl. and the average daily output per well was 47.2 bbl."

For over a year the demand for Gulf Coast oils has been much greater than the supply. The present supply is about 45,000 bbl., while the demand is nearly 100 per cent. greater. There were 729 wells completed in the Gulf coast district in 1906. Of this number 220 were unproductive of oil in paying quantities, leaving 509 that were productive. Of the once famous Jennings oil field in Louisiana the *Oil Investors' Journal* says:

"The field was extended over a thousand feet to the east, principally along the working line between the Crowley Oil and Mineral Company farm and the Arnaudet line on the south. The production at the close of the year was the lowest in the field's history since the development of the deep sand in 1904. The quality of the oil is poor in some instances, and requires treating to make it marketable. Plants for this purpose have been erected on the field. Most of the producing wells completed in 1906 started off in gusher style, but either sanded up or went to salt water, being later put on air and made to produce from 300 to 2000 bbl. The conditions of the market in 1906 made the operation of wells with air profitable. The Jennings-Heywood Oil Syndicate installed in October what is probably the second largest air-compressor plant in the United States. A battery of 50 boilers is required for steam making purposes, giving the plant a capacity of 11,500 cu.ft. a minute. The Texas Company in March completed a 6-in. line from the field to Lake Charles, where loading racks on the Kansas City Southern railroad were installed and the movement of oil inaugurated. The construction of this line gave Jennings another rail outlet, the Southern Pacific being the only railroad over which shipments had been made up to this time."

Present Conditions.—These may be best understood by quoting from an article entitled "Petroleum Developments in Louisiana and Texas," which appeared in the *Engineering and Mining Journal* of Jan. 5, 1907, as follows:

"The former conditions and supposed course of development of the coastal field and its actual standing at the end of 1906 are aptly and briefly

set forth in a circular issued recently to stockholders of the J. M. Guffey Petroleum Company. A portion of the circular is as follows:

'The J. M. Guffey Petroleum Company was organized in May, 1901, shortly after the discovery of oil at Spindletop, which was the first oil field discovered in Texas. At that time it was supposed that the supply was practically inexhaustible and the company at its organization acquired a very large territory, upward of 1,000,000 acres, of what was supposed to be undoubtedly and practically invaluable oil territory; and for this it issued $15,000,000 of capital stock. It was very soon discovered, however, that the Spindletop field was of exceedingly limited area, the fact being that 90 per cent. of the entire production of that field has come from about 100 acres. It was also found that the life of the field, contrary to all expectations, was exceedingly short, the wells very soon ceasing to flow, and within a few years it practically disappeared as a producing field. A little more than five years have elapsed since the discovery of the Spindletop field, and in that time five additional fields have been found. The total productive area of all these fields probably does not exceed 1000 acres, and at the same time they have been found widely scattered, the fields or pools so developed being as much as 190 miles apart, which has entailed not only a large expenditure in the discovery of the fields, but also very great outlays of capital for pipe lines and storage facilities. These fields also have all of them had but a short life. At the present time the total production of oil in the State of Texas is about 30,000 bbl. per day, and of this only about 12,000 bbl. is of such quality as to be available for refining. The Louisiana fields are producing about 15,000 or 20,000 bbl. per day, but there are no pipe lines connecting this field with the properties of the Guffey and Gulf companies, and the oil is not generally of such quality as to be refinable.'

"While the above quotation is not optimistic, it must be borne in mind that Texas refineries and tankage represent a large investment of capital and that they have exceptional shipping facilities by land and water and they will use Indian Territory oil. The center of the Southern petroleum industry will remain in Beaumont and Port Arthur and active prospecting will continue during 1907, because the lack of suitable coal within a reasonable distance will keep fuel oil at a price that will make even small producing wells remunerative and any large gusher output would still be handled by the refineries at prices commensurate with quality. Moreover, while actual conditions during 1906 were unfavorable, it cannot be assumed that the coastal field is destined to a steadily declining production, for it requires only a few months to develop the maximum production of its gusher fields, one of which may be located any day. Past experience shows, however, that to maintain its production one or two new pools must be discovered each year."

The Mid-Continental Oil Field.

C. N. Gould, of the Oklahoma State University, writes as follows regarding the Kansas-Oklahoma oil field: "In Oklahoma and the Indian Territory oil has been found chiefly in three areas, at least two of which are along anticlines. There are a number of other anticlinal folds in the Carboniferous rocks of the Territory where no drilling has yet been done, but where oil and gas will probably be found in considerable quantities. The anticlinal theory of oil and gas deposits has thus received additional confirmation in Oklahoma. All the oil so far found in that State is from Carboniferous rocks, the greater part of it coming from the lower part of the Pennsylvanian. In Osage, Cherokee and Creek nations, where the greatest developments have taken place, the rocks dip at rather constant angles to the west. In this region, however, there are occasional broad anticlines, the axes of which run approximately parallel to the strike of the rocks. It is along these anticlines that the greater part of the oil has been found. The most extensive field extends from Peru, Kan., south along the Osage-Cherokee line, past Tulsa and Red Fork, as far as Mounds, a distance of over 50 miles. Another similar field is being developed 25 miles farther east. The oil industry in Oklahoma is still in its infancy. The drillers have been slowly working south and have not yet penetrated what appears to be the most prolific oil regions. There are a number of anticlines in regions yet unexploited, which will probably develop into fields more productive than any yet found."

The *Oil Investors' Journal* (Jan. 18, 1907) publishes the following information as to the number of wells completed in the Mid-Continental Field in 1906: "Three thousand five hundred and fifty-four wells were completed in the Mid-Continental region in 1906. Of the total number, 2626 were producers, 501 dry and 427 gassers, representing an average of 295 completions a month throughout the year—218 producers and 77 dry or gassers. In 1905, 4020 wells were completed in the Mid-Continental field. The record for 1906, therefore, shows a falling off of 466 completions, the decline in operations being due to a desire on the part of operators to bolster up a falling market and bring the output within range of the capacity of existing pipe-line facilities.

"The year opened with a 52c. market and closed at 39c. Field work attained its greatest activity in May, when 498 wells were completed—404 producers and 94 dry or gassers. Next to May, January holds the record, with 391 completions—315 producers and 76 dry or gassers. On the last day of May, 311 wells were drilling and 44 rigs up, while at the end of January, 238 wells were drilling and 34 rigs up.

"Oklahoma and Indian Territory are credited with 2779 completions in 1906—2266 producers and 513 dry or gassers. An average of 146 wells

were drilling and 30 rigs up at the close of each month was the record for the year. May and January were the banner months. Four hundred and four wells were completed in May—346 producers and 58 dry or gassers, with 259 wells drilling and 41 rigs up at the close of the month. January is credited with 310 completions—264 producers and 46 dry or gassers, with 196 wells drilling and 32 rigs up at the close of the month.

"Kansas is credited with 775 completions in 1906—360 producers and 415 dry or gassers. An average of 36 wells were drilling and two rigs up at the close of each month throughout the year. May and January were the busiest months. In May, 94 wells were completed—58 producers and 36 dry or gassers, with 52 wells drilling and three rigs up on the close of the month. January is credited with 81 completions—51 producers and 30 dry or gassers, with 42 wells drilling and two rigs up on the close of the month."

Further details of the petroleum industry in the Mid-Continental Field in 1906 will be found in the paper by Erasmus Haworth, State Geologist of Kansas, which follows this article.

Other Fields.

Illinois.—During 1906 Illinois forged rapidly to the front as a petroleum producing State. Several years ago attention was first called to the field by the drilling of several small wells in Clark county. The field as now outlined extends from Oakland in northern Coles county, through Cumberland, Clark, Crawford and Jasper counties to Robinson. Early in 1906 "wild-catting" became a feature and numerous gushers were struck. The production increased so rapidly that it was not possible to transport all the oil, or erect tankage rapidly enough to handle it. The production in five months increased from 10,000 to 30,000 bbl. a day. As there were no facilities for transportation, much of the oil was stored until a pipe line was completed. The Geological Survey has taken up the investigation of the gas and oil resources of the State. This, together with the "wild-catting" which is being carried on, promises to develop oil and gas at a number of points. The production of the State for 1906 was approximately 4,900,000 bbl. Further details will be found in the paper by Dr. H. Foster Bain, which follows this article.

Colorado and Utah.—Considerable prospecting for oil and asphalt was carried on in Colorado and Utah during 1906. The Rangely oil field in Rio Blanco county, Colorado, attracted some attention. This field lies in the extreme northwestern part of Rio Blanco county, near the Routt county line and not far from the Utah State line. This field extends about eight miles from north to south and 15 miles from east to west. The structural formation is that of a dome of very large dimensions, and many experts are of the opinion that some large gushers will be struck. The shales in

which the oil is found are fissured, due to the crustal movement. The high grade oil is found in pools in this fissured shale, and is really a natural distillation product, or a seepage from the underlying strata.

At Granby in the Middle Park oil was struck at a depth of 50 ft. in 1905. This was apparently asphaltic oil, being easily accounted for by the large deposits of gilsonite, occurring a few miles north of this place.

At Yampa, Routt county, Colorado, and vicinity there are fine indications of oil, such as petroleum shales. There are no railroads in northwest Colorado and consequently developments in this field have been rather slow.

The region along the eastern and northern shores of the Great Salt Lake in Utah from 10 to 60 miles northwest of Salt Lake City is being prospected for oil and asphalt. Many hundreds of claims have been taken up, but little oil has as yet been encountered. About 150 miles further south, at Fillmore, Utah, a small amount of boring is being conducted with a view to tapping commercial quantities of petroleum. As yet no petroleum pools have been located in this region.

California.—Although the production of petroleum in California in 1906 showed a material falling off from that of 1905, this was caused by the production growing more rapidly than the demand, and not because of a scarcity of petroleum. No new fields were developed during the year. In fact many good wells were not operated. This decline in production is the result of a well organized movement to curtail the production until the consumption of oil shall equal the capacity of the wells.

Although the price of crude oil in California was low during 1906 and the supply greater than the demand, prospecting was carried on extensively, not as a development to increase production, but to extend and prove large areas of land and to provide for future requirements when the price of oil shall be sufficiently high to warrant development for the production. In the Santa Maria field many properties were proved by extensive drilling. The Sunset-Midway and the McKittrick have not been attractive fields for "wild-catting" on account of their inadequate transportation facilities.

Santa Barbara is undoubtedly the coming county in the petroleum industry, being so near the sea and some of its fields being very productive. Several oil companies in this county are planning to market their own petroleum. The project calls for an independent steamship line to run from Santa Barbara and San Luis Bay to San Francisco.

The great permanency of the oil wells of California is now an established fact. Wells that have produced constantly for 10 or 12 years are still paying. The great thickness of the sand, averaging more than 400 ft., the great saturation of the sand with petroleum and the absence of water in it, tend to produce long-lived wells. The *Pacific Mining and Oil Reporter*

(Feb. 5, 1907) publishes the following article with reference to the Panama pipe line:—

"According to the statements from the offices of the Union Oil Company, the first shipments of oil have been run through the Panama pipe line without difficulty, the system meeting all the expectations of the big producing and marketing company, and at once putting California in close touch with the oil markets of the Atlantic. California oil can now be as cheaply transported to the Atlantic coast as it can to Alaska or the Hawaiian Islands, where it has now taken preference over all other fuels in matter of convenience as well as economy.

"The plan of building a pipe line across the isthmus was formulated some three years ago by John Baker, manager of the marine, sales and manufacturing department of the Union Oil Company. As early as July, 1903, he conceived the possibilities of marketing California oil in the East. Transportation was the only drawback. Being an independent company it could not secure rebates, and shipments by trans-continental railways were out of the question. The two months voyage around Cape Horn was too dangerous and expensive. Therefore an isthmian pipe line was the only practicable solution of the question. The problem has been successfully carried through, affording an outlet for a vast quantity of California oil. The line is an 8-in., and has an approximate capacity of 25,000 bbl. daily. Its entire length is but 50 miles, and the maximum elevation 210 ft. It runs along the Panama Railway right of way, which is almost parallel to the proposed route of the Panama canal. The Pacific terminus is near the city of Colon. There is but one pumping station, which is at the initial station, Panama."

From figures gathered by the *Pacific Mining and Oil Reporter*, the production of mineral oils in California in 1906 showed a decline of 5,133,000 bbl. The total production for the year was approximately 30,538,000 bbl. against 35,671,000 bbl. in 1905. The accompanying table shows the production by years and districts since 1902.

Although the production of petroleum in California in 1906 was much less than in 1905, the future of the industry is assured. There is a rapidly growing demand for petroleum and its products, and there are many factors which will aid this great industry in its growth. Several recent contracts with Japan will nearly double the exportation of crude oil from California. The recent completion of the pipe line across the Isthmus of Panama will afford this oil ready access to eastern markets. The aggregate amount of business by this service will, it is claimed, amount to several millions of barrels per year. The great demand for California asphalt outside of the State, due to its superior quality, will also aid the petroleum industry, as most of the asphalt is obtained as a by-product in the refining of petroleum. Most of the asphalt has been used for roofing and paving,

but it will doubtless be used largely in the manufacture of new fuels. An extended market, better methods of operation, and a broader view of the possibilities for the industry will undoubtedly cause an advance in prices of California crude in the near future.

PRODUCTION OF CRUDE OIL IN CALIFORNIA. (a)
(In barrels.)

District.	1902	1903	1904	1905	1906
Coalinga	500,750	2,138,058	5,114,000	8,869,000	8,500,000
Sta. Maria and Lompoc	116,500	208,890	670,500	5,300,000	5,400,000
Kern River	8,872,115	16,342,100	17,500,000	14,000,000	11,000,000
Los Angeles	1,047,300	793,765	1,200,000	3,000,000	1,700,000
Sunset	144,200	353,100	400,000	400,000	500,000
Midway	50,000	29,200	910	5,000	5,000
McKittrick	639,500	1,353,500	1,875,925	720,000	500,000
Newhall and Ventura	626,540	683,500	663,100	500,000	400,000
Fullerton and Brea Canyon	1,195,015	1,427,700	147,500	1,750,000	1,700,000
Whittier and Puente	687,030	878,015	748,000	960,000	750,000
Summerland	94,550	131,000	120,000	75,000	56,000
Sargents	....	....	35,090	20,000	15,000
Halfmoon Bay	....	....	1,000	2,000	2,000
Arroyo Grande	....	....	....	5,000	10,000
Total	13,973,500	24,337,828	28,476,025	35,671,000	30,538,000

(a) Reported by the Petroleum Miners' Association.

Petroleum in Foreign Countries.

Australia.—In several places in New Zealand, but notably at New Plymouth on the west coast of the North island, and near Greymouth on the west coast of the South island, there are indications that reservoirs of oil exist at considerable depths from the surface. In the New Plymouth district, oil was discovered near the sea coast, some miles from the base of the extinct volcano known as Taranaki, or Mount Egmont. The history of boring for petroleum in that district extends back to 1865. It was one of general failure. Further unsuccessful efforts were made to obtain British capital to continue the boring. Eventually a New Zealand syndicate was formed. A bore was put down in the vicinity of the abandoned workings to a depth of 905 ft.; oil was struck but water was also present in the bore, and as the water rose and fell with the tide, it was evident that there was a fissure running out to sea. Operations were continued in another direction, and a bore put down to 905 ft. yielded a flow of 10 barrels a day. The continual falling in of a soft layer of "papa" rock, however, soon stopped the flow of oil. In 1896 a fresh bore was put down, and, at a level of 1976 ft. a considerable flow of oil was struck; but again the water interfered with the flow, and, eventually, the derrick and works were destroyed by an explosion and fire. More recently the operations were continued, and, during 1906 it was announced that all difficulties had been overcome, and that excellent oil, with a total absence of water, was flowing from the well. The first flow was at the rate of about 30 bbl. a day, but there was a great quantity of mud and sand mixed with the oil.

On April 30, however, it was found that there was an enormous pressure of gas, and that the oil was flowing at the rate of 400 bbl. a day. The bore is down to a depth of 2331 ft. The company has secured rights over 5000 acres of land in the vicinity. The company intends to erect a refinery, and hopes within 18 months to have six other bores sunk and working. The company's expert anticipates a yield of 4000 or 5000 barrels a day from the present bore.[1]

Austria-Hungary.—The oil fields of Galicia continue to decline in their production. The production in 1904 was 827,116 tons. This fell to 801,796 tons in 1905 and was still further reduced to 760,441 tons in 1906. The production of the Boryslaw field has shown its effect in sustaining the total production of petroleum in Galicia since the beginning of the present century. The output of the Boryslaw field is given in the accompanying table:

PRODUCTION OF BORYSLAW FIELD

Year.	Tons.	Year.	Tons.	Year.	Tons.
1898..	13,000	1901..	132,000	1904..	546,000
1899..	18,000	1902..	226,000	1905..	546,500
1900..	55,000	1903..	373,000	1906..	562,000

All of the Galician fields with the exception of the Boryslaw field have shown a marked decline.

PRODUCTION OF OTHER GALICIAN FIELDS

Field.	1904 Tons.	1905 Tons.	1906 Tons.	Field.	1904 Tons.	1905 Tons.	1906 Tons.
Potok	22,864	22,479	16,325	Other West Galician fields	34,411	35,670	30,883
Rogi	47,531	24,234	11,452	Schodnica	72,627	60,201	47,151
Rowne	2,454	1,609	1,535	Urycz	27,420	20,346	17,930
Tarnawa-Wielopole	10,707	32,956	24,870	Mraznica	4,915	3,646	1,610
Krosno	48,228	43,559	34,267	Other East Galician fields	9,909	10,600	12,220

Burma.—In the Burma oil fields steps have been taken toward the adoption of the bulk system of transport and distribution. A pipe line has been laid from the oil field at Yenangyat through the Singu field to Yenangyaung, the principal oil producing area, a distance of more than 50 miles, and arrangements are now being made to connect the last mentioned field in the same way with the refineries at Rangoon, 275 miles distant. At the same time tank steamers have been purchased and storage tanks have been constructed at the principal ports of India. In 1905-6 Burma exported £58,600 worth of kerosene, £62,000 worth of paraffin wax and £109,000 worth of paraffin wax candles to foreign countries, while £1,-

[1] London *Times*, Financial Supplement, June 25, 1906.

975,000 worth of oil was sent to India. A high grade of kerosene is now being produced to compete with American and Russian oils. The industry is in a flourishing condition and is growing rapidly.

Canada.—There are two important oil districts in Canada, one in southwestern Ontario and the other in Alberta. At present the main supply of petroleum in Canada is secured from Ontario, but the production is steadily diminishing. There are immense deposits of petroleum in Alberta which have but recently come to public notice. It was not until June, 1905, that it became known that another great oil field had been discovered, although oil had been found by boring as early as 1902, and seepages of oil had attracted attention a hundred years ago. This great discovery of oil will revolutionize the industries of Canada. The Canadian government allows a bounty of 1.5c. on each gallon of oil produced at home, and the Alberta oil is of such a quality that it can be refined for 40c. to 50c. per bbl. cheaper than any other oil known. It is far superior to any oil found in Canada or the United States, with the possible exception of the oil of Wyoming and Montana. Plans have been made for the building of refineries and pipe lines for the distribution of the oil throughout the West. J. S. Williams, an oil expert from California, is of the opinion that a great underground oil lake exists along the eastern slope of the Rocky Mountains. This probably extends 250 miles north of Athabasca Landing to the southern extremity of the province and 800 miles south to Colorado.

Germany.—According to J. I. Brittain, U. S. consul at Kehl, the petroleum industry of Germany would not pay expenses if it were not for the import duty on refined oil, which is $2.38 per 110 ℔. He says: "The petroleum district contains three basins in the vicinity of Hagenau, Alsace, namely, at Pechelbronn, Biblisheim, Sultz, and Durrenbach. At present about 400 wells are being pumped; there are no flowing wells. At each of the above named towns a refinery is located. The wells are not large producers, the best pumping but 15 to 20 bbl. per day, while the less productive pump but 3 to 5 bbl. The oil is very dark in color, and the chemical properties are said to resemble those of the Pennsylvania oil, while the specific gravity is about the same as Russian oil. The crude oil contains about 35 per cent. petroleum, 45 per cent. vaseline and lubricating oil; 10 per cent. coke, 5 per cent. water, 2 per cent. benzine, and the remainder waste matter. At present not many new wells are being drilled, as many of the wells recently brought in produce less than 10 bbl. per day, and a producer under that quantity does not warrant the expense incident to drilling a well, which is about 3000 marks ($714). The average depth of the wells is about 1150 ft. The output of refined petroleum at the three refineries is about 20,000 tons annually."

Italy.—As early as 1893 a French company obtained a concession from the Italian government to explore a certain tract in the Appenines, near

Piacenza, and to exploit any deposits of petroleum found there. The success of this company was sufficient to cause the formation of another French syndicate four years ago, and last July these two were absorbed by a Genoese company, with a capital of $3,000,000. The wells already bored are 95 in number, of which 70 are practically exhausted. The remaining 25 produced about 40,000 bbl. of crude oil in 1905; and with the eight wells now boring it is expected that the total production for 1906 will be over 65,000 bbl. The concession of the new company comprises about 11,000 acres. The wells, none of which is a gusher, reach a maximum depth of 1300 ft. and the engineers in charge say that those which are exhausted may be made to yield again by deepening. This has not yet been undertaken, perhaps because by the terms of the grant it is necessary to bore new wells in order to maintain control of the territory. A central motor is used to operate, by cables, the pumps of the various wells.[1]

Japan.—The petroleum fields of Japan have been developed rapidly in recent years. The total production in 1898 amounted to about 309,000 bbl., which had increased to 786,000 bbl. in 1900. Most of this petroleum comes from the province of Echigo. All but one of the companies working these fields are Japanese, the exception being an offshoot of the Standard Oil Company. The Japanese government has recently imposed a duty of 9c. per gal. on imported oil. This will undoubtedly give the petroleum refining industry an immediate impetus. Hitherto Japan's petroleum trade has been dull, though the demand for the refined products has been yearly increas ng.

Mexico.—The contracts between Pearson & Son and the Mexican government for the investigation and exploitation of the petroleum deposits in the states of Chiapas, Campeche, Vera Cruz, the valley section of San Luis Potosi and the southern district of Tamaulipas, appeared in the *Diario Oficial* for May 14, 1906. The concessionaires are authorized to explore the above mentioned areas for deposits of petroleum and its derivatives, and to exploit such deposits when found. The concessionaires are to enjoy the following privileges: export free of duty all products of their exploitation, free importation of machinery and materials required in the work of exploitation. In return the concessionaires are to turn over to the National Treasury 7 per cent., and to the respective states 3 per cent. of the annual profits of the deposits.

Prof. J. F. Kemp, in speaking of the Ebano oil fields, mentions that the formation seems to be volcanic, the oil-bearing rock being a volcanic tufa under a cap of shale. The country is marked with small cones showing that the disturbances which fissured the ground and permitted the escape of the oil were of volcanic origin. The oil of this region is the heaviest

[1] *Eng. and Min. Journ.*, March 16, 1907.

known in the world, having a specific gravity in its crude state of almost that of water. There are numerous flows of asphaltum in the region, which is the residuum of oil which has seeped through to the surface and evaporated.

The dearth of facilities for obtaining small but necessary additions to the supplies of oil well materials, for making needed repairs of various kinds of oil well machinery, etc., in this part of Mexico, caused the Mexican Petroleum Company to provide itself with an adequate means of making repairs, and carrying supplies, which resulted in the installation of a boiler and blacksmith shop, an oil-well machine shop, a sawmill, a cooperage plant, an electric light and power plant, an ice and cold storage plant, a small standard railway system, and a compressed-air, pneumatic-tool, steel tank-building plant. With the last plant, 11 steel storage tanks have been built, having a total capacity of about 375,000 bbl. Two reservoirs have also been constructed, having a capacity of about one million barrels. All the above receptacles now contain oil. The petroleum is adapted particularly for fuel purposes. The first oil-burning locomotive to be introduced in Mexico was brought in by the Mexican Petroleum Company in May, 1903. The locomotive was fitted up after the manner in use by the Atchison, Topeka & Santa Fe Railway Company in the United States. The results obtained were eminently satisfactory, the petroleum proving to be of greater value for fuel purposes, per given bulk, than the petroleum fuel of California or Texas. On its two locomotives, the Mexican Petroleum Company has been burning fuel petroleum for three years, without any difficulty from any cause.

It being developed that the crude petroleum (which has a Beaumé gravity of about 14 deg. at the temperature at which it issues from the wells), yielded about 50 per cent. of commercial asphaltum, suitable for paving, roofing and painting purposes, a refinery consisting of six stills was installed and has been in operation for three years. All of the asphaltum of Mexico is produced from this refinery, and more than half of the paving done in Mexico during the last three years has been done with the product from Ebano petroleum, considerable quantities of which are now being shipped to Great Britain and the continent of Europe.

During the five years that development has been going on at Ebano, over 3,500,000 pesos have been expended there. Of this amount about 2,000,000 pesos have been paid out in wages to employees, the remainder representing the cost of material, supplies and transportation. About 500,000 bbl. of Ebano petroleum have been sold and consumed in Mexico. The average selling price has been about one peso per bbl. at the well. At the present time, development is being carried on at a rate that the fuel-petroleum markets of Mexico seem to justify.[1] The Mexican Central R. R. uses

[1]From a report by E L. Doheny, President of the Petroleum Company.

much of the oil as it comes from the wells as a fuel in its engines, which have been equipped with oil burners. When all of the railroad locomotives are equipped for the use of oil, several million bbl. of oil will be required annually to meet this demand. At present about 100 locomotives are equipped with oil burners and in the near future it is thought that at least 500 will be so equipped.

Peru.—According to a recent British consular report, in addition to the London & Pacific Company's works at Talara and Negritos, and those of the Peruvian Petroleum Syndicate, established at Lobitos, about 15 miles north of Talara, the latter of which is still in the development stage, a new company has been formed by Lima capitalists to work some deposits at Fernandez in the district of Mancora, on a property owned by Nicolai Taiman. This concern is still in the initial stage of development, but promises fairly well. The property is inland, about 20 miles from the coast, but with a steady fall to the sea, and should the boring give good results, the oil could be piped by gravity to tankage which would be erected close by the sea.

The output of the Talara company (with headquarters in London) is from 3000 to 4000 tons of crude oil per month. It works a considerable extent of territory, owns two tank steamers of about 3000 tons each, and disposes of about one-half of its production of bulk oil in Callao, and the remainder in Caleta Buena, Chile, where a large and increasing demand for fuel oil exists. Its production of refined oil, i.e., kerosene and gasoline, is sold in Peru. The Peruvian Petroleum Syndicate's property at Lobitos is in an advanced stage of development. It has about 1000 tons of oil stored in tanks, and during the course of the present year will doubtless commence shipments. New tankage, a wooden pier and other shipping facilities are projected. A great number of petroleum lands in Peru were applied for in 1906. In the majority of cases such lands are being held for speculative purposes, and a large proportion of them revert after a time to the government, the applicants failing to keep up their half-yearly payments to the Department of Mines.

Rumania.—The petroleum industry of Rumania has progressed steadily during the last 10 years. The production in 1906 was approximately 887,000 tons, against 615,000 tons in 1905. The developments are not only in the quantity of the oil produced but also in the methods of production. There is a tendency gradually to discard the old method of exploitation of hand-dug wells, and to extend exploitation by borings. The hand-dug wells have been used only in the upper strata and when these strata have become exhausted the wells have either been abandoned or boring has been continued in them to greater depths. Of the total number of dug wells classified as productive only about one-third produce regularly.

In 1905 there were 996 wells classified as producers; of this number 331 were bore holes and 665 dug wells. In 1906 there were 1136 wells classified as producers, and of this number 443 were bore holes and 693 dug wells. The growth in the petroleum industry of Rumania has not been limited to the production of crude petroleum. The refineries have increased rapidly their output of the products of petroleum. In 1905 the output of refined products was 199,300 tons; this increased to 278,600 tons in 1906.

Large areas of the Rumanian oil fields are the property of the State, and have been unworked up to the present time, but a law has recently been passed providing for the leasing of these lands to private companies and containing important restrictions with the object of preventing monopolies. With the introduction of improved methods and the opening up of new territory the petroleum industry in Rumania undoubtedly has a bright future.

Russia.—The production of petroleum in Russia in 1906 was approximately 446,100,000 poods, or about 7,249,000 tons. The production in 1904 was 615,300,000 poods and in 1904 about 400,000,000 poods. The Baku field is beginning to recover from the disasters due to the recent disturbances. The production of 1906 shows an advance over that of 1905, but it will be several years before the field regains its normal condition.

By the fire of August, 1905, about 500 drilling contrivances were destroyed and scarcely half of these were restored by the end of 1906. The normal price of the drilling crane is from 4500 to 5000 rubles, but the demand for new drilling cranes is so great that the price has advanced to 8000 rubles. In 1906 it was impossible to fill numerous orders. Many firms were forced to accept a waiting position and the drilling activity seems, on the whole, to have been stopped. In the productive wells a considerable flooding of the oil stratum is already taking place and a smaller output is the natural result. In many wells the columns of pipes have settled, which both increases the cost of operation and decreases the production. It also appears that many wells which were formerly productive have been completely destroyed and must be abandoned. Much of this damage is caused by caving, and inflow of water which has forced back the oil. All these circumstances prove that the time is still far distant when the Baku naphtha production will resume its former standing. As regards quiet and order there is now little ground for complaint, although thefts, damaging of pipe lines and stealing of crude oil are still common occurrences.

An experiment has recently been made in Batoum export trade. A special ship has been fitted up for carrying kerosene in bulk to the Turkish ports where it is sold directly to the consumers. This method of shipment and sale will doubtless experience considerable extension.

A valuable paper upon the specific gravities of Russian crude oil and its products at various temperatures was read recently before the Russian Imperial Technical Society at St. Petersburg by M. P. Kazankin. The material used for the investigation started with light benzine and ran to the heaviest lubricating oil, being prepared from the crude of Baku. In all 22 different samples of oil were investigated and their specific gravity ascertained at different degrees, namely, at 0, 8, 15, 22, 30, and 40 deg. C. The apparatus which the author used for the purpose of ascertaining the exact specific gravity was a bottle of 50-52 c.c. and with an expansion above the neck of the flask. After being filled up with the necessary crude oil or its products, the specific gravity bottle was placed in a water bath, the temperature of which had been regulated in such a manner that it could be maintained for any length of time without varying a tenth of a degree. The necessary correction which now has to be made for every degree C. of the various products is given in the accompanying table.

SPECIFIC GRAVITY CORRECTIONS FOR RUSSIAN PETROLEUMS

Oil.	Specific gravity.	Correction per degree C.	Oil.	Specific gravity.	Correction per degree C.
Baku petroleum ether	0.64483	0.000949	Nobel distillate	0.85084	0.000704
Schibaieff benzine	0.70947	0.000905	Grosny crude	0.85640	0.000706
Grosny benzine	0.71035	0.000890	Schibaieff pyro-naft	0.85960	0.000684
Nobel benzine	0.71713	0.000898	Nobel pyro-naft	0.85991	0.000682
Heavy benzine	0.75585	0.000816	Solar oil	0.86960	0.000680
Grosny light kerosene	0.80041	0.000749	Baku crude oil	0.87262	0.000673
Grosny heavy kerosene	0.81300	0.000744	Nobel distillate	0.87927	0.000664
Baku kerosene	0.82364	0.000731	Nobel solar oil	0.88084	0.000662
Baku kerosene (second sample)	0.82394	0.000733	Nobel distillate	0.89156	0.000654
Schibaieff kerosene	0.82436	0.000737	Nobel spindle oil	0.89895	0.000635
Schibaieff heavy benzine (astraline)	0.83106	0.000723	Nobel lubricating oil	0.90975	0.000632

The above corrections will be welcomed by not only professional men but by the trade, as this is the first time that the table has been revised since it was first prepared by Professor Mendeleeff 25 years ago.

The Shale Oil Industry.

The shale oil industry has assumed large proportions in recent years. Large areas of oil shale are found in the United States but they have not been worked much as yet. In Scotland and New South Wales this industry is growing rapidly, and the oil shale deposits of other countries are being exploited previous to working. The output of oil shale in Scotland is about 2,500,000 tons a year, and the petroleum products extracted from the shale each year are valued at $10,000,000. The chief products extracted are illuminating oil, naphtha, paraffine wax and sulphate of ammonia. The *Petroleum Review* (March 2, 1907) contains a description of the oil shales of Scotland and the methods of working

them. The shale occurs in stratified beds like coal and is worked in very much the same way. Large deposits of oil shale occur in New South Wales, which cover an area of about 20 square miles. These deposits have been worked over 40 years and the value of the petroleum extracted up to the present time amounts to about £2,100,000.

Petroleum in Illinois.

By H. Foster Bain.

The important oil fields now being developed in southeastern Illinois first attracted attention in 1865. At that time about a dozen wells were drilled within what is now known as the Casey district. These wells were drilled with little if any casing, and the water in the holes so held back the oil that the small showing discouraged the operators and the attempt was abandoned. In 1904 drilling in this territory was resumed, the work being taken up by J. J. Hoblitzell & Son, of Pittsburg. Small amounts of oil and some gas were found in April in the first hole drilled, and in October the second well came in as a 45-bbl. producer. Since that time the development of the field has been rapid. The productive territory has expanded to the northwest and southeast, until it now stretches from Coles to Lawrence county. The principal production is from Clark, Cumberland, Crawford and Lawrence counties. The northern field, lying mainly in Clark, but including a portion of Cumberland county, is known usually as the Casey or shallow pool. The wells stretch from Westfield on the north to the south boundary of the county. The Crawford county pool crosses the entire county, and in Lawrence county productive wells have been found to a short distance south of Bridgeport. Commercially three districts are recognized, as is shown in the following statement of wells producing on March 1, 1907: Clark and Cumberland counties, Casey district, 2085; Crawford county, 932; Lawrence county, 194; total, 3211.

In each several pools are recognized. For example, within the Casey district are the Westfield pool to the north, the Siggins to the northwest, and the Johnson township pool to the southeast. The general distribution of the productive territory is shown in the accompanying map. It may serve to orient this if it be noted that the field extends approximately from the St. Louis line of the Big Four Railway to the main line of the Baltimore & Ohio Southwestern. It is crossed also by the Pennsylvania, Illinois Central, and the Cincinnati, Hamilton & Dayton, while the Cairo line of the Big Four runs from north to south, just east of the productive territory.

Pipe Lines and Production.—The field is now served by the pipe lines of the Ohio Oil Company. An 8-in. line extends from Bridgeport on the south to Martinsville, a second one being under construction. At Martinsville the lines from the Casey district are joined, and two 12-in. lines

extend to Montpelier, Ind., where a connection is made with the main east-west lines of the Standard Oil Company.

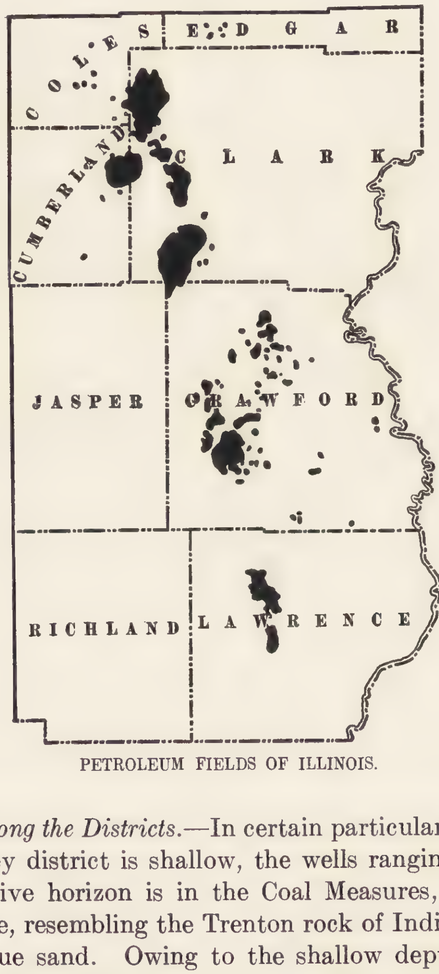
Shipments began in June, 1905, with 5489 bbl. At the close of that year 156,502 bbl. had been sent out, tank cars being used. In 1906 the pipe lines were connected up and 4,626,835 bbl. were sent out. In addition, tank car shipments were made by the Sun, Pure and Cornplanter companies. In January the pipe line runs were 752,670.62 bbl., and in February the business of the Ohio Oil Company was reported as follows: Gross stocks, 2,756,740.10 bbl.; runs from wells, 918,620.35 bbl.; regular deliveries, 1,013.57 bbl.; delivered to other lines, 451,032.39 bbl.

It may be noted that the runs for February averaged 32,808 bbl. per day, 3066 bbl. in excess of the receipts from the Lima oil districts of northwestern Ohio and Indiana. The oil is now selling for 68c. and the pipe lines are not yet able to take all that is offered. With the opening of spring the second line to Bridgeport is to be connected up and other improvements made with a view to handling the product. In the meantime a refinery is being built at Alton, and small plants are said to be planned for near St. Louis, Mo., and at Robinson, just east of the field.

Characteristics of the Field.—In general the field includes few flowing wells and no gushers. The wells are usually shot and most of them must be pumped. They hold up under pumping extremely well so far, and in a particular case 2000 bbl. per acre have been won in six months, with the tract only about half drilled. In a number of cases the wells have furnished enough oil to pay for themselves by the time they were connected up. In Johnson township of Clark county the average initial production of six wells was 57 bbl., the range being from 15 to 200. In the Bridgeport pool the average of 17, including one dry hole, was 106 bbl., the maximum being in this case 245 bbl. Wells running as high as 1500 bbl. are found, and a number have run up to 1000.

An examination of the accompanying map will show certain striking features characteristic of the entire field. The most pronounced is its extreme irregularity. It is very "spotted," to use the local phrase. Strips and patches of barren and productive territory are interwoven in a very intricate manner. In many cases dry holes are drilled in territory completely surrounded by productive wells, and within 400 ft. of a good well one may prove entirely barren. The map also brings out the fact that the pools follow a general northwest, southeast line. This, when extended, touches the Princeton-Indiana field to the south and the well-known Lasalle anticline to the northwest. It seems reasonable to believe that it is related in some way to this structural feature; the more so, since field observations show that within the oil territory there is a slight doming of the beds along this line. A third characteristic, not shown by the map, is the erratic distribution of gas. In Lawrence county the wells

so far opened show almost no gas. In the other counties a small amount of gas is found with the oil, not in general enough for pumping and drilling. In addition there are scattered gas wells which produce no oil. These wells have good pressure and volume, the latter running up to over 7,000,000 cu.ft. per day, but they seem to be short lived. Individual wells, when connected up and used, have lasted from a few days to three or six months. The gas is being used locally, but the present available supply does not encourage the laying of big gas mains.



PETROLEUM FIELDS OF ILLINOIS.

Differences Among the Districts.—In certain particulars the three districts differ. The Casey district is shallow, the wells ranging from 325 to 400 ft. The productive horizon is in the Coal Measures, the "sand" being in part a dolomite, resembling the Trenton rock of Indiana in appearance, and in part a true sand. Owing to the shallow depth and consequent low cost of the wells (approximately $1500) this field has been very rapidly explored. The principal gas production is from Johnson township. The wells here supply Casey, Martinsville and Marshall. In Crawford county a deeper sand is found, the depth being approximately 900 ft. Gas is

commonly found with the oil. The productive sand seems to be in the Chester formation, presumably representing the Birdsville horizon. It is often a sandy shale, and frequently consists of several beds of sand separated by shale. The Casey sand is thought to be recognized in several of the wells, approximately 240 ft. above the Crawford sand. It is not generally productive.

In Lawrence county three pools are recognized; the Lewis, Petty and Bridgeport. The first well was brought in about seven months ago, and there are now 160 wells producing. They range in depth from 1300 to 1450 ft. and the oil comes from a sand about 400 ft. below that of Crawford county. It is coarser grained, and so far is seemingly more regular. Of 40 wells drilled in the town of Bridgeport, only two proved dry. There is a strong salt water flow, and the oil pours out at the surface in a steady stream which is highly satisfactory.

Future Development.—As may well be imagined the discovery of these oilfields has led to a large amount of wild-catting in other parts of the State. Oil was found some years ago near Litchfield, in Montgomery county, and gas has been known at Sparta in Randolph and at Pittsfield in Pike counties. All these areas are being redrilled. Within the year small amounts of oil have also been found at Tolono, in Champaign county, at Iola, in Clay county, and at various points in Jersey, Saline and other counties. Next summer there will undoubtedly be a large number of drills going in outside territory, and it is expected that several new pools will be discovered. Just at present attention is being concentrated on a small pool being developed near Duncanville, in eastern Crawford county, and another north of Mermon, in Sullivan county, Indiana.

The geology of this area is as yet too little understood to warrant any attempt at prediction. A preliminary report, giving conditions up to May, 1906, was prepared for the Illinois Geological Survey by Prof. W. S. Blatchley, and issued as Bulletin No. 2. Further studies will be carried on in this field during the coming season.

Oil and Gas in the Mid-Continental Field.

By Erasmus Haworth.

The production of oil and gas in the Mid-Continental Field was much greater in 1906 than in any previous year. The oil was bought principally by the Prairie Oil and Gas Company, the Standard Oil Company in Kansas. At present there are nine independent refineries in Kansas and one in Indian Territory, but their consumption was small, due in part to their small size and in part to the fact that several of them did not begin operation until late in the year. A considerable amount of oil was used for fuel and was shipped to many places here and there over the State. One producer, C. A. Stannard, operated a system of tank cars and supplied

customers in western and central Kansas with a comparatively large amount of crude oil.

Month.	Total Run.	Daily Average.	Deliveries.	Stored.	Month.	Total Run.	Daily Average.	Deliveries.	Stored.
	Bbl.	Bbl.	Bbl.	Bbl.		Bbl.	Bbl.	Bbl.	Bbl.
Jan.	1,472,214	47,491	539,009	933,205	July	2,022,215	65,233	668,439	1,353,776
Feb.	1,352,531	48,305	527,995	824,536	Aug.	1,779,262	57,396	900,755	878,508
Mar.	1,693,182	54,619	719,320	973,862	Sept	1,546,719	51,557	846,128	700,591
April	1,779,251	59,308	752,625	1,026,626	Oct	2,009,650	64,827	1,559,520	450,130
May	1,741,941	56,192	802,838	939,103	Nov.	1,945,195	64,839	1,680,928	264,267
June	1,688,433	56,281	642,128	1,046,304	Dec	1,920,562	64,019	1,528,524	392,038

The total of the deliveries in 1906 was 11,168,209 bbl. The total runs were 20,951,155 bbl. (average of 690,067 per month), to which is to be added 973,750 bbl. for other sales, giving a total production of 21,924,905 bbl. in the Mid-Continental Field. No available method is at hand at present to determine definitely what portion of the above grand total given came from Kansas and what portion from Oklahoma. It is probable, however, that about one-fourth of the entire amount came from Kansas and three-fourths from Oklahoma.

Developments.—During 1906 the only new development of any considerable note was in the Glenn pool area, a few miles southeast of Red Fork. Here a wonderful pool of oil was discovered, which already furnishes many wells producing more than 1000 bbl. a day. Apparently the boundary of the pool has not yet been located, as new wells occasionally are found almost on all sides of it.

The next most remarkable discovery was in Section 27, east and a little south of Dewey, four miles northeast of Bartlesville. Here is an area nearly one mile across which was wonderfully productive in which also a few 1000-bbl. wells were brought in. Drilling was so active, however, that they soon had the boundaries of the pool located and the wells have now run down to 200 to 300 bbl. daily capacity.

A large amount of development was carried on during the entire year with an aggregate of close to 2000 wells drilled. During the first half of the year development was very active in the shallow pool around Alluwe and a very large production was obtained, some wells reaching 400 to 500 bbl. capacity. The next greatest production was in the Osage near the eastern boundary, south of Bartlesville. Next to the Glenn pool this area remains the most remarkable in the Mid-Continental Field. It seems that we have here a long, slender north and south pool lying parallel with the ninety-sixth meridian, which is the boundary between the Cherokee and the Osage lands, lying principally in the Osage, with a few good wells east of the boundary line and a few from five to 10 miles to the west. In a north and south direction the pool extends nearly to Tulsa, with only

occasional dry wells interspersed. A number of these wells started at 1000 bbl. per day and they held up their capacity as well, or better, than any other wells in the area. Farther west in the Osage drilling has been done irregularly with quite indifferent success. Some wells are good oil wells, others good gas wells, and quite a number of them are dry.

But little drilling was done in Kansas during 1906. Early in the year there was some development in the vicinity of Paola, Osawatomie and Rantoul, also a small amount in the Huffman field, Chautauqua county, five miles west of Peru, and an occasional well elsewhere here and there over the entire Kansas territory. But in general all the drilling done in Kansas was in searching for gas, as will be explained later.

Markets.—The Prairie Oil and Gas Company is the principal purchaser of oil, and, therefore, fixes the price. The lesser concerns consume but little, and, although occasionally they bid a little above the market price, they have little, if any, influence on the market. In fact, some of them can hardly obtain as large a quantity as they desire. There is a sentiment abroad that if a producer once sells to an independent concern the Prairie Oil and Gas Company will not take his supply or any fraction of it. It is probable this has not been put to a test, but the widespread belief that such would be the case has kept a number of people selling to the Prairie and declining bids from others.

At the beginning of the year the price paid was 52c. per bbl. for oil with a specific gravity of 32 deg. B. or lighter. During the summer this price gradually dropped until the low mark of 39c. was reached in August, at which point it has remained. For oils heavier than 32 deg. B., in most cases a reduction of 5c. was made for each half degree.

Gas.—The Mid-Continental Field has witnessed the most activity in developing and marketing gas. The Kansas Natural Gas Company got its pipe line laid to Kansas City, Kan., Topeka, Lawrence and Leavenworth before the close of 1905, and by February the pipes were in Atchison and St. Joseph, Mo. Certain difficulties regarding a franchise kept it out of Kansas City, Mo., until the autumn of 1906. By the close of 1905 also a pipe line had extended eastward from the gas fields to Joplin and Carthage in Missouri with laterals ramifying the entire Joplin mining region. During the summer of 1906 a pipe line was carried westward from the gasfields toward Wichita and Hutchison and has just now reached the former city. All of these mains have laterals from which they supply the various towns and villages along their lines.

To supply such large quantities of natural gas for domestic consumption it is necessary to have a large amount developed in the gas fields. On this account drilling was fairly active in many different parts of the gas fields of Kansas. The pipe line reaching northward to Kansas City, etc., is supplied principally from the vicinities of Iola, Humboldt, Chanute,

Neodesha and intervening locations; that eastward to the Joplin area is supplied from the Montgomery county fields in the vicinity of Independence and Coffeyville, and the supply for the western pipe line will come principally from western Montgomery, Chautauqua and other counties in Kansas, and probably from the Cherokee and Osage country, although that matter is not definitely settled. It is reported that a prominent gas producer applied to the Secretary of the Interior for permission to pipe gas out of Indian Territory and was denied the privilege. There seems to be no law, however, to prevent such an undertaking.

During 1906 the most extensive developments of gas were in Montgomery county, Kan., south of Independence and in the Cherokee Nation throughout a strip from five to 10 miles wide, reaching from the south side of Kansas southward almost to the Arkansas river. In fact gas has been developed in Indian Territory to such an enormous extent in connection with oil development that the Cherokee and Osage territories probably could supply two or three times as much gas as is developed in Kansas. A great deal of this Territory gas is shut in awaiting a market, while other portions are piped to various towns and villages and retailed for domestic consumption. Practically no large manufacturing concerns are established south of the State line. This is due principally to the chaotic condition of land titles in Indian Territory. Different laws of Congress and different rulings of the Interior Department have matters in such a shape that to the ordinary mind they are very confusing, and, as a result, capitalists prefer to remain away rather than jeopardize their holdings by insecure titles.

In Kansas, however, factories are on the increase so rapidly that the best judges are becoming alarmed lest the gas should become entirely exhausted. A number of zinc smelters in the State have enlarged their capacity and portland cement plants have increased in number and in size. Five such plants are now in operation, consuming from 40,000,000 to 50,000,000 cu.ft. of gas per day, and two or three others are in contemplation.

Gas wells near the southern end of Kansas and those in the Cherokee Nation generally are very large. A well from 10,000,000 to 15,000,000 cu.ft. per day is nothing unusual, and some of them run up to 30,000,000 or 40,000,000. The somewhat famous well in the Cherokee Nation, which caught fire in the spring of 1906 and burned for several weeks, was estimated at 30,000,000 cu.ft. before the gas was ignited, and before it was completely closed it was estimated at 60,000,000 cu.ft. capacity, although no accurate measurements were made. In the outlying borders, however, wells average much less and some of them are even less than 1,000,000 cu. ft.

Gas Markets.—The price of gas for domestic consumption along all of the long pipe lines is 25c. per 1000 cu.ft., meter rates. The Kansas Natural

Gas Company is offering gas to large factories at 10c. a thousand, but this is generally considered higher than coal and as a result but little coal has been driven out of the factories. In the gas fields a number of the towns and cities still pay the same flat rates they did years ago, that is, from 10 to 15c. each for lights per month and from $1 to $2 each for stoves. Also, a number of the factories, such as the glass factory at Coffeyville, the brick kilns at Coffeyville, Independence, Sycamore, etc., get gas at the old rates of 3c. per thousand. But no new contracts on this basis are sought for by the gas companies. Occasionally parties owning gas will make such contracts for factories or for large consumers. It must be said, however, that all indications are that the day of cheap gas for large factories in this territory is almost past.

Petroleum Refining in the United States.[1]

By Charles E. Munroe.

The total quantity of refined products reported separately on the census schedules for 1904 amounted to 50,638,490 bbl. of 50 gal. each, which would be equivalent to 60,283,917 bbl. of 42 gal. Deducting this from the 66,982,862 bbl. of crude petroleum reported as used by refineries there is a difference of 6,698,945 bbl. of 42 gal. each, or 5,627,114 bbl. of 50 gal. each. This difference represents to some degree the quantity of crude petroleum used in the manufacture of "all other products." In 1900 this quantity was represented by 1,731,644 bbl. of 42 gal. This volume is somewhat in excess of the true quantity used for "all other products," since all manufacturing processes are accompanied by some necessary waste and in the processes of refining by destructive distillation it is practically impossible to prevent the formation of gases and vapors which escape condensation.

Mabery, who has exhaustively examined a large number of samples from widely different sources, says:[2] "Now, after these years of arduous labor, I have reached the conclusion that petroleum from whatever source is one and the same substance, capable of a simple definition—a mixture in variable proportions of a few series of hydrocarbons, the product of any particular field differing from that of any other field only in the proportion of these series and the members of the series." However, from a commercial standpoint, petroleum from different localities is regarded as different substances. Thus, Pennsylvania petroleum is classed as an oil with a paraffin base; Texas and California petroleum, as oils with an asphalt base; petroleum from the Lima field in Ohio, as a sulphur oil; and so on.

The processes of refining may be roughly classified as follows: Treatment by settling to remove suspended matter and water; filtration; fractional

[1] Abstract from Bulletin No. 70, Census of Manufactures, 1905, Bureau of the Census, U. S. Department of Commerce and Labor.

[2] *Journ.* Am. Chem Soc., *1906*, *xxviii*, p. *417*.

PETROLEUM REFINING—DETAILED SUMMARY, BY STATES—1904.

	United States.	California.	Ohio.	Pennsylvania.	States.[1] All other
Number of establishments	98	19	12	34	24
Total capital	$136,280,541	$5 453,012	$10,384,741	$32,846 578	$87,596,210
Average number of men employed	16,770	678	1,900	4,227	9,965
Wages paid	$9,989,367	$477,118	$1,053,598	$2,371,027	$6,087,624
Miscellaneous expenses: total	$5,297,508	$146,719	$752,712	$1,218,572	$3,179,505
Materials used:					
Total cost	$139,387,213	$4,130,809	$7,662,397	$38,921,919	$88,672,088
Crude Petroleum—					
Barrels of 42 gal	66,982,862	4,369,600	4,195,871	17,977,686	40,439,705
Cost	$107,487,091	$3,431,754	$5,143,137	$31,957,135	$66,955,065
Sulphuric acid—					
Short tons	162,152	13,103	10,787	45,177	93,085
Cost	$2,003,031	$316,831	$120,594	$489,741	$1,075,865
Caustic soda—					
Pounds	11,161,376	469,929	1,005,404	2,680,308	7,005,735
Cost	$208,440	$10,018	$17,380	$54,987	$126,055
Sulphur—					
Short tons	888	843	43	....	2
Cost	$13,380	$11,437	$1,871	....	$72
Pyrites—					
Long tons	20,661	....	2,833	3,780	14,048
Cost	$79,784	....	$14,247	$18,910	$46,627
Coopers' and carpenters' materials, cost	$5,628,274	$6,542	$362,970	$569,458	$4,689,304
Tinners' materials, cost	$6,361,764	$10	$140,817	$1,001,930	$5,219,007
Barrels, cases, and tin cans (purchased), cost	$5,880,310	$124,852	$432,872	$1,613,024	$3,709,562
Fuel	$5,136,266	$191,386	$497,738	$1,543,851	$2,903,291
Rent of power and heat	$3,668	$1,377	....	....	$2,291
Mill supplies	$403,886	$2,368	$21,814	$133,991	$245,713
All other materials	$5,666,501	$21,977	$908,792	$1,187,618	$3,548,114
Freight	$514,818	$12,257	$165	$351,274	$151,122
Products consumed:					
Sulphuric acid, short tons	49,379	1,988	5,665	8,398	33,328
Products					
Total value	$175,005,320	$5,748,598	$10,948,864	$47,459,502	$110,848,356
Burning oils—					
Barrels of 50 gal	34,344,522	1,379,149	1,961,105	9,977,418	21,026,850
Value	$100,571,825	$2,641,916	$5,188,808	$28,412,940	$64,328,161
Residuum—					
Barrels of 50 gal	3,187,921	2,152,437	111,339	152,282	771,863
Value	$3,138,361	1,667,414	$210,911	$408,634	$851,402
Paraffin oils—					
Barrels of 50 gal	1,644,400	20,666	114,201	371,724	1,137,809
Value	$6,210,279	$120,077	$411,121	$1,173,727	$4,505,354
Reduced oils—					
Barrels of 50 gal	2,783,148	17,504	150,169	626,449	1,989,026
Value	$6,068,360	$57,602	$586,258	$1,684,169	$3,740,331
Neutral filtered oils—					
Barrels of 50 gal	504,042	114	300	253,874	249,754
Value	$1,942,153	$290	$2,900	$754,370	$1,184,593
Filtered cylinder oils—					
Barrels of 50 gal	1,366,661	15,734	71,446	618,390	661,091
Value	$9,332,299	$43,791	$540,617	$3,565,552	$5,182,339
Grease (lubricating, etc.)—					
Barrels of 50 gal	202,439	23,875	19,659	88,085	70,820
Value	$1,394,130	$79,594	$158,885	$377,137	$778,514
Naphtha and gasolene—					
Barrels of 50 gal	5,811,289	238,015	467,594	1,774,626	3,331,054
Value	$21,314,837	$926,063	$1,676,529	$6,402,492	$12,309,753
Paraffin wax—					
Barrels of 50 gal	794,068	3,898	47,533	279,511	463,126
Value	$10,007,274	$38,919	$549,515	$3,017,004	$6,401,836
Sludge acid—					
Short tons	165,104	18,045	....	38,216	108,843
Value	$400,480	$25,829	....	$140,627	$234,024
Coke and black naphtha, value	$149,653	$10,008	$30,448	$23,481	$85,716
All other products, value	$14,475,669	$137,095	$1,592,872	$1,499,369	$11,246,333
Equipment:					
Stills—					
Heated by steam, number	282	14	32	112	124
Heated by superheated steam, number	15	....	1	4	10
Heated by fire, number	1,610	92	184	400	934
Storage tanks—					
Crude petroleum, capacity in gal.	245,760,493	12,439,724	8,964,030	42,683,656	181,673,083
Refined petroleum, capacity in gal	576,458,825	49,410,383	84,888,935	110,111,758	332,047,749
Total horsepower	49,337	1,323	4,406	13,268	30,340

[1] Includes establishments distributed as follows: Colorado, 2; Indiana, 1; Kansas, 1; Louisiana, 1; Maryland, 1; New Jersey 4; New York, 5; Texas, 7; West Virginia, 1; Wyoming, 1.

distillation; destructive distillation; and, subsequent to or combined with the process of distillation, the treatment of the distillates, known as fractions, with acids or alkalies, or both, and sometimes with other chemicals; also, treatment by chilling, pressure, and filtration.

Refining by Settling.—The cleaning of crude petroleum by settling is the process used in the case of heavy, viscid oils found in loose sand of great fineness, since the sand and water become mixed with the oil and are pumped up with it. An example of this method is found in the practice obtaining in the Kern River oil field of California, where great difficulty is experienced from sand. It is customary there to pump the oil into small excavations made in sandy soil as close to the well as possible. These holes, called "sumps," are originally of from 500 to 2000 bbl. capacity, but they rapidly become shallower from the deposits of detritus from the oil. From the sumps the oil flows by gravity to storage reservoirs, which are shallow excavations made in the soil and covered with light wooden roofs. These reservoirs are often of great size, and it is customary to carry in them as large a quantity of oil as possible, in order that the sand may settle and the water separate completely. During the summer months the oil is continually at a high temperature and becomes clean by this simple treatment. During cooler weather, or whenever the reservoir purification is thought insufficient, the oil, before shipment, is passed through a small steel tank provided with steam coils, where it is heated for a sufficient time to remove these impurities. The degree of temperature, which is from 110 to 150 deg. F., and the duration of time, which is usually only a few hours, are determined by the specific gravity of the oil and the amount of impurities it contains. According to Prutzman,[1] "The high degree of purity which is obtained by the use of these simple methods is quite astonishing. Even where the impurities originally amount to 50 per cent. of the bulk of the crude oil, which is often the case, the oil finally shipped will not contain more than 2 per cent. of foreign matter of all kinds, and the larger part of the fuel oil in the San Francisco market, at least, will be found to contain less than 1.5 per cent. of impurity." This treatment results in other advantages, for the gas with which the oil is charged as it comes from the well, and which affects its gravity and flash point, is also very largely removed. It is interesting to note that when oils, such as these heavy crude oils, are exposed in shallow pools to sunlight in hot climates the oil is so oxidized that its gravity is lowered while its viscosity is raised, and the oxidation may proceed so far as to convert the oil into a tarry mass. If the dissolved gases be removed from the oils and sunlight be excluded, the effects of high natural temperatures are not serious, and it thus becomes possible to store oil for considerable periods in such reservoirs.

[1]Bulletin No. 32, California State Mining Bureau, p. 56.

Refining by Filtration.—Since the development of refining by distillation, filtration for the removal of color and odor has been confined largely to the denser natural oils which are used for the production of lubricating oils and which may lose some of the qualities that especially fit them for this purpose, if subjected to the conditions which obtain in the process of distillation. Crude oils which contain lubricating oils, but owing to the presence of volatile portions are too fluid for direct use, are reduced to the desired consistency by partial evaporation, either by exposing them in shallow tanks to solar heat, or by driving off the more volatile portions in stills, or both, and they may be further cleansed and purified before or after concentration by filtration.

Partial distillation is employed in the production of reduced oils. This process requires the use of shallow wooden tanks on the bottom of which flat steam coils are placed; water is run into the tanks to a depth of from 8 to 10 in., and a layer of oil 1 in. in depth is placed upon the water; the whole is then heated until the oil becomes very limpid, and this temperature is maintained until the desired specific gravity is reached. An advantage in this treatment, as in the method of settling described above, lies in the removal of every kind of dirt, especially the minute particles of grit which may have been held in suspension in the viscid oil, and if allowed to remain would seriously detract from the value of the reduced oil as a lubricant. Another method practiced in the manufacture of reduced oils consists in suspending sheets of loosely woven cloth vertically above troughs in a heated chamber, and through a perforated pipe spraying the crude oil upon the upper edge of these curtains. As the oil slowly descends the curtain it spreads out as a thin film, thus exposing a very large surface to the heated atmosphere of the chamber; and thereby the more volatile portions of the oil are rapidly driven off. At the same time the fiber acts as a filtering medium, retaining sediment and other impurities of the crude oil, so that the surplus oil, as it drips from the lower border of the curtains into receiving troughs, is not only reduced and rendered more viscid but is also purified and cleansed.

Refining by Distillation.—The distillation of crude petroleum is usually carried out in either cylindrical or "cheese box" stills. The size and the arrangement of these stills and the method of heating them vary with the character of oil to be treated and the products sought. The cylindrical still consists of a cylinder of boiler plate, 30 to 40 ft. in length and 12 ft. 6 in. to 14 ft. in diameter, the lower half of which is usually of steel. The still is set horizontally in a furnace of brickwork, which is usually so constructed that the upper surface of the still is exposed to the air. Stills are often set in batteries of from two to 10 for convenience in operation. The "cheese box" still has a body and a dome-shaped top made of boiler plate, and a double curved bottom made of steel plate. They may be

30 ft. in diameter and 9 ft. in hight, and they are set vertically on a series of brick arches. The working charge of the cylindrical still is from 600 to 1000 bbl., and of the cheese box still, 1200 bbl.

Either form of still may be heated by direct fires or by coils of steam pipes, either closed or perforated, which may be fitted into the stills to heat the oil, or by the injection of steam to facilitate, on the principle of partial pressures, the passing over of the distillates. Some stills are connected with an exhaust pump by which a vacuum may be maintained in them during distillation. The top of the still is usually provided with a dome into which the vapors rise and from which they pass to the condensers. The condensing apparatus consists primarily of long coils of pipe immersed in tanks through which cooling water flows. All the coils or lines are made to converge near the terminal so that they enter the receiving house within a few inches of each other. A trap is placed in the pipe near the end of each line for the purpose of leading off the gases or difficultly condensable vapors which are produced, and these are either collected for fuel or discharged into the atmosphere. The condensing pipes generally deliver the distillates into box-like receptacles, sometimes known as "sight boxes" because they have sides of plate glass through which the running of the distillate may be observed. As the character of the distillate changes, the delivery pipes from the condensers are turned to different receptacles. The receptacles lead to storage tanks or reservoirs, sometimes styled "cut tanks."

In addition to the above stills there are in use what are known as tar stills, which are made of iron or steel and are cylinders 10 ft. in diameter and 20 ft. in length. They are set horizontally in brickwork in batteries of two and are heated by fire. They hold about 260 bbl., and are provided above with air condensers leading from a dome in the center of the top of each cylinder. The condensers are of 4-in. iron pipe coiled in three turns one above the other in a rectangle of about the length and width of the still. As the coil makes one complete turn, it is trapped to carry off the products which have condensed at that stage.

The process of distillation varies with the circumstances, but the operation may take place as follows: The oil is first allowed to stand in large tanks so that part of the water and sediment are removed before it is pumped to stills into which live steam is introduced. Distillation commences at once and the distillate is collected in a receptacle continuously until its specific gravity reaches 0.74 (60 deg. B.). The condensers are now connected to another receptacle, and as the temperature rises and distillation proceeds the distillate is collected until its specific gravity reaches 0.81 (40 deg. B.). The heavy oil which remains is often distilled with superheated steam for the production of lubricating oil. The first original distillate is redistilled by steam and separated, commonly, into

five different fractions. The second original fraction may be subjected to a second distillation to drive off lighter oils, which are then added to the first original fraction. The third original fraction may be chilled so as to cause the paraffin present to crystallize. The semisolid mass of paraffin thus produced is then subjected to pressure to drive out the oil which is present. This oil is further subjected to the action of steam in a still to remove from it certain oils which possess a pungent and offensive odor. The distillates are then subjected to chemical treatment or to filtration, or to both if further refining be desired.

It was noted long ago in the making of coal gas that if the less volatile products from the distillation of the coal were allowed to condense and fall back into the hot retort, the liquid was decomposed into other substances, some of which were much more volatile than the original condensation product. This process of breaking up an organic liquid by heat is called "cracking." It is a process of destructive distillation and may be applied to many substances. It has long been applied to the treatment of petroleum, for by its use a larger portion of illuminants and oils of low boiling point may be obtained from a crude petroleum than is usually obtained by simple distillation only, since in the latter case a large percentage of the petroleum may remain as heavy oils or paraffins. In this process the operation of distillation is carried out as before, using fire, but when the second original fraction has been separated and collected, the fires are slackened and the distillation allowed to proceed slowly, in consequence of which the vapors of the heavy oil are repeatedly condensed upon the dome of the still and fall back upon the hot oil beneath, with the result that there is produced a large volume of gas, composed chiefly of marsh gas and hydrogen; a distillate of suitable specific gravity for the production of illuminating oil; and a heavy, tarry residue, called "residuum," which remains in the still. This residuum goes to the tar still, where, on further distillation by fire, there is collected at the first trap heavy distillate, at the second trap intermediate distillate, and at the third trap light distillate, while tar coke is left in the still. The first of the above distillates is wax bearing, and in the last part of its run this distillate is known as wax tailings; the third consists largely of illuminants, while the second distillate is of an intermediate character. All are joined to analogous materials passing through the refinery and are reworked by methods similar to those described above.

Oils that are to be subjected to chemical treatment to improve their color, or to remove components which might interfere with their use for particular purposes, are pumped from the cut tanks into the agitators. The latter are narrow upright cylinders with conical bottoms, generally lined with sheet lead and provided with an air blast descending from above and with outlets below for the spent chemicals and the treated oils. Agi-

tators may hold 50,000 gal. of oil at one charge. The reagents usually employed are concentrated sulphuric acid and caustic soda or other alkalies. The sulphuric acid forms sulphonic acids, and compounds with the unsaturated hydrocarbons, through which they become soluble in water, and may be removed. At the same time other of the components of the oil are oxidized, so that, as a result of the reaction, when the oil is mixed with acid by means of the air blast, the mixture becomes thick and black and there is an evolution of sulphur dioxide. The mass is allowed to stand and thereby separates into layers of oil and spent or "sludge" acid. The latter is drawn off and the oil washed by agitation with water. It is then treated with an alkaline solution, by means of which not only any free sulphuric acid but also any acid salts or other bodies present may be neutralized. After the alkaline solution has settled and been drawn off, the oil is washed until all traces of alkali are eliminated, and then it is drawn off to settling or sunning tanks. In special cases it may now be again distilled to fraction it more completely, or it may be treated in a steam still to reduce it.

The quantity of sulphuric acid required in refining petroleum and the length of exposure to its action depend on the original purity of the distillate and the purity sought. Cracked oils require more acid than uncracked, and the Ohio oils require more than the Pennsylvania. Usually the amount of acid required increases with the density of the distillate. It is, as a rule, added in repeated doses until the desired result is obtained. The acid sludge is sometimes treated so as to regain the sulphuric acid for re-use; at other times it is used in the manufacture of fertilizers and for other purposes in chemical manufacture.

In 1905, 165,104 short tons of sludge acid, having a value of $400,480, or $2.43 per ton, were reported as having been sold from the refineries. The weight of sludge acid thus accounted for formed 78.1 per cent. of the total sulphuric acid used in refining in that year. In 1890 there were reported 33,911 tons of sludge acid used for fertilizers and chemicals, and 19,962 tons for recovered sulphuric acid, the total, 53,873 tons, constituting 56.1 per cent. of the total sulphuric acid reported as used in refining petroleum at that census. In 1880 there were reported 22,163 tons of sludge acid used for fertilizers, and 21,159 tons for recovered sulphuric acid, the total, 43,322 tons, constituting 94.5 per cent. of the total sulphuric acid reported as used in refining petroleum at that census. The alkali sludge has sometimes been heated to destroy the organic matter present and recover the alkali, but this is usually found unprofitable.

When sulphur is present in petroleum, it is difficult of removal, so that special treatment must be given oils, such as Lima oil, which are high in sulphur contents. Some refiners effect this by distilling the petroleum over scrap iron and treating the distillate first with

an alkaline solution of lead oxide, and then with flowers of sulphur to remove the last traces of lead. Much the greater part, however, is distilled over copper oxide, the oxide being regained by burning off the sulphur. According to Mabery,[1] "Probably 50 tons of sulphur daily is a conservative estimate of the amount extracted from Ohio oil and burned off into the atmosphere. It is claimed for this process that it is capable of removing the sulphur to 0.02 per cent., which is probably correct."

Classification of Petroleum Products.

This is a matter of difficulty because the same name has at various times been given to different substances, some of which are not products of petroleum, and because the same material has been known by different names. With this introductory explanation, it may be stated that the term "naphtha and gasolene" embraces pentane, boiling point 100 to 104 deg. F., specific gravity 0.625; petroleum ether, boiling point 104 to 158 deg. F., specific gravity 0.65 to 0.66, which is sometimes known as Sherwood oil, and is used as a solvent for caoutchouc and fatty oils, and for carburetting air in gas machines; 76 deg. gasoline boiling point 158 to 194 deg. F., specific gravity 0.66 to 0.69, known also as 680 spirit, motor spirit, petrol, carburine, and boulevard gas fluid, and used in naphtha lamps and internal combustion engines, in the extraction of oil from seeds and fat from garbage and wool, and in carburetting water gas; naphtha, boiling point 177 to 230 deg. F., specific gravity 0.69 to 0.70, known also as Danforth's oil, ordinary spirit (when in the condition of untreated distillate), deodorized spirit (when purified), and city naphtha, and used as petrol in motor cars, for burning in vapor stoves and street lamps, as a solvent for resins in making varnishes, and in the manufacture of oilcloths; stove naphtha, specific gravity 0.70 or 70.4 deg. B.; ligroin, boiling point 176 to 248 deg. F., specific gravity 0.71 to 0.73 or 67 deg. to 62 deg. B., used as a solvent in the chemical laboratory and in pharmacy and for burning in sponge lamps; benzine (deodorized), boiling point 248 to 302 deg. F., specific gravity 0.73 to 0.75 used as a substitute for turpentine for cleaning printers' type and for dyers', scourers', and painters' uses.

Illuminating Oil.—The distillate collected between 60 deg. B. and 40 deg. B., or specific gravity 0.744 to 0.829, is crude illuminating oil. The refined illuminating oils are known by a large number of names, the most common general name being kerosene. The oils are graded by their color, their flashing point tests, their burning point tests, and their specific gravities. Water white oil of 120 deg. F. flashing point test, 150 deg. F. burning point test, and 48 deg. B. gravity is standard, but there are many other gravities of water white oil. Other grades, by color, are prime white, having a faint

[1] *Journ.* Am. Chem. Soc., 1906, Vol. XXVIII, p. 432.

yellow color, and standard white, having a pronounced yellow color. Water white oil of 36 deg. to 38 deg. B., and 300 deg. F. burning point is known as mineral sperm oil, mineral seal oil, mineral colza oil, coach oil, and 300 deg. oil, and is used as an illuminant in railway coaches and lighthouses, and for other purposes where readily ignitible oils are objectionable. Such an oil produced from wax oil, when pressed, and not lighter than 34 deg. B., is used in Pennsylvania in compounding miners' lamp oil. An oil having a specific gravity of from 0.85 to 0.86 and a flashing point above 100 deg. F. is known as gas oil. Stove oil is generally a cut from the crude still following the kerosene.

The lubricating oils vary so greatly as to be beyond description within reasonable limits. All should have high burning points, and a natural lubricating oil to be of real value must not ignite under a temperature of 325 deg. F. The lightest of the lubricating oils, varying in gravity from 32 deg. to 38 deg. B., are known as neutral oils, or when further purified by filtration through boneblack or fuller's earth, as they usually are, as neutral filtered oils. Heavier lubricating oils are styled "spindle oil" and "cylinder oil." The most important characteristics which distinguish these oils are high burning point, and viscosity, and low cold test. Cylinder oils are obtained by distilling the heavy oils, from which the naphthas and illuminating oils have been removed, with superheated steam, taking care that no cracking takes place. Or they may be produced by distillation in a vacuum. Paraffin lubricating oils are obtained by chilling the first distillate from the tar stills or other wax bearing distillates, these being chilled in the chilling house by cold brine from an ammonia ice machine. The chilled mass is pressed to separate it into paraffin and oil, and this oil is then redistilled and cut into several fractions. The common cuts for paraffin oil are one of 29 deg. to 30 deg. B., and a heavy cut of 23 deg. to 26 deg. B. To give the oils higher burning points and viscosities, they are cut from one-half to one degree higher than wanted and then reduced in a reducing still by means of steam and fuel.

Semisolid Products.—Besides the various oils, semisolid products, represented by vaseline, and solids, such as paraffin and petroleum coke, are obtained in petroleum refining. Vaseline is obtained by filtering heavy cylinder stock through boneblack filters until the required color is obtained; the first runnings from the filters, which are sufficiently light in color, may be used for vaseline, and the darker part used as filtered cylinder stock. To secure the necessary consistency and melting point, pure paraffin is melted and added to the filtered material. Rod wax obtained from the tube and rods of pumping wells and the salvy residues from oil tanks and pipe lines, which is known in the industry as B. S., are employed in the manufacture of these semisolid petroleum products. They are used directly as ointments or employed with drugs in the manufacture of oint-

ments and salves, and are styled in the United States Pharmacopœia *Petrolatum molle*, or soft petrolatum, the requirement being that they have a melting point of from 104 deg. to 113 deg. F. If the rod wax be pressed, it yields a solid with a low melting point and a salvy half-paraffin nature, which, either directly or when mixed with chicle or balata gum, is used as chewing gum. Paraffin, obtained from the chilling and pressing of the wax bearing distillates, preferably distilled at high temperatures to insure crystallization, is generally manufactured in three varieties, with melting points of 125 deg. F., 128 deg. F., and 135 deg. F., known as C., B., and A. paraffin, respectively. The *Petrolatum spissum*, or hard petrolatum, of the United States Pharmacopœia, should have a melting point of from 113 deg. to 125 deg. F. Thus petroleum paraffin stands next in the order of petroleum products as classified by the melting points. The paraffin is purified by chemical treatment like that for the other distillates, by filtration to remove color, by recrystallization from solution in benzine, and by sweating. This last is done by chilling it in cakes in shallow trays having wire-mesh bottoms. These are stacked in rooms, which are gradually heated. The paraffin, having the lowest melting point, becomes liquid, drips out, and is collected; with another increase in temperature and change of receivers the paraffin of the next higher melting point is obtained, and the operation is thus continued until the desired degree of separation is effected.

Paraffin.—Paraffin is used for many purposes in the arts. The harder varieties are used largely in the manufacture of candles, about 5 per cent. of stearic acid being added to prevent the candle from softening and bending. They are used also for finishing calicoes and woven goods and in laundry work to produce a luster. The softer varieties are used for coating jellies and fruits in preserving jars, for the preparation of translucent and waterproof paper, for waterproofing cloth, for mixing with stearic acid and wax in candle making, for impregnating the wood of Swedish matches, and as the absorbent in the process of enfleurage or extraction of the perfume from flowers.

Residual pitches are obtained in the distillation of the "asphaltic" petroleum of California, the "semiasphaltic" petroleum of Texas, and of some paraffin petroleum. The residues from California petroleum have been used to a considerable extent in the paving industry and are generally known as "D" grade asphalt or by some special trade designation or brand. According to Richardson[1] this "D" grade asphalt, when properly made, contains not over 10 per cent. of fixed carbon, while the asphalt from Texas oil contains a higher percentage of this constituent. The more liquid portions of these residuums are used in compounding sheet asphalt, in which they constitute from 12 to 50 per cent. of the composi-

[1]The Modern Asphalt Pavement, 1905, p. 253.

tion. For this purpose, California oil should be from 10 deg. to 13 deg. B.; Texas oil, 14 deg. to 16 deg. B.; and Eastern oil, 18 deg. to 22 deg. B.; and all should have a flashing point above 350 deg. F. to be suitable for use.

Other petroleum products used in the paving industry are Pittsburg flux, produced by heating a gallon of ordinary Pennsylvania petroleum residuum with about 1 ℔. of sulphur; Ventura flux, made by treatment of the California residuum in a similar manner; byerlyte, formed by oxidizing Pennsylvania residuum by sucking air through it, Byerly of Cleveland having found that oxygen, like sulphur, effected a condensation of the residuum; and hydroline B., produced by blowing air through the "asphaltic" residuum from Texas petroleum. These substances are prepared for use as fluxes for native asphalt. By blowing the crude oil with air similar oxidation and inspissation takes place. If the oil be warmed at the start, the oxidation not only maintains the temperature but causes it to rise to as high as 900 deg. F. Water vapor is evolved, though but little, if any, of the more volatile components of the oil are driven off. The blown residues thus formed are used for waterproofing, for paints, for rubber substitutes, and for use in the arts. Residuums are used in oiling dirt roads. Petroleum coke, which is the porous, brilliant black solid left in the tar stills, is used in the manufacture of electric light carbons.

Greases are semisolid to solid products used in lubrication. They are usually made by mixing a lime soap with a petroleum distillate. A rosin-lime soap and mineral oil produces axle grease or set grease, and a mixture of lime soap made from horse fat or cottonseed oil with mineral oil is styled "engine grease." Both may be mixed with lead oxide, mica, soapstone, or graphite. Wax tailings are used on the rolls in iron and tin-plate mills under the name of roll grease. Compounded oils are produced by mixing mineral oils with animal oils, such as neat's-foot, lard, tallow, sperm, and whale, or vegetable oils, such as rape-seed, olive, or palm, with soaps, such as lead and aluminum soap, and with solids, such as graphite, mica, and soapstone. A very common oil, known as mineral castor oil, is made by compounding an aluminum soap with petroleum distillate. The number of mixtures possible is well-nigh infinite, and a very large number of these have been made, offered in commerce, and used.

PHOSPHATE ROCK.

By Reginald Meeks.

The phosphate industry in general showed remarkable strength and activity in 1906, producers being generally unable to keep pace with the demand. Phosphate rock has been sold as far ahead as the end of 1907. The situation both here and abroad has become exceedingly tight and high grade rock is scarce.

Market and Prices.—The scarcity of material and cars, the heavy rainfalls and the incessant demand tended to elevate prices, and rock fertilizer was freely bought whenever offered. Buyers were forced to turn their attention to the low-grade rock, previously ignored, and they bought all they could get of it. Florida hard rock analyzing 77 to 80 per cent. bone phosphate of lime advanced from \$7.50@7.75 to \$8@9 per long ton, f.o.b. Fernandina, Fla., and \$14.30@15.80 c.i.f. European ports. Land pebble advanced from \$4 to \$6 toward the close of the year. European quotations were \$11.40@12.20 c.i.f. Tennessee export rock, 75 per cent., brought from \$3.50@5.50 f.o.b. Mt. Pleasant, and \$12.85@13.60 c.i.f. European ports. South Carolina land rock, 58 to 63 per cent. brought \$4@4.50 f.o.b. vessels, Ashley river; and river rock, 55 to 60 per cent. brought \$3@3.75; the value c.i.f. European ports was \$8.25@8.50.

IMPORTS OF FERTILIZERS INTO THE UNITED STATES.
(In tons of 2240 lb.)

Year.	1903		1904		1905		1906	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Guano	21,007	\$251,966	35,876	\$478,388	25,651	\$365,823	22,947	\$320,565
Crude phosphates	132,965	697,112	130,214	745,744	56,421	275,880	23,281	147,547
All other fertilizers	....	2,353,496	....	2,856,141	....	4,048,403	....	4,231,723
Of which— Kieserite and Kainite	158,313	773,758	218,957	1,050,082	....	....	....	....

STATISTICS OF PHOSPHATES IN THE UNITED STATES. (*a*)
(In tons of 2240 lb.)

Year.	Production	Imports.	Exports. (*b*)	Consumption.	Year.	Production	Imports.	Exports.	Consumption.
1897	1,007,367	12,950	551,046	469,271	1902	1,600,813	145,793	802,086	944,250
1898	1,257,645	71,388	570,948	758,085	1903	1,581,576	153,972	785,259	950,289
1899	1,663,476	118,613	867,790	914,299	1904	1,874,428	166,090	842,484	1,198,034
1900	1,527,711	144,006	619,995	1,051,722	1905	1,933,286	82,072	934,940	1,080,418
1901	1,483,723	180,714	729,539	934,898	1906	2,052,742	46,228	904,214	1,194,756

(*a*) Production statistics of 1901 and subsequent years, except 1905 and 1906, are those of the Geological Survey and are based on marketed products. (*b*) Neglecting the insignificant re-exports of foreign product.

PRODUCTION OF PHOSPHATE ROCK IN THE UNITED STATES. (*a*)
(In tons of 2240 lb.)

Phosphate.	1903.		1904.		1905. (*b*)		1906. (*b*)	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
Florida hard rock..	214,876	$1,988,243	531,087	$2,672,184	579,228	$4,198,210	561,370	$4,771,645
Florida land pebble	390,882	885 425	460,834	1,102,993	401,997	1,660,248	603,382	3,620,292
Florida river pebble	56,578	113,156	81,030	199,127	90,065	371,968	41,742	187,839
Total Florida....	860,336	$2,986,824	1,072,951	$3,974,304	1,071,290	$6,230,426	1,206,494	$8,579,776
S. Car. land rock...	233,540	$721,303	258,806	$830,117	221,712	$915,671	270,000	$1,215,000
S. Car. river rock...	25,000	62,500	12,000	31,200	30,284	117,199	45,000	168,750
Total S. Carolina ..	258,540	$783,803	270,806	$861,317	251,996	$1,032,870	315,000	$1,383,750
Tennessee.........	460,530	$1,543,567	530,571	$2,037,804	610,000	$2,450,000	520,381	$2,341 715
Other States........	2,170	5,100	100	200	......	....	10,867	37,500
Total United States	1,581,576	$5,319,294	1,874,428	$6,873,625	1,933,286	$9,713,296	2,052,742	$12,342,741

(*a*) Statistics for 1903 and 1904, are those of the U. S. Geological Survey. (*b*) Statistics of 1905 and 1906 are by J. M. Lang & Co., Savannah, Ga., and are based upon shipments, the values being based upon quotations at ports of shipment.

PRODUCTION OF PHOSPHATE ROCK IN THE WORLD.
(In metric tons.)

	1899	1900	1901	1902	1903	1904	1905
Algeria..............	324,983	270,920	269,878	260,859	301,112	344,969	334,784
Aruba (Dutch W. Indies).	(*c*)	(*c*)	(*c*)	10,530	15,511	22,764	(*c*)
Belgium.............	(*a*)190,090	(*a*)215,670	(*a*)222,520	(*b*)135,850	(*b*)184,120	(*b*)202,480	193,305
Canada..............	2,722	1,284	937	776	1,205	832	1,180
Christmas Island.....	(*c*)	(*c*)	42,125	61,179	70,096	71,757	97,952
France..............	645,868	587,919	535,676	543,900	475,783	423,521	476,720
French Guiana.......	(*c*)	(*c*)	(*c*)	4,230	7,769	(*c*)	(*c*)
Norway.............	1,500	300	738	2,295	1,795	1,456	2,522
Redonda (British W. Indies).	1,507	2,230	*Nil.*	132	1,102	1,729	(*c*)
Russia..............	16,863	25,663	21,276	13,709	(*c*)	(*c*)	(*c*)
Spain...............	3,150	4,170	4,220	1,150	1,124	3,505	1,370
Sweden.............	(*c*)	(*c*)	(*c*)	3,895	3,219	2,929	(*c*)
Tunis...............	63,209	171,288	178,018	263,482	332,888	455,789	559,645
United Kingdom.....	1,469	630	71	87	71	59	*Nil.*
United States........	1,539,953	1,515,075	1,507,548	1,514,159	1,606,881	1,904,419	2,135,449

(*a*) Cubic meters. (*b*) Metric tons of phosphate of lime; in addition there were 315,200 cu.m. of phosphatic chalk in 1902, 350.250 cu.m. in 1903, 311,640 cu.m. in 1904 and ...... cu.m. in 1905. (*c*) Statistics not available.

Exports and Imports.—The exports of crude phosphate in 1906 amounted to 904,214 long tons ($7,373,945) as compared with 934,940 tons ($7,465,592) in 1905. In addition to this there were exported 31,999 tons ($1,088,004) of guano, dried blood, bones, etc., classified as "all other." Germany stands first in importance as an importer of American phosphate, shipments to that country amounting to 272,785 tons. Great Britain imported 149,489 tons, France 116,357 tons and Italy 93,744 tons during 1906. Great Britain and France showed slight increases, but Germany and Italy fell off sharply in their imports of phosphate rock. About one half of the phosphate exported from the United States is Florida hard rock and a large proportion of the remainder is Tennessee product. South Carolina phosphate is rarely exported.

Imports of Peruvian guano were practically the same as in 1905, but

crude phosphates suffered a marked decline in 1906. Since 1904 they have fallen off 82.2 per cent. which may be seen from the table of imports.

Review by States.

Arkansas. (By A. H. Purdue.)—The phosphate deposits of northern Arkansas occur over the area between the towns of Hickory Valley, northeast of Batesville, Independence county, and St. Joe, Searcy county. The phosphate horizon which is practically flat, outcrops on the hill-sides above the streams. As prospecting and careful geological field work have been confined to that part of the area lying within Independence county, it is not known whether or not workable deposits occur to the west of this area.

Only one company, the Arkansas Fertilizer Company, is operating in the region. The developed deposits are near the mouth of Nafferty creek, in the western part of Independence county. Operations were begun in 1900, when a phosphate plant was established at the mines. After a few months the plant was destroyed by fire, but later it was decided to erect a new plant at Little Rock and to ship the crude phosphate there for manufacture into acid phosphate. The new plant, which has an annual capacity of 40,000 tons of acid phosphate, was not completed until the latter part of 1906.

The phosphates occur as bedded deposits and probably are of Ordovician age. The total thickness of phosphate rock usually is several feet, but the only bed sufficiently rich to work with profit is from 18 in. to 6 ft. thick. Within the vicinity of the developed deposits, this bed is a compact, homogeneous, light-gray rock. The color is due to small white particles thoroughly mixed with dark-gray material. The white particles apparently are the ground-up fragments of shells and to them probably is due the phosphatic character of the rock. The dark-gray material consists of small, angular to subangular fragments, which give the rock, on close inspection, a conglomeratic appearance.

Only about 3000 tons of crude phosphate were mined in 1906, but with the completion of the new plant, the output for 1907 should be considerable.

Florida.—The phosphate business in Florida in 1906 was generally prosperous. Supplies continued to fall short of the demand. At the end of the year there was nothing offered for sale before the last half of 1907, while recent prices were being fully maintained, and in some cases considerable advances were being asked for by the miners. There was much activity in the erection of new plants, but owing to labor conditions, poor railroad facilities and much rainy weather the production in 1906 showed a decrease from that of 1905.

In the pebble district of Polk county, the Phosphate Mining Company and the Memminger Phosphate Company started the construction of new plants, both near Mulberry, early in 1906. A tract of phosphate land on Peace river, in the same county, was sold to W. B. Chisholm and associates, while P. B. and R. S. Bradley of Boston, Mass., purchased what are known as the Disston lands, consisting of 6090 acres, 10 miles south of Bartow, Polk county. There were many unauthenticated rumors of transfers in the same district throughout the year. The Mutual Mining Company of Savannah, Ga., was incorporated to develop phosphate and timber lands in Citrus, Sewanee, Marion and other countries. Col. Albertus Vogt of Bartow, Fla., and associates purchased two tracts of lands in Citrus and Hernando counties and declared their intention of erecting plants. The tracts comprise 280 and 1365 acres respectively.

The Dutton Phosphate Company operated its new plant and dredge near Floral City, Citrus county. The Standard Phosphate Company of Lakeland, Fla., completed its buildings early in 1907 and expects to produce about 75,000 tons of phosphate rock. The American Agricultural Chemical Company acquired the entire capital stock, ($300,000) of the Pierce Phosphate Company, which owns 2000 acres of phosphate land.

In the pebble district the Charleston Mining and Manufacturing Company is erecting a large plant near Fort Meade. The Electric Phosphate Company, having recently lost its plant by fire, has sold its land. The Swift Fertilizer Company is arranging to erect a plant near Bartow and is operating a saw-mill to supply the necessary lumber.

According to John Bannon,[1] the pebble phosphate-mining industry in Polk county, Fla., has grown to large proportions. Within an area of 30 miles there are nine large mining plants, the annual output of which, in the aggregate, exceeds 500,000 tons. A tenth plant is now in course of construction, and provision has been made by a Chicago packing-house for the installation of another. Still another Chicago packing-house has secured 600 acres in Polk county which contain phosphate deposits, and will clear the overburden and mine. It is estimated that the visible supply of phosphate in Polk county will last 25 or 30 years; this refers only to land now in course of development. The phosphate deposits are found in beds varying from a few inches in thickness to 25 ft. with a fair average of about 9 ft. The phosphate stratum lies between the overburden, which consists of a sandy clay and earthy conglomerate material, and a yellow sticky variety of clay beneath.

The grade of Florida land pebble which is classed as merchantable must contain not less than 66 per cent. of bone phosphate of lime nor more than 4 per cent. of oxide of iron and alumina on dry basis. In an

[1]*Manufacturers' Record*, May 3 and 10, 1906.

average deposit the first named will range, as a rule, between 68 and 70 per cent., while some deposits produce a grade as high as 75 per cent. of phosphate of lime, but this is rare. The guarantees under which land pebble are usually sold are, for the export trade 68 per cent. flat, or 70 per cent., with fall to 68 per cent. bone phosphate of lime, 3 per cent. oxide of iron and alumina combined, absolute maximum of 4 per cent., with a reduction of two for one from bone phosphate of lime for the excess of iron and alumina over 3 per cent.; moisture 3 per cent. The domestic trade demands guarantees of 68 per cent. flat, or 68 per cent., with fall to 66 per cent. bone phosphate of lime, 4 per cent. oxide of iron and alumina and 2 per cent. of moisture.

The scarcity and increased cost of labor combine with the greater cost of mining to check the response that otherwise might be made to the greater demand and higher prices. The valuable deposits of high-grade phosphate rock are in the hands of comparatively few people. These holders constitute strong companies which are acting more and more in harmony with one another, and may be expected not to produce more rock than is needed for the actual requirements of their trade. The Winston & Bone Valley Railroad, which extends from Lakeland some 25 miles through the phosphate region, has located upon it the following plants: The Electric Phosphate Company; the Prairie Pebble Company, which operates three plants; the Palmetto, which also operates the Tiger Bay; the Pebbledale, the Green Bay and the Dominion phosphate companies. These nine plants produce approximately 400,000 tons of phosphate per annum, and give employment to from 1500 to 2000 men.

Idaho. (By Robert N. Bell.)—Important discoveries of phosphate rock have been developed to considerable extent recently in Bear Lake county, the extreme southeastern county of Idaho, and 6000 tons of the mineral were shipped during 1906. The material occurs in a series of "oolitic" beds interstratified with limestone and bedded shales of upper Carboniferous age. One of the most extensive openings on these deposits is at a point four miles east of Montpelier, a station on the Oregon Short Line, where a series of flat dipping beds of phosphate rock occur that range from a few inches to 5 ft. in thickness. A large part of the shipments of 1906 were from this point and averaged 70 per cent. bone phosphate. The lowest bed of the series at this point is 5 ft. thick and is operated like a flat lying coal seam; the rock is dark colored and its remarkable uniformity, both in thickness and grade, permits of its being mined its full width without sorting. The shipments from this point were sent to the San Francisco Chemical Company at Martinez, Cal.

There are several companies operating in this field at the present time. The same series of beds has been traced for miles, in places expanding

to 10 ft. in thickness, and extending into the adjoining States of Utah and Wyoming. Some extensive tracts of phosphate-bearing territory have been acquired by the different companies under the mineral land law, most of the territory having been public land. The deposits are of sufficient extent to warrant the anticipation that they will eventually become the basis of an important industry. At present they are at a geographical disadvantage, because the market for fertilizer material on the Pacific slope is limited. However there is a good demand in Honolulu, Japan and Australia and the immediate future of the new industry is largely dependent on the railroads granting low enough rates to Pacific tidewater to enable manufacturers of superphosphates to reach the foreign markets.

Pennsylvania.—According to George W. Stone in a bulletin of the U. S. Geological Survey, a deposit of wavellite, phosphate of aluminum, occurs near Mount Holly Spring, Pa., about 10 miles southeast of Harrisburg. The ore is found in white clay associated with manganese and iron ores. The American Phosphorus Company was organized in Philadelphia to work the deposit and a mill was built in the vicinity. Mining and reduction, in a small way, have been carried on since 1900. Other small deposits of wavellite have been found within a few miles of the original discovery.

South Carolina.—Reports from this State showed a satisfactory condition in the phosphate market during 1906. Producers were active and shipments were steady. The phosphate rock in South Carolina occurs in the form of nodules, found both on land and in river beds. In size these nodules range from lumps the size of a man's head down to that of a pea. Mining land pebble is usually accomplished by steam shovels which remove the overburden of sand, 6 to 10 ft. thick, and then the phosphate pebbles and intimately mixed foreign material. This mixture is screened, dried in kilns or burned in piles and marketed. The river pebble is mined with dredges and then treated the same as land pebble. The rock varies from 55 to 60 per cent. bone phosphate of lime and contains about 6 per cent. iron and aluminum oxides,

Tennessee.—In 1906 there was much activity throughout the phosphate fields. The producing companies were handicapped by scarcity of labor and by weather conditions early in the year. The demand for high grade rock was greater than producers could supply. A number of new incorporations were recorded and the year 1907 should see a larger number of operators in the field.

(By H. D. Ruhm.)—The Tennessee phosphates occur almost entirely in Silurian and Devonian strata, but more particularly in the former, and in the transition strata between the two. In December, 1893, blue phosphate rock was discovered in Hickman county. A few companies

struggled through 1894, 1895 and the first half of 1896, digging "rabbit burrows" under the hills and stripping back to "face up" their mines. This had the inevitable result of breaking away the main face of the hill, rendering it worse for permanent work than if it had never been touched. Two companies constructed and began to operate narrow-gauge railroads, having grades up to 5 per cent. and curves as high as 50 deg. With these facilities, supplemented by all the wagons they could get to work, the several companies managed to produce 19,188 tons in 1894, 38,515 tons in 1895 and 17,000 tons in the first half of 1896.

All that time, the blue-rock hunters were walking over the brown rock of Indian and Swan creeks and Totty's bend, and prospectors were daily getting off the train at Mt. Pleasant, and driving out to the blue-rock field on Big Bigby creek; they even mined some of this rock, hauling it to Mt. Pleasant and shipping it, never dreaming they were walking and driving every day over the finest deposit of phosphate ever known to the world.

Prior to the discovery of the blue rock, Florida phosphate of about the same grade was selling in the interior at about $6 per ton for 65 to 70 per cent. on the basis of Centerville freight rates. The first sale of Tennessee phosphate was at $4.65 per ton by the old Southwestern Phosphate Company. Competition reduced the price to $2.25 per ton. The hight of folly was accomplished when Louisville and Birmingham parties purchased the Tennessee Phosphate Company holdings on upper Swan creek, expecting to construct a road from the N. & F. division of the Louisville & Nashville Railroad down Swan creek, and thus to cut out the expensive haul of the other mines, which in most instances amounted to 75c. per ton. They also expected, by using machinery, to lessen the cost of production, so they sold 100,000 tons at $1.95 per ton in order to freeze out the small producers. Some of the small producers had in the meantime, however, been holding tight this precious knowledge of the nature of the Mt. Pleasant deposit, and about the time the above action had already put the blue-field companies into bankruptcy or worse, the small producers purchased a few acres of land and calmly went to mining brown rock at Mt. Pleasant at a cost of about 85c. per ton, f.o.b. cars, shipping it on their old contracts at $2.25 for blue rock.

The manufacturers, however, had bought blue phosphate rock guaranteed 65 per cent., and they maintained that they did not want brown phosphate rock even though it ran 75 per cent., unless a reduction was made in the price. The reduction was, of course, promptly forthcoming, for the reason that the parties who had shipped the rock were greatly in need of the 85c. per ton it had cost them to produce and load, and when a compromise of $1.25 per ton was proposed, it was eagerly accepted.

The manufacturers immediately began to stipulate for a 75 per cent.

guarantee and for rock containing not more than 5 per cent. of iron oxide and alumina. The people who had invested in the blue-rock regions were disgusted with the whole phosphate business, and it was soon found that the field was too extensive to allow concentration of ownership. Some large interests did, however, move into the Mt. Pleasant field, bought one of the best prospects at $100 per acre and began operations by reducing its blue rock 65 per cent. prices to $1.50 and $1.40 for 75 per cent. brown rock. For a long time thereafter $1.10 was the ordinary selling price, and $1.25 the maximum.

Slowly the miners began to find out that it cost more to produce than they had figured on, and prices advanced, until in 1898 rock was sold at $2 per ton for 75 per cent. In the meantime the export market had been tested and the guarantee for this was made 78 per cent. bone phosphate of lime, and 3 to 4 per cent. iron and alumina. The first price at which this grade of rock was shipped was about $1.65 per ton, and in not very many months it had come up to $2.25 per ton. During the fall of 1898 and spring of 1899 an era of activity was seen. Property changed hands at a lively rate; laborers flocked to the field, everybody mined all he could get out, regardless of sales and shipments, and still the prices climbed, until in the fall of 1899 and spring of 1900 export rock brought as high as $4 per ton and domestic $3. The fury of production was redoubled and simultaneously the extent of the Maury county field was ascertained to be much greater than had been thought possible. About this time the Boer war broke out and as three-fourths of the carrying trade for phosphate was done in English vessels, the export business came to a standstill, and the export rock, not only of this field, but of Florida, was dumped on the domestic market.

In the meantime first the Southern trust and then the Northern trust was formed, the two companies comprising most of the fertilizer factories in the United States. These two concerns partially supplied their wants by purchasing phosphate rock property, but failing to supply their entire requirements, attempted to get them at their own prices later by depreciating the prices of their product.

This depreciation, coupled with the enormous increase in the use of fertilizer, caused the starting up of a large number of independent factories, and consequently an increasing number of buyers of Tennessee phosphate rock. Also the packing-house people being unable to supply the demand for their pure blood and bone fertilizers, and finding that the quality was very much improved by the use of high-grade rock, became large customers of the Mt. Pleasant and other Tennessee fields. Numerous cotton-oil mills having been purchased by the Southern trust, many more were started up, and these all being producers of cotton-seed meal, largely used in the manufacture of fertilizer in the South, they in turn

became fertilizer manufacturers and users of Tennessee phosphate rock. Also experiment stations in several States had been making actual experiments with raw ground phosphate rock without acidulation, as a direct fertilizer, and instead of reaching the position taken by the Tennessee station that it was worthless and that its sale should be prohibited, they arrived at the directly opposite conclusion. Lastly, with the end of the war, the export trade expanded more strongly than before. The weak condition of the producer was thus changed to one of great strength.

The first thing that impresses itself on any visitor to the phosphate fields is the almost universal dependence on hand labor. This is partially due to the fact that they "just started that way," and partially to the fact that the varying conditions met with in the deposits make it difficult to devise appliances suitable for one portion of a mine that will answer the requirements in closely adjacent portions. For instance, it is possible in the same open face of a mine to find the overburden varying from 2 ft. to 20 ft. and the rock from a few inches thick, sticking tight to the top of a lime boulder, to 15 ft. in the "dip between two boulders," while the rock itself will vary from the shaly, partially disintegrated top rock through various sizes to heavy blocks 6 to 8 in. thick and often 10 or 12 ft. long.

It will therefore be seen how difficult it is to design a machine that will accommodate itself to the handling of this material. The removal of the overburden has been generally accomplished with wheel scrapers. Two companies have used the New Era or Western machine plow, with elevator belt, loading the dirt into dump-bottom wagons alongside. Two steam shovels are now in use, being of the traction type, and occasionally these have been used in digging the rock, though apparently with not entire success. Cableways have never been used to transport the material, and this is done largely by wagon and team, though many tram roads, with cars propelled by either mules or dinky engines, are in use.

The bulk of the rock, however, is dug by the miner with pick and fork, loaded into wagons, hauled out and dumped in windrows on the ground, stirred with a potato plow and harrows, allowed to dry in the sun, taken up again into wagons and hauled either direct to cars for shipping or put under sheds for storage. When an extra good quality of rock is wanted, as for export, a few layers of cordwood are put down and the sun-dried rock put on that. Then, when ready to ship, the wood is fired, and after the rock is cool it is broken and loaded with forks, when most of the dirt sloughs off, leaving the rock almost perfectly clean.

Some rock can be put from the mines immediately on the wood and burned for export, but generally this will only be a safe domestic rock. Some companies which have water accessible pick out the large pieces and send them direct to the dry kilns, and then the small pieces with

the dirt, known as "muck," are passed through washers, the rock coming out clean and being deposited on cordwood and burned as above described.

The resulting rock, after being crushed, is passed through screens which separate it in three sizes—from 1½ in. up going for export, that between 1½ and ¼ in. for domestic, and the dust and ¼-in. pieces being ground up and sold for direct use, or to small factories.

The Century Phosphate Company has installed a system of driers, and does not wash the rock, but dries it thoroughly in mechanical driers, and then screens it as above. Miners are generally turning their attention to labor-saving devices for doing the preliminary operations, and to systems for saving the valuable products that have heretofore gone to waste in large quantities. At least 10 per cent. of the present output is thrown away to prepare the high-grade rock now demanded. I should estimate that this waste rock can be made into acid phosphate at a cost such as to yield an annual net profit of $162,500. This excess fertilizer could profitably be employed in enriching the barren lands whose cordwood has been consumed in drying phosphate rock at Mt. Pleasant.

The principal localities in the State where operations are now in progress, are: Mt. Pleasant, Kleburn, Jameson and Century, in Maury county; Lower Swan creek, Twomey and Totty's Bend, in Hickman county; near Gallatin, in Sumner county; Wales station, in Giles county, and near Mashville, in Davidson county.

The principal localities where developments will gradually take place as the demands of the business require are: Southport, Estes bend, Bear creek, Neeley's valley, Little Bigby, Westfork, Baptist Branch[1] and Leiper's creek,[1] in Maury county; Richland creek, in Giles county; Station Camp creek, in Sumner county; north and west of Franklin, in Williamson county; Brentwood and Bellevue, in Davidson county; Beech river, in Decatur county; Tom's creek, Buffalo river, Hurricane creek[1] and Cane creek in Perry county; Forty-eight Mile creek,[1] in Wayne county; upper Swan[1] and Indian[1] creeks, in Lewis county; Lower Swan,[1] Indian creek,[1] Ship's bend, Gray's bend, Persimmon, Haleys and Leatherwood[1] creeks, in Hickman county.

The total quantity of phosphate rock of 60 per cent. and better grade now available in the Tennessee field is estimated at 44,800,000 tons, and of this the State may be counted on to produce 35,000,000 tons.

Of the present working localities the principal is Mt. Pleasant, and while the property owners there are beginning to figure a little on how much they have left, still the prevailing impression that Mt. Pleasant is about through mining is an exceedingly mistaken one. With the present rate of output, the visible supply of the Mt. Pleasant field proper

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[1]Blue Rock.

will last for seven years longer, without taking into consideration the Southport field, which is practically part of the Mt. Pleasant field. With the Southport field, mining will last, at the present rate of output, for 11 years. It is very easy to understand that as work progresses at Mt. Pleasant and the end approaches, some miners drop out by selling, some by working out their small deposits and these naturally go to the other fields. In none of the other fields is found the persistently uniform, high-grade brown rock of Mt. Pleasant except Southport, Century and Kleburn. Operations are now in progress at Century and Kleburn, while at Southport the extension of the Mt. Pleasant Southern Railway will soon cause development work there. As these deposits afford practically the same grade of rock as Mt. Pleasant proper, they will be worked out simultaneously with it and will cater to the same market.

The producers of this character of rock are gradually increasing their prices and reducing their output, thus giving opportunity for the marketing of the lower grades in the other fields, notably the Swan creek and Indian creek deposits in Hickman county. This means that the producers at Mt. Pleasant will make more money from their product, and that it will last a considerably longer time, so that it is safe to say that mining in force will be carried on at Mt. Pleasant and kindred localities for at least 20 years.

During the next decade, to supply the diminution of Mt. Pleasant's output, will come the gradual development of the vast blue-rock field of Maury, Hickman and Lewis counties, and the white rock of Perry and Decatur counties, which form the backbone of the phosphate industry in Tennessee, and whose millions of tons will cause those counties to be considered the phosphate reservoir of the world for the next 75 or 100 years.

The change of base will be gradual, and the trade will have ample opportunity to adjust its operations so as to utilize the lower grade blue rock as it becomes advisable and necessary to do so. Its many points of superiority for acidulation and for direct use without acidulation will largely make up for its lower grade.

The blue-rock field proper covers a territory bounded approximately by a trapezoid having as its four corners Centreville, in Hickman county, Kinderhook and Mt. Joy, in Maury county, and Lewis Monument in Lewis county. Traversing this territory are Duck river, Indian, Swan, Blue Buck and Cathey's creeks, and their tributaries, and outcropping along these valleys and underlying the ridges between them are deposits of blue rock running in bone phosphate from 60 to 78 per cent., with less than 3 per cent. iron and alumina, that will aggregate in the neighborhood of 40,000,000 tons.

This field will soon be developed by the extension of the Nashville,

Chattanooga & St. Louis branch up Swan creek and the Louisville & Nashville branch down Swan creek, with side lines and spurs leading off each, surveys for which have been made, and work on construction will soon be under way. If, however, the Florence Northern Railroad should ever be built from Florence to Nashville it will run through the heart of this territory as well as the magnificent iron deposits of Wayne and Lewis. With the above road and a road from Huntsville on the southeast to Milan on the northwest, all of the phosphate territory would be fully developed, and this section of Maury, Hickman, Lewis, Perry, Giles, Davidson and Williamson counties would be the site of more fertilizer factories than will be found elsewhere in the world in the same space.

Phosphate Mining in Foreign Countries.

Algeria.[1]—The principal deposits or beds in the province may be divided into four groups, viz : (1) District of Tebessa; (2) district of Setif; (3) district of Guelma; (4) district of Ain-Beïda. The beds appear under two different conditions, viz : as nodules or in regular layers over a length of several miles. The rock phosphate is of two qualities; the first grade contains 63 to 70 per cent. phosphate of lime and the second 58 to 63 per cent. The mines of Djebel-Kouif, situated 16 miles from Tebessa, at nearly 5000 ft. above sea level, formerly belonged to a British company, but were sold to a French company six years ago for £200,000. A private railway of 18 miles connects these mines with the railway at Tebessa. The phosphate is principally extracted from open quarries, gallery work being adopted only when it becomes too expensive to remove the top of the mountain to uncover the phosphate. The beds (of varying thickness) cover an area of about 12 miles. The cost of the phosphate on trucks at Tebessa station is about $0.96 per ton. The cost of hauling from Tebessa to Bône, which is $1.48 per ton, is further increased by the necessity of transferring the phosphate at Souk-Ahras from the narrow gage line of Tebessa to the broad gage line between Souk-Ahras and Bône. Altogether, the cost of a ton of phosphate ex-dock at Bône is $2.64@2.88.

French Guiana[2].—There are deposits of phosphate of alumina in the islands of Salut and Connetable off the coast of French Guiana. These deposits were discovered by Americans and are being developed by them. The mineral is a high grade phosphate of alumina, similar to wavellite, and is found white, yellow and sometimes pink; it is somewhat translucent.

Like phosphate of lime, the natural phosphate of alumina, of the island of Connetable, is readily assimilated by the soil and is easily soluble in

[1] London *Mining Journal.*
[2] Abstract from *l'Echo des Mines et de la Metallurgie,* May 2, 1906.

the phosphate or lactate of ammonia. It loses its water of crystallization upon being heated to a low red heat. The ground dehydrated powder is soluable in citrate of ammonia.

New Zealand.—Mining is being carried on by the Ewing Phosphate Company at Clarendon, Otago. There were 5,000 tons of phosphate rock produced for treatment at the chemical works at Burnside in 1905. A considerable quantity has also been mined by the Milburn Lime and Cement Company from its property at Milburn. Deposits have been discovered near the Waiau river, Southland.

Russia.[1]—There are important though little exploited deposits of phosphate rock in Russia. They are badly situated for economical exportation, and are not convenient for supplying the Russian manufactories of superphosphate, which are limited in number. One factory is located at Riga, four in Poland, one at Odessa, and another at St. Petersburg. The phosphate fields are found in widely-separated regions. The quality of the rock is extremely variable ranging from very rich to low grade. Geologically, the phosphates may be classified into three groups, as follows:

(1) The phosphates of Podolia which are found between Staraja, Uschtschiza and Mohileu, as well as in the valley of Dniester, district of Chotin, on the Galician frontier. Their content in phosphoric acid runs from 23 to 28 per cent. A small quantity of iron is present. These phosphates constitute excellent material for the manufacture of superphosphate, and are utilized by all the factories of southeastern Russia, a portion also being exported.

(2) The Wologda phosphates were discovered in 1896, on the banks of the rivers Lyssola and Wisinga, in the district of Ustj-Syssolsk. Their average content in phosphoric acid is from 26.8 to 37.6 per cent. This deposit is well situated in the proximity of both railroads and canals, and the towns of Perme and Wjatka should be excellent locations for installation of superphosphate plants. The pyrites of the Ural could be employed for manufacturing the sulphuric acid necessary.

(3) The phosphates of the center are found in the Government of Moscow, Vladimir, Kostroma, Jaroslaw, Nijni-Novgorod, and Smolensk. Their phosphoric acid varies from 13 to 30 per cent. The phosphates from the southeast of the Moscow basin are associated with fine grains of quartz sand and agglomerated glauconite. The glauconite is a silicate of potash and iron and aluminum hydrates, which Grigorjeu states contains an average of 7.90 per cent. of potash. This latter, being assimilable by plants, augments the value of the fertilizer.

There was discovered, in 1899, in Finland, on the borders of Lake Chaiain, near Ladoya, a large deposit of phosphate rock suitable for

[1] From *American Fertilizer*, February, 1907.

making superphosphate, but up to this time absolutely nothing has been done in the way of development.

Tunis.[1]—Phosphate mining stands first among the mineral industries of Tunis and the exports of crude rock of this country rank next to those of the United States. The shipments from the Gafsa fields at the Port of Sfax in 1906 were 593,276 tons. The figures for the mines of Kalaat Senaam and Kalaat Dgerda are not available, but are estimated at 200,000 tons. This would bring the total for Tunis close to 800,000 tons.

The phosphates of Ain Moulares, recently conceded to the Gafsa company, and which adjoin that company's concession of Metlaoui, will soon swell the exportations. The concession holders undertake to send 250,000 tons per annum over the line now in course of construction from Sousse (Susa) to Ain Moulares, a distance of 280 km. This stipulation forms a heavy burden on the company, as it could very well forward the material over the line to the nearer port of Sfax, which line, moreover, is its own private property. However, the Government, in thus acting, secures the revenue of a new line opening up a valuable tract of country which badly needs railway accommodation. Prices advanced from 5¼d. per unit of tribasic phosphate to 7¼d. during the last few months of 1906. The rock yields from 58 to 68 per cent. and averages 63 per cent. tribasic phosphate, which is equivalent to 28.8 per cent. P_2O_5.

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[1] London *Mining Journal*, Jan. 12, 1907.

PLATINUM.

The production of platinum in the United States in 1904 was 200 oz. troy; in 1905 it was 318 oz. troy. Complete statistics of the production in 1906 are not yet available, but apparently there was a further increase. This is due directly to the increased attention devoted to the black sands of the Pacific Coast as the result of the investigation of the U. S. Geological Survey. There is no doubt whatever that those sands contain considerable platinum, together with a good deal of gold, that can be economically recovered. Several individuals have already engaged in the business with considerable profit to themselves. At present, however, the great problem is to collect the sand. Once collected, the method of treatment has been satisfactorily determined. This subject has been discussed in many articles in the *Engineering and Mining Journal* during the latter half of 1906 and the first half of 1907.

STATISTICS OF PLATINUM IN THE UNITED STATES.

Year.	Production. (a)		Imports.			Consumption.
			Unmanufactured.		Manufactured	
	Troy Oz.	*Value.*	*Troy Oz.*	*Value.*	*Value.*	*Value.*
1896	163	$944	83,080	$926,678	$106,338	$1,033,960
1897	150	900	83,080	960,299	43,921	1,005,120
1898	225	3,375	101,018	1,178,142	52,283	1,233,800
1899	300	1,800	187,778	1,462,157	55,753	1,539,710
1900	400	2,500	118,919	1,728,777	36,714	1,767,991
1901	1,408	27,526	85,438	1,673,713	24,482	1,725,721
1902	94	1,814	105,450	1,950,362	37,618	1,989,794
1903	110	2,080	114,521	1,921,772	135,889	2,059,741
1904	200	4,160	103,802	1,812,242	105,636	1,920,478
1905	318	5,320	104,196	1,985,107	188,156	2,176,263
1906	(e) 400	8,800	137,556	3,601,021	187,639	3,797,460

(a) Statistics of the U. S. Geological Survey, except for 1906. (e) Estimated.

None of the new discoveries of platinum in the United States which we reported in Vol. XIV appears to have developed into anything of importance. As usual, there were many reports of new discoveries in 1906, including one near Bunkerville, Lincoln county, Nev., but all of these are likely to prove as erroneous or inconsequential as those reported in 1905 and previous years.

According to the *South African Mines, Commerce and Industries* of Feb. 23, 1907, a remarkable discovery of platinum has been made in the Albany district, on Spring Grove farm, about 15 to 20 miles from Grahamstown. A huge outcrop of quartz, half a mile long, is said to occur there, in which platinum is undoubtedly present. In the same journal

for Mar. 2, 1907, an assay is reported of this rock, which showed 1.6 oz. platinum, 1.2 oz. gold and 1.7 oz. of an alloy of platinum, iridium and osmium. A good deal of excitement has naturally been caused by the discovery, and development work is now said to be in progress.

It is also reported that copper sulphide ore containing 17 dwt. platinum per ton has been discovered in the Thomson River mine, at Walhalla, Victoria, Australia.

In 1905 the price of platinum at New York opened at $19.50 per oz. and continued at that figure during January and February, rising to $20.50 per oz. in March, at which figure it remained steady for the rest of the year. In 1906 the fluctuations in the price of platinum were spectacular. The monthly averages, together with the highest and lowest prices, are given in the accompanying table. The price for scrap platinum is quoted, because it is upon that basis that the refiners buy the crude platinum offered by the miners. The average for 1906 was $28.21 per oz. for refined and $22.00 per oz. for scrap.

PRICE OF PLATINUM PER OUNCE AT NEW YORK IN 1906.

Month.	Refined.			Scrap.		
	High.	Low.	Average.	High.	Low.	Average.
January	20.50	20.50	20.50	....	....	16.00
February	25.00	25.00	25.00	....	....	19.00
March	25.00	25.00	25.00	....	....	19.00
April	25.00	25.00	25.00	....	....	19.00
May	25.00	25.00	25.00	....	....	19.00
June	26.00	25.00	25.40	....	....	19.75
July	26.00	26.00	26.00	21.50	21.50	21.50
August	26.00	26.00	26.00	21.50	21.50	21.50
September	33.00	28.50	32.10	24.00	24.00	24.00
October	33.00	33.00	33.00	26.00	24.00	25.50
November	33.00	38.00	35.50	30.75	26.00	28.38
December	38.00	38.00	38.00	31.75	30.75	31.25

A great deal was written in the commercial and technical press during 1906 on the subject of the platinum industry and the remarkably high price recorded for the metal. The simple explanation of the matter appears to be that the production of the metal has not materially increased, while the demand for consumption, through the great expansion in the incandescent gas and electric lighting industries, together with the manufacture of dental supplies, and the miscellaneous requirements for chemical and jewellers' purposes, has greatly increased. Advantage has been taken of this situation by speculators in Europe who reap a great profit. Evidently the producers in Russia have not benefited to any considerable extent, their output having been contracted far ahead at lower prices. The disturbances caused by the Russo-Japanese war, followed by the internal troubles in Russia, and finally by various rumors, such for example that the Russian Government intended to impose an export duty on platinum

exported from Russia, have been advanced as pretexts to account for the extraordinary rise in the price asked for the metal.

PLATINUM PRODUCTION OF RUSSIA. (a)

District.	1904.		1905.	
	Kilograms.	Ounces.	Kilograms.	Ounces.
Tcherdinsk	153.6	4,938	125.4	4,032
Perm	1,107.1	35,593	1,221.0	39,255
South Verkhotoorsk	3,538.5	113,763	3,536.9	113,711
North Verkhotoorsk	207.3	6,666	311.6	10,018
South Ekaterinburg	5.6	179	46.4	1,492
Total	5,012.1	161,139	5,241.3	168,508

(a) Privately communicated by W. A. Abegg, Warsaw. The total production in 1903 was 6003 kg. (193,000 oz.) and in 1902 it was 6133 kg. (197,173 oz.)

PRICE AND PRODUCTION OF PLATINUM IN RUSSIA PREVIOUS TO 1899. (a)

Year.	Price. Rubles.	Product Poods.	Year.	Price. Rubles.	Product Poods	Year.	Price. Rubles.	Product Poods	Year.	Price. Rubles.	Product Poods.
1869	1,600	140–0	1890	12,000	270–0	1893	....	311–3	1896	....	301–0
1874	....	120–0	1891	5,000	258–6	1894	....	318–0	1897	....	342–0
1882	....	250–0	1892	7,000	279–2	1895	....	369–5	1898	13,000	367–0

(a) From *The Mineral Industry*, Vol. XIV.

According to an article in *Le Genie Civil*, Apr. 6, 1907, the price for platinum at Paris in January, 1906, was 3400 francs per kilogram, and recently has been 6000 francs. The great rise and wide fluctuations are attributed mainly to speculation to which platinum especially lends itself, so large a proportion of the production—90 to 95 per cent. of the total—being obtained from mines in the Urals, which are in the hands of a few individuals or companies. Among other causes explaining the exceptionally high price, the disorders in Russia rank first. Their effect has been to make manual labor scarce, and transport risky. (Nevertheless, the statistics do not show any diminution in the production of Russia in 1906; to the contrary.) The annual production of platinum in the world is now about 5000 kg, which is remarkably below consumption, the averages of which are 3300 kg. for the United States, 3050 kg. for England, 2130 kg. for Germany and 2000 kg. for France, the total being 10,480 kg. On the other hand, one company supplies more than half the output of the Urals, and is the only one organized on satisfactory industrial lines. However, the large difference between the statistics of platinum production and consumption are explained by the fact that an important quantity of old platinum comes again into the market for further use. This quantity is estimated at 30 to 35 per cent. of the consumption. Moreover, as in the case of gold production in Russia, a large amount of platinum is stolen at the mines, and consequently does not appear in the statistics of production. The thefts are estimated at 25 per

cent. of the actual production. It is impossible to fix accurately the amount of the thefts or the quantity of old platinum which comes again into the market, but beyond doubt both of these factors have increased greatly with the remarkable increase in the price. However, assuming the figures stated above, the actual production would still show a deficit of nearly 2000 kg. per annum as compared with the consumption.

In a recent interview published in the *Torgovo Promishlennaia Gazeta*, S. I. Gulishambarov, an official of the Russian Ministry of Commerce and Industry, expressed the opinion that the rise in price of platinum was brought about by the fact that, while the demand is increasing, the supply has lately diminished. The Compagnie Industrielle du Platine, of Paris, which is the largest producer of crude platinum on the Urals, and is also owner of large platinum-refining works in Paris, forms also the connecting link between the remaining independent platinum producers and the combination of platinum buyers headed by Johnson, Matthey & Co., of London. This French company is making every possible effort to keep the price at the highest level possible. Last year it offered the following terms for purchase of platinum: On delivery of the platinum the company pays as an advance 12,000 rubles per pood of crude platinum containing 83 per cent. of pure metal. After the platinum has been refined and sold, the whole of the difference between the amount advanced and the sum realized is handed over to the seller. The company's profit is represented by interest at the rate of 5 per cent. per annum on the money advanced, and ¼ per cent. commission for every three months. Moreover, the valuable metals of the platinum group which occur in the crude platinum become the property of the company as compensation for the cost of refining, etc.

The combination of platinum users pays to the company 3 fr. per gram of refined platinum, the company also receiving 50 per cent. of the profits derived from the sale of the articles made out of the platinum. This price of 3 fr. per gram fluctuates in accordance with the conditions of demand and supply. Last year, for instance, when the largest platinum producers reduced their output, it was found possible to pay 3.30 fr. per gram, with the same 50 per cent. participation in profits. Since then the French company further reduced the output, and in consequence of this the price rose further. It is therefore clear that the present high price of platinum can be quite easily explained by the natural conditions of supply and demand, without reference to any rumors about intended Government action.

E. De Haupick, of the Imperial Russian Engineers, in a paper in the *Mining Journal*, (London) May 4, 1907, gives the following valuable data as to the platinum industry in Russia:

The principal platinum placers are Goroblagodatsk, on the Asiatic

slope of the Urals, north of the basin of the River Iss (placers of Count Schouvalov) and Nigni-Tagilsk, on the European slope, the basin of the rivers Vissim and Martian (placers of Prince Demidof). Placers of the second rank are scattered about Verk-Issetsk, Chermo-Istchinsk, Ekaterinbourg, Kytim, Solva, Slatoust, and on some affluents of the River Toura.

The Urals have produced from the first days of the platinum industry until 1907, six million ounces troy of crude platinum, and to these official figures we must add about 1.5 million ounces of platinum stolen by the miners, which has secretly found its way to the market, and is not accounted for by the Government statistics. The production at the Nigri-Tagilsk district is twice as great as that of the Goroblagodatsk and the quality of the gravel is better than that of the latter.

The development of the Ural platinum industry has been very uneven. From 1828 to 1846 the platinum output was pushed from 6500 oz. to 110,000 oz. per annum, but after the Russian Mint suppressed the coining of platinum the output fell in 1852 to 6400 oz. It was revived again by the English in 1862, when they began to export Ural crude platinum to London. The total output of the last few years is given in the accompanying table.

PLATINUM PRODUCTION OF RUSSIA.

Year.	Ounces.	Year.	Ounces.	Year.	Ounces.	Year.	Ounces.
1894	203,250	1898	203,100	1903	226,000	1905	200,450
1896	200,000	1900	212,500	1904	190,120	1906	210,318

In 1906, 120 platinum placers were exploited, employing 6200 workmen, who washed 1,886,000 tons of alluvium, with an average of 0.09 oz. crude platinum per ton of gravel. In olden times the placers were richer in metal: from 1825 to 1829 the average extraction was 2.7 oz.; in 1829-1838, 0.5; in 1838-1850, 0.4; in 1850-1883, 0.33; in 1883-1894, 0.1; and in 1894-1907, 0.09 oz.

AVERAGE PRICE PER OUNCE TROY FOR PLATINUM INGOT.

Year.	£	s.	d.	Year.	£	s.	d.	Year.	£	s.	d.
1874	1	5	2	1900	3	19	9	1906 (Jan.).	4	15	2
1888	1	13	8	1901	4	1	11	1906 (Oct.).	7	19	8
1890	1	15	8	1902	4	4	0	1907 (Jan.).	7	0	0
1893	1	17	9	1903	4	6	1	1907 (Feb.).	6	19	6
1895	2	2	0	1904	4	8	1	1907 (Mch.).	6	19	0
1898	3	13	6	1905	4	10	4	1907 (Apr.).	6	18	8

The Ural platinum market since 1862 has been in foreign hands. Johnson, Matthey & Co., of London, control the Nigni-Tagilsk placers; another English company (the Anglo-Russian Platinum Company, Limited) exploits the Cheridinski placers, the property of the Schaidourof Brothers;

a German firm (Heraeus & Co.), of Hanau, and a French one (Desmoutis, Lemaire & Co.), of Paris, also control a great many placers. But the most important of the Ural firms is the Compagnie Industrielle du Platine, a French company, which has 40 different platinum placers on the rivers Iss, Peschanka, Toura, Schumakka, Gussevo, Melnitchoi, Taliza, Viy, Kissloi, Mokroy, and others. This company also has bought until 1915 the whole output of numerous placers of Count Schouvaloy, in the Goroblagodatsk district.

PRICE FOR CRUDE PLATINUM OF 82 PER CENT. IN THE URALS.

Year.	Rubles. Per Pood.	£ s. d. Per Oz.	Year.	Rubles. Per Pood.	£ s. d. Per Oz.	Year.	Rubles. Per Pood.	£ s. d. Per Oz.
1874	4,800	0 19 0	1899	7,000	1 7 8	1906 (Jan.).	22,000	4 7 9
1888	6,000	1 4 0	1901	16,200	3 3 9	1906 (Oct.).	34,000	6 15 8
1890	6,200	1 4 8	1902	17,300	3 9 4	1907 (Jan.).	30,000	5 18 11
1893	6,500	1 5 10	1903	18,500	3 13 3	1907 (Feb.).	29,000	5 15 8
1895	6,600	1 6 7	1904	21,000	4 3 8	1907 (Mch.).	28,000	5 12 0
1898	6,800	1 7 0	1905	22,000	4 7 9	1907 (Apr.).	27,000	5 8 0

The Compagnie Industrielle du Platine has a contract with another French company—the Compagnie des Metaux—to which it transfers the whole output of platinum for its disposal on European and American markets. The yearly production of this company amounts to 70,000-90,000 oz. Russia does not consume more than 6000 to 7000 oz. and this amount is used by two firms in St. Petersburg—Kolbe & Lindfors and the Tentelef Chemical Manufacturing Company. Germans import Ural platinum only for their own consumption, without taking part in speculation. About 80 per cent. of the whole platinum output goes from the Urals to France and England.

The whole import of platinum into the United States is made from Paris, where the center of gravity of the world's platinum market is at present. As the whole yearly Ural platinum output is about 200,000 oz., the United States is the greatest consumer of Russian platinum.

The large profits from the prevailing high prices for platinum do not benefit the Ural mine owners, because the entire output for a varying term of years has been bought up under contracts at very low prices—about £2 6s. per oz. for 82 per cent. crude platinum—and they are excluded from participation in any gain in the rise of its value. Consequently the mine owners are not anxious to increase the platinum supply, awaiting the time when they can dispose of their product to better advantage.

We can thus explain the fact that, in spite of the unprecedented rise in price, the output of platinum during the last 10 years has not increased. If we now study the figures relating to platinum ingot prices, we observe that previous to 1898 it was £2 2s. per oz. and since that time it has in-

creased to £3 13s. 6d. in 1898, £4 4s. in 1902, £4 10s. 4d. in 1905, and £7 9s. 8d. in 1906. We can account for this when we bear in mind that the Compagnie Industrielle du Platine began its activity in 1898 with 65,-500 shares, representing a capital of £1,000,000, which pays a high dividend, as well as the shares of the Compagnie des Metaux, which sells the platinum of the Compagnie Industrielle du Platine.

The handling of over 65 per cent. of the Ural platinum output is controlled by a coterie—the so-called platinum trust—which can manipulate the market almost at will. Yet, so far as can be ascertained by diligent inquiry, the recent break (in the early part of 1907) originated in the United States, which accumulated during 1906 a considerable stock of platinum. At present American consumers are convinced that the prices will decline, and are placing as small orders as possible. To the impartial critic of the situation it would seem that the market actually is weakening, and we can expect for some time a decline in the price of platinum, but only for a time. It may be confidently predicted that the causes above mentioned, viz, the increase of the platinum price, the limited number of placers, the diminution in their yield, and the speculative tendency of the French company, do not give us a hope to expect in future a decrease of the price of the metal to a reasonable figure, as its demand for almost all purposes increases every day.

POTASSIUM SALTS.

BY REGINALD MEEKS.

No new deposits of potash salts were discovered in 1906 and Germany continues to be the world's source of supply, except for the nitrate. The German mines, and the chemical works connected with them, employ about 10,000 miners, 15,000 laborers and 800 chemists and technical experts, besides a large clerical force.

PRODUCTION OF POTASSIUM SALTS IN GERMANY. (a)
(In metric tons and dollars; 1 mark=$0.238.)

Year.	Kainite.		Potassium. Chloride		Potassium. Sulphate.		Potassium Magnesium Sulphate.		Other Salts. of Potassium.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
		$		$		$		$		$
1896	856,290	2,989,736	174,515	5,718,559	19,682	813,381	4,623	85,977	902,707	2,964,750
1897	992,389	3,486,007	168,001	5,764,423	13,774	565,720	7,812	149,079	953,798	3,030,143
1898	1,103,643	3,835,856	191,347	6,380,220	18,853	763,397	13,982	259,485	1,105,212	3,576,628
1899	1,108,159	3,838,250	207,506	6,801,250	26,103	1,027,500	9,765	195,000	1,384,972	4,202,000
1900	1,178,527	4,134,000	271,512	8,793,750	33,853	1,249,250	15,368	280,500	1,874,346	5,643,750
1901	1,500,748	4,327,250	282,750	8,782,250	27,304	1,460,000	15,612	286,500	2,036,326	5,443,250
1902	1,322,633	4,571,980	267,512	7,507,710	28,279	1,079,092	18,147	334,390	1,962,384	4,949,448
1903	1,557,243	5,208,154	280,248	8,125,320	36,674	1,389,444	23,631	441,252	2,073,720	4,993,478
1904	1,905,893	6,322,470	297,238	8,425,676	43,959	1,664,572	29,285	545,972	2,179,471	5,305,972
1905	2,387,643	7,976,808	373,177	10,580,528	47,994	1,804,040	34,222	614,754	2,655,845	6,396,250
1906	2,679,264	8,918,574	403,387	11,034,632	54,490	2,032,520	34,239	644,028	2,803,732	6,538,336

(a) *From Vierteljahrshefte zur Statistik des Deutschen Reichs.*

EXPORTS OF SALTPETER FROM INDIA. (a)
(In tons of 2000 lb.)

Year.	Quantity.	Value.	Value per 100 lb.	Year.	Quantity.	Value.	Value per 100 lb.
1896-97........	29,583	$1,430,410	$2.42	1901-02.......	17,721	$1,189,400	$3.22
1897-98........	20,889	1,329,155	3.06	1902-03.......	20,531	1,442,435	3.37
1898-99........	18,263	1,164,480	3.06	1903-04.......	19,549	(b)1,470,000	(b)3.75
1899-00........	19,870	1,281,050	3.64	(c)1904-05.......	17,437	1,175,567	3.37
1900-01........	17,432	1,471,245	4.05	(c)1905-06.......	16,822	1,249,887	3.71

(a) From "Mineral Production of India," by T. H. Holland, Government Geologist. (b) Value estimated. (c) From Statistical Abstracts for the British Colonies, Possessions and Protectorates.

The "Kalisyndikat," which controls the price of potash fertilizer salts, decided at the meeting Nov. 23, 1906, to advance the export prices on raw salts to the level ruling in July, 1904, prior to the renewal of the combination, all European countries, Great Britain, France, Holland and Belgium, particularly, being affected. The question of similarly increasing the export price to the United States was held in abeyance and later,

Feb. 11, 1907, it was decided that this step would not be taken in 1907.

There are now 36 companies included in the syndicate and 30 more are operating independently. It is reported that 200 prospecting companies were organized in 1906 and that this is a falling off in number from those organized in 1905.

The Refined Salt Market During 1906.[1]

Caustic Potash.—There was a moderate increase in the business in 1906 over that of 1905 and prices advanced toward the close of the year. Old process caustic potash, which brought 4½ @ 5c. in the beginning advanced to 4½ @ 5½c. in the last quarter of the year. New process salts opened at 5 @ 7c. but the price was shaded to 5 @ 6¼c. in July and later the minimum price advanced to 5½c.

Carbonate.—The market was dull for more than half the year but in the autumn an active demand developed a heavy movement on contracts. The demand was considerably in advance of the supply toward the close of the year. Prices opened at 4@4½c. for hydrated, 80 to 83 per cent., dropped to 3⅞c. and rose later to 4½c. Quotations for calcined, 80 to 85 per cent., were 3¾ @ 4¼c.; for calcined, 96 to 98 per cent., 4⅝ @ 5c.; these prices held for the year except for a slight shading off in July.

Chlorate.—Early in the year supplies were scarce and jobbing lots brought fancy prices. Contract prices, however, remained between 8½ @ 9½c. for crystals and 9c. for powdered, and these prices were firmly maintained up to the close of the year.

Bitartrate.—Cream of tartar prices for the year were steady except in January and the last two months. The year opened at 23¾c., but this price soon fell to 22¾c., where it remained until November, when a reduction of ¼c. was made.

Nitrate.—The market as a whole held remarkably firm, prices fluctuating between 4 and 4½c. for crude salts, while refined was quoted at 4½ @ 5½c. until September, when the price advanced ¼c., due to depleted supplies.

Bromides.[2]—No settlement between American and German manufacturers resulted during the year and low prices prevailed. Local dealers endeavored to buy the salt in order to fill orders from abroad, but the agents of the foreign makers refused to sell except to consumers and stipulated that these purchases were not to be exported. Prices were stationary for the year at about 16@17c. for potassium, 21@22c. for sodium and 23@24c. for ammonium bromide.

Prussiate.—A steady advance in price occurred in 1906. The year opened with prices at 14c.; they touched 15c. in June, 16c. in September, 17c. in October and remained at that price until the close of the year.

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[1]From the *Oil, Paint and Drug Reporter*, Feb. 11, 1907.
[2]Refer also to the article on "Bromine" in an earlier part of this volume.

The output was under close control and the salt was scarce. In some cases consumers were obliged to come into the market and purchase in excess of their regular contract deliveries. The average price for the year was 15.33c.

The Potash Mines at Stassfurt, Germany.

These mines are situated near the Harz mountains. Some are in the province of Hanover, some in Brunswick and some near the Thuringian forest. At first the deposits of rock salt only were utilized, the value of the other potash salts not being known. In 1857 a large bed of rock salt was penetrated at a depth of 1080 ft. and overlying this bed was a layer of potassium and magnesium compounds. Four years later a process was perfected for extracting potassium chloride.

According to George Ryce[1] the lowest stratum consists of the least soluble material, anhydrite ($CaSO_4$). The next above is rock salt interspersed with lamellar deposits of anhydrite which are gradually replaced by polyhalite, or sulphates of lime, potash and magnesia. Above this region, known as the polyhalite region, comes the kieserite region in which, imbedded between rock salt, occurs kieserite, or magnesium sulphate. Above this occurs the carnallite region, consisting of the chlorides of potash and magnesia. This stratum is 50 to 150 ft. thick and yields the most important of the crude potash salts from which is obtained the bulk of the refined product. Finally, strata of impervious salt-clay, anhydrite and a remarkably pure deposit of rock salt are found, in the order named, and above all there are strata of gypsum, clay, sandstone and limestone which outcrop at the surface.

The deposits cover an area of about 100 square miles and in depth extend to probably 5000 ft. Circular shafts, 18 to 20 ft. in diameter, are sunk to levels 1200 to 2500 ft. below the surface, and are lined either with concrete or iron tubing. Extreme caution must be exercised to prevent any inflow of water and it is customary to have a reserve shaft at nearly all the mines. Another danger is from falling roofs, and all worked out portions of the mines are filled with waste and rock salt.

Of the various crude salts mined, by far the most important is carnallite. It occurs clear or transparent, or any shade of yellow, red, gray or black. These colors are due to impurities such as oxide of iron, clay, organic matter, etc. On account of its hygroscopic nature and its small percentage of potash, carnallite is used as a fertilizer only in localities which are not too far from the mines. Next to carnallite in importance is kainite, the sulphates of potassium, magnesium and chloride of magnesium. It is too closely intermixed with rock salt to attempt separation, and consequently it is put on the market containing about 30 to 40 per

[1] London *Min. Journ.*, Jan. 19, 1907.

cent. of rock salt. The last mineral of general importance is sylvite. It contains about 22 to 30 per cent. KCl, 50 to 65 per cent. NaCl, and 4 to 12 per cent. K_2SO_4. It is richer in potash than kainite.

The primary material for all manufacturers is carnallite, having an average of 9 per cent. pure potash. The roughly-ground carnallite, KCl $MgCl_2$ $6H_2O$, comes first into large dissolving vats, wherein it is treated with chloride of magnesia liquor, steam being passed through to aid dissolving. The chloride of potash, which is more soluble than sodium chloride in a saturated solution of magnesium chloride, is dissolved out, and a turbidity results. When the turbidity, which is due to NaCl and $MgSO_4$, has settled, the solution is run into large iron crystallizing tanks, and allowed to cool for three or four days. It still contains from 25 to 40 per cent. NaCl. By evaporation and subsequent cooling of the mother liquor an artificial carnallite is obtained, which, after being again dissolved in hot water, forms crystals of pure chloride of potash.

The chloride of potash from the crystallizations is ultimately washed with cold water, in order to remove the chloride of sodium and chloride of magnesium. After drying, the final product is a chloride of potash, 70 to 99 per cent. pure, according to the process adopted. The salts are then dried in ordinary furnaces or in calcining furnaces.

From kainite, by a somewhat complicated process, sulphate of potash and magnesia is obtained, in two forms, viz, crystallized with 40 per cent. and calcined with 48 per cent. sulphate of potash. The latter is used for fertilizing; the greater part of the former is utilized for the manufacture of sulphate of potash 90 or 96 per cent. pure. After producing the chloride of potash, there remains a mother liquor, consisting mainly of chloride of magnesium and 2 per cent. of bromine. From this solution the $MgCl_2$ is recovered by simple evaporation and the bromine either by treatment with chloride or by the electrolytic method.

About two-thirds of the total production of the chloride, one-eighth of the sulphate and the whole of the crystallized sulphate of potassium and magnesium are used for the manufacture of the various salts and compounds of potassium.

Potassium Salts in the United States.

By William M. Courtis.

There has not yet been discovered in the United States a deposit of potassium salts similar to those found in the Stassfurt district of Germany. Very little exploration has been done in this country, for the reason that there are few noticeable surface indications which could be developed commercially and, owing to the solubility of the salts, the deposits do not come to the surface. Boring is expensive and returns are uncertain. Moreover, the indications of supply in this country are at

a great distance from the market. Finally, the cost of production would have to be low enough to compete with the Kali syndicate, and German potash can be produced at a very low figure.

There are several large and well-known supplies of surface potash in this country, which could be utilized in the event of the imports of the United States being entirely cut off, but at present they are remote from railroads, in the desert portion of this country, and the cost of working would be prohibitive. These deposits are largely mixed with soda salts and would require washing, and this would be costly. The salts consist of sulphates and nitrates only, the chloride not having been found in any quantity. It is probable that with so many surface indications, occurring where conditions are similar to those of the Stassfurt district, boring on a large scale would supply information which would eventually lead to the discovery of a bed similar to that of Germany, but it would require a large expenditure for blind boring. However, in Germany the surface indications were so slight that the immense deposits were not discovered until a comparatively few years ago, and then only by accident.

Favorable Localities.—Saline water has been discovered in Massachusetts near Mount Tom, but there is no record that this water contains potash; in New Jersey, near Passaic, a deep well was sufficiently saline to suggest the possibility of the proximity of a bed of potassium salts, but nothing has been done to prove this. None of the wells of the eastern coast has been sunk to a sufficient depth to throw any light on this point, as they were abandoned when salt water was struck, it being useless for the purpose of the manufacturing concerns doing the drilling.

There are favorable places in Massachusetts, New Jersey, Ohio, Missouri, Wyoming, Nevada, California and Lower California, and in all of these sections the geological formation is suitable and the underground waters suggest some such deposit. There are numerous small basins, such as the Big Horn (Wyoming), the Magnesian and the Soda Lake basins near Laramie, Wyo., Death Valley and the sinks of Humboldt and Carson, Nevada, which possess favorable indications that would warrant drilling. The waters of these districts contain a large amount of potash, but this may be due to the action of the sulphur springs on the potash-bearing feldspars of the granite underlying a large portion of the country.

Classes of Deposits.—There are three classes of deposits in the arid districts of the West: (1) Evaporated lakes, called "summer salts" (which usually contain only a small percentage of potash); (2) deposits resulting from volcanic action; (3) deposits of organic origin (such as in bat caves).

As examples of the first class may be taken deposits in the Carson and Humboldt sinks, Churchill county, Nevada. The potash in these will average 3 to 12 per cent., although higher in occasional streaks. In

the second class belongs the Niter hill deposits, in the same locality. Niter hill seems to be a volcanic cone, the solfataric action of which has produced large deposits of sulphate of soda, sulphate of magnesia, and mixed nitrates. In the crevices of the rocks near the surface the potash has crystallized out to 98 per cent. pure potassium nitrate, and the whole bed will probably average 5 per cent.; it has never been tested below a depth of 15 ft. The mud volcanoes of Lower California also produce some potassium sulphate.

Deposits of the third class are found in Utah, one near the town of Turner, and others in the Big Horn mountains near Dayton, Wyo. Here apparently was a very large deposit of exceedingly pure niter, running about 87.5 per cent. potassium nitrate. On testing it developed that this deposit was merely porous sandstone, coated with from 1 to 3 in. of solid niter. In these deposits the aggregate of potash would be very large but would necessitate working an immense amount of waste rock. Recent reports from Idaho state that deposits of material have been found there that will run 25 per cent.

Interesting deposits which cannot, however, be included in any of the three classes mentioned, are those of Death Valley. They cover an extensive territory in the valley and now that mining is being developed and railroad facilities are possible, some attention may be paid to them.

Deposit indications are not as good on the eastern coast as in some of the basins above mentioned. The cost, however, would not be so great and possibly $25,000 for boring on the Atlantic coast might lead to profitable results.

Cost of Development.—To test thoroughly for potassium salts in the United States it would require an outlay of probably not less than $175,000. The interior basin, like the Big Horn, would require probably $50,000; Death Valley, Niter hill, Nevada, the San Diego desert and the Cuchapas mountains of Lower California (in the latter two the indications are particularly good) would require considerable more expenditure. Boring might prove that deposits of the sulphates of potash, soda and alumina, and in some places large deposits of carbonate of soda and borax minerals, are in sufficiently large quantity to be commercially available.

It would be necessary to test all these places on the upturned edges of the bed, so that the Permian underlying rocks would be cut at a depth not to exceed from 1200 to 1500 ft. as the salts could not be worked so profitably at greater depth. The modern method of pumping the brine is not feasible and the salts must be mined and used in the raw state. The German salts are now being worked at a depth of 1600 ft. and boring has proved them to exist to a much greater depth.

PRECIOUS STONES.[1]

By George Frederick Kunz.

As yet without result, at three distinct places in the United States prospecting for diamonds was carried on in 1906; in Wisconsin, in what apparently is glacial drift; in Kentucky, in the peridotite of Elliott county; and in California at Oroville. In Montana sapphires were extensively mined by three companies, very actively by one, partially so by another, while some development was done by another in Fergus and Granite counties Sapphires have been found also in Idaho, and a number of stones, but none of fine color, have been locally cut as souvenirs. In North Carolina ruby was mined for at the Cowee Valley locality, and corundum was found in place, the rock being a decomposed pegmatite, although it differs somewhat from the redder ruby found in the regular alluvial soil. No emeralds were obtained during 1906.

Beryl and aquamarine were only slightly mined for in North Carolina, nearly all the material sold being from older workings, and the small development of a few local prospectors. A few fine beryls were found near New Milford, Conn., some of them yielding exceptionally fine material that cut into gems over 25 karats each. Of unusual interest were the large beautiful crystals of rose beryl found in San Diego county, Cal.; some of these crystals were 3 or 4 in. across, and one magnificent mass of pegmatite, on which were implanted several large crystals, has found its way into the Morgan Collection of the American Museum of Natural History at New York. A few of them have been cut into rose-colored gems, which are interesting, but not of great value.

Never have so many and so varied tourmalines been found as in 1906, at the several mines in the vicinity of Mesa Grande, Cal., as well as at the localities further south, near Pala. The former locality yielded some crystals weighing 4 to 6 lb. each, that gave little suggestion of being of gem value, but they were magnificent as mineralogical specimens, a number of them being doubly terminated. Fortunately the finest of these have gone to two of our best museums. Transparent portions in the interior of some of these crystals have furnished several

thousand karats of rich red gems. Similar material was also found at other localities in the State; and probably never have more tourmalines been produced and disposed of than during 1906. Some beautiful interesting green crystals have been found, as well as a great variety of material that has been cut into cabochon stones, irregular beads, and other East Indian-like objects.

A new locality has been discovered for colored tourmaline, near East Haddam, Conn, and a number of interesting crystals and a few gems have been obtained. In Maine, also, near Paris, Auburn, Mount Black, and several other localities, tourmaline has been discovered.

Topaz, in some cases beautiful wine-colored crystals, has been obtained near Topaz, Utah; and also near Ramona, Cal. At the latter locality a single crystal weighing more than 1 lb. was remarkable as a specimen; and some small gems were obtained from the blue, green and white material found there.

Kunzite was worked at three or four mines in California, and some very interesting crystals were obtained, although the amount taken out may probably not exceed $10,000 for the year. Of special interest is the finding of this mineral near Andover, Maine, although transparent only in spots. A remarkable fact is that not only is the lilac-colored variety obtained here, but the mineral is also found of a bluish-green color, like that of dilute sulphate of copper. Upon exposure to the heat I found that this variety lost its green tint and became pink. This same bluish aspect has been noted as a dichroitic property of the California material.

Of the 10 or more companies interested in mining turquoise, only three did much work during 1906, and the output did not exceed more than one-fourth that of 1905. An interesting feature was the finding of turquoise in one of the mines in even layers of veins in the rock. This onyx-like structure has been taken advantage of by the lapidaries, and some beautiful cameos have been made, one of which is nearly 3 in. across, showing a blue turquoise vein on a fawn-colored rock. The contrast of a sky-blue on a brown ground is so striking as almost to raise a doubt of their genuineness, of which, however, there is none whatever.

Fully 10 tons of rose quartz from the Black Hills, selling for about 10c. per lb., were cut up into beads, spheres, pear-shaped ornaments, and the like, for pendants and other uses. Moss agate from two localities in Utah also was worked. Amazon stone from Virginia and several other localities was cut into beads in great profusion during 1906, the green gems of all kinds being in favor. Great quantities of chlorastrolite from Isle Royale were obtained, varying in size from a small pea to more than 1 in. across. This beautiful green stellate material finds a ready sale, especially in the cities bordering on Lake Superior.

The year 1906 has never been equalled by any previous year, in the United States, in the importation of diamonds, pearls, rubies, emeralds and sapphires. The last is the only gem which has not been honored with special favor previously, but in 1906 this least of the precious stones has been appreciating in value as had the others already. A great demand existed not only for the regular precious stones, but also for aquamarine, topaz, peridot, and all varieties of the semi-precious stones, and especially for the rich green jadeite from near Bahomo in Burma. It is called Chinese jade by the celestials; in fact, they almost believe that it is of Chinese origin, because nearly all of the material passes through China and is generally there worked into all manner of ornaments. When of almost emerald green color this mineral sometimes brings very high prices, as much as $10 to $20 per karat being paid for exceptionally choice bits, and a long string of beads has been sold for more than one thousand dollars. Much of this material cut in China is now mounted in fine gold in the United States by Chinese workmen in absolutely correct Chinese designs. Great quantities of the New Zealand jade have been imported and cut into every variety of ornament. The demand for green stones also brought into general use the green aventurine, a rich pale or leek-green variety of quartz, scattered through which are particles of brilliant mica. This Indian material is unusually beautiful, and, like the New Zealand jade, is not expensive.

Malachite and a mixture of malachite and azurite from Arizona have been cut into a variety of forms, the latter being a pleasing mixture of spots of blue and green. Large quantities of artificially nickel-stained chalcedony have been sold under the name of chrysoprase. This is of a bluish green color and not of the rich golden green of the true chrysoprase which was mined to a considerable extent in California in 1906.

With the high price of emeralds in recent years, it has surprised many that stones of wonderful brilliancy and purity of color, with an almost entire absence of flaws, finer than they had ever seen before, were offered for sale. These emeralds were brought to the gem marts by the high price, the finest stones, belonging to many of the older families of Europe, being offered. Frequently a stone nearly 1 in. in diameter was sacrificed to be cut smaller for a pure spot that did not represent one-fifth or one-tenth of the original weight of the material, but from the fact that it was so much purer it was still advantageous to sacrifice the weight for the greater beauty.

During 1906 the price of diamonds was advanced by the syndicate 7 per cent. on the rough material, and three successive strikes led to advances being given to the diamond cutters equivalent to 7 per cent. more, but notwithstanding this, the output of cut material did not exceed the demand.

Precious Stones in Foreign Countries.

By Reginald Meeks.

Aquamarine.

Brazil.—Crystals of aquamarine having exceptional luster are found at Alegre, Bahia, Brazil. These deposits have superseded those of Porteiras and Boquerias, which were formerly centers of production. Two large stones have been found in Brazil. In 1814 an aquamarine was sold which weighed 15 lb., and in 1904 another, of equal weight was sold for $4368.

Diamonds.

Australia.—Diamonds have been won at Vingara and in the neighborhood of Cope's creek, Inverell district, N.S.W. Until recently only small stones were obtained, the largest of which there is any authentic record weighing only 6.25 karats. The discovery is reported of a fine diamond weighing $28\frac{5}{16}$ karats, at Mount Werong, in March, 1905.[1]

Brazil.—Up to 35 years ago Brazil furnished 90 per cent. of the diamonds of the world, but since the discoveries in South Africa the output has dwindled, until 10 years ago there were exported only about 3000 karats. It is reported that in 1905 the industry had begun to recover and 100,000 karats were exported. The estimated production of Brazilian diamonds up to date is over 12,000,000 karats, distributed as follows: From Diamantina, 5,000,500 karats; other mines in Minas Geraes, 15,000,000; Bahia (La Chapada), 2,500,000; stealings, etc., estimated, 2,500,000 karats.

Diamond deposits were recently discovered at Douradinho, district of Coromandel, Estrella do Sul, in the bed of the river. The diamonds are of fine quality and perfectly limpid. Good prices have been realized. The famous Estrella do Sul was found in this locality two centuries ago, which gave its name to the place. A French company recently explored the Boa-Vista mines, near Diamantina, and at present an English company operates the Agua-Suja mines, Bagagem, but operations are still in the installation stage. Lately several diamond-dredging claims of Brazil, on the river Jequitinhonha, the diamontiferous river, have been examined by American companies. On the river Coxipo, in Matto Grosso, many diamonds have been gathered with the gold, and with appropriate plant diamond dredging may offer very satisfactory results when competently directed.

Carbonado.

Carbonado, or black diamond, is obtained in the province of Bahia, Brazil, in La Chapada and Lavras districts, where it is mined from stream beds and other alluvion. The miners sell their finds to agents of exporting

[1]From report by E. F. Pitman, Department of Mines. N. S. W.

firms in the city of Bahia. Previous to 1870, carbonado was practically valueless. From 1870 to 1872 it was employed as an abrasive for cutting and polishing the white gems, and thousands of karats were sold at 50c. per karat, to be crushed into powder for this purpose. A few years later, when carbonado was employed in diamond drilling, it sold at $2 to $4 per karat, and then rose gradually to $10 per karat, at which price it remained until 1895, after which it advanced by leaps and bounds to $50 per karat. The price fell back to $25 per karat, but rose again to $85 per karat, at which figure it now stands. The situation is due to the decline in the supply of carbonado during the last ten years, while the demand has increased. The carbon settings for a drill-bit are expensive. Bits as large as 1 ft. in diameter, set with $5000 to $8000 worth of carbon, are frequently employed in Europe.

India.[1]—Diamonds are found at Panna, central India, in conglomerate resting directly on the upper surface of the Kaimur sandstone, at the base of the Rewa shales. Diamond mining in India is conducted in the crudest manner and is profitable only on account of the exceedingly cheap labor. By systematizing mining and introducing modern methods it is believed that the industry can be made profitable.

South Africa.—South Africa continues to be the world's main source of supply and the chief producers are, as heretofore, the De Beers Consolidated Mines Company and the Premier Diamond Mining Company. The combined output of these two companies was valued at over $33,500,000 in 1906. The 18th annual report of the De Beers Consolidated Mines, Ltd., deals with the company's operations during the year ending June 30, 1906. In that period, £5,607,718 worth of diamonds was produced, while the expenditure amounted to £3,504,182; the net profit being £2,103,536 or £154,437 more than in 1905. Dividends amounting to £1,800,000 were disbursed, and a sum of £916,057 was carried forward. On Jan. 30, 1906, the contract with the diamond trust expired, but was immediately renewed for a period of five years.

PRODUCTION OF DE BEERS MINES.

Mine.	Output of blue ground for year.	Yield per load.	Value per karat.	Value per load.	Cost per Load.		
					Mining. (*b*)	Washing.	Total.
	Loads. (*a*)	*Karat.*					
De Beers	2,253,988	0.410	$14.64	$5.95	$1.13	$0.58	$1.71
Kimberley	2,433,905				1.72	0.73	2.45
Wesselton	1,771,372	0.282	10.51	2.97	0.63	0.35	0.98
Bultfontein	1,771,372	0.363	10.31	3.74	0.79	0.52	1.31
Dutoitspan	1,685,714	0.245	1.943	4.77	0.99	0.72	1.71
Average	8,144,979	0.325	$13.72	$4.36	$1.05	$0.58	$1 63

(*a*) The "load" occupies 16 cu. ft. and weighs about 1600 lb. (*b*) Including cost of handling waste rock.

[1]From report of E. Vredenberg, Geological Survey of India, Vol. XXXIII.

The blue grounds and lumps on the floors at the close of the year amounted to 6,769,126 loads, an increase of 2,519,387 loads over the stock similarly accounted for at the close of the previous fiscal year. The total quantity of blue rock crushed and washed during the year was 5,625,592 loads, an increase of 497,577 loads. Diamonds to the value of $27,287,155 were obtained from the year's washing.

The total area of the De Beers Consolidated Mines, Ltd., is 4692 claims, made up as follows: De Beers, 622; Kimberley, 470; Wesselton, 1162; Bultfontein, 1753; and Dutoitspan, 1777. It is estimated that there are in sight in the various mines of the company 64,315,580 loads of blue ground as follows: In the De Beers mine, above the 2040-ft. level, 5,156,600 loads; Kimberley, above the 2520-ft. level, 1,367,080; Wesselton, above the 500-ft. level, 10,975,200; Bultfontein, above the 600-ft. level, 7,820,000; Dutoitspan, above the 750-ft. level, 31,900,000. This makes a total of 57,218,880 loads and with the ground on hand 64,315,580 loads. At the present rate of washing it is estimated that there is enough in sight to last 11 years.

The Voorspoed diamond mine was discovered in September, 1905,and a company was formed under the title of Voorspoed Diamond Mining Company, Ltd., with a capital of £400,000. The mine is situated on the Voorspoed farm, 20 miles north of Kroonstad, the nearest railway station being Honingspruit. The mine consists of 1200 claims and is proved to a depth of 500 ft. The workings consist of an open cut, and an inclined haulage track down to the 50-ft. level, the face of the cut forming a curve of, approximately, 300 ft. in length, the intention being to work the levels back to the limits of the pipe before attacking a fresh one. The mine is fairly clean and free from floating waste rock, locally known as "reef." The amount so far has not averaged more than 10 per cent.

A small temporary plant is now working, the intention being to erect one capable of handling between 1,500,000 and 2,000,000 loads per annum. Washing commenced in July, 1906, and the returns for four months showed 6723 karats from 28,523 loads of blue ground, or an average of 23.5 karats per 100 loads. The value is stated to be $7.20 per karat.

The diamond production of the Transvaal in 1906 was 1,069,391 karats, valued at £1,563,141. During the year ended Oct. 31,1906, the Premier Diamond Mining Company, near Pretoria, produced 899,697 karats,valued at £1,277,570, an increase of 54,000 karats over the previous fiscal year. Returns for the fiscal year inclusive show a recovery of 0.301 karats per load from 2,988,471 loads washed. The company's annual report states that the revenue derived from the mines for the year ended Oct. 31, 1906, amounted to £404,009. At the same time the ordinary shareholders in the company received £169,340 in dividends. Mining expenses amounted to £286,000, and sorting and washing to £183,000. Development work

cost £42,000, extra equipment, £55,000, and office expenses, marketing expenses, etc., brought the total for the year to £660,000. The water difficulty was surmounted by the recent completion of the pumping station on the Wilge river, and a water supply of 2,000,000 gal. per 24 hours is assured. Additional pumping plant is to be erected before long by means of which this supply will be doubled. New equipment is also under consideration in order to increase the output. It is not probable that this additional expenditure will be embarked on until the famous Cullinan diamond, weighing $3024\frac{3}{4}$ karats and valued at more than $2,000,000, is disposed of. This diamond figures in the assets of the company at a purely nominal sum. When it is disposed of, the proceeds will provide the additional capital required for extending the plant. The company now disposes of its output itself, the original contract with Neumann's having expired.

Reports have recently emanated from South Africa to the effect that the De Beers Company is considering the establishment of a diamond cutting industry which will give employment to 15,000 diamond cutters. A correspondent of the *Diamond Fields Advertiser* points out that such an industry would be impracticable and impossible, and brings out some important facts regarding diamond cutting in Europe and elsewhere. There are about 14,819 men employed in the business, distributed as follows: Amsterdam, 9000; Antwerp, 4000; St. Claude, 700; New York, 400; Paris, 150; Hanau, 150; Idar, 125; Bienne, 100; Geneva, 75; Nemours, 50; Steinbach, 25; London, 24; Lucerne, 20. In Amsterdam and other European cities a rose-cutter (usually an apprentice) receives from $4@6 per week, while a cleaver, generally an experienced workman, is paid about $40 per week. The average weekly wage for all classes of work is about $16. In New York the diamond cutter is able to earn from $55@60 per week, but few workmen are attracted to this country on account of the higher cost of living.

Emerald.

Austria.—The most important emerald mines of Europe are in Austria, located in the Salzburg mountains, seven miles from Habach station on the narrow-gage railway from Zell-am-See to Humel. Until recently they were being worked in a dilatory manner and in 1902, the last year in which anything like active mining operations were carried on, about 68,000 karats were turned out by six miners in less than four months. These mines were thoroughly examined and more or less prospected in 1905 by Spargo & Sons, of Liverpool, for the Northern Mercantile Corporation, Ltd., of London and Manchester, which on receipt of report purchased them.

Colombia.—Most of the world's emeralds come from Colombia, where the mining is a government monopoly. Although found in various places near Bogota, they are mined only at Muzo, about three days' mule riding

north from the capital. In the past, short leases were made to parties who paid the Government a royalty of 80 per cent. of the stones they found, and even so, the leases were profitable.

The emeralds are found on a mountain side, the gem-bearing strata alternating with barren strata. Under short leases the only way a mine could be worked was by catching the rainfall in little reservoirs on the slope above, and as water was collected, at rare intervals, booming off the picked surface. As perfect emeralds are worth more than diamonds and the demand for the beautiful stones is constant, the Government has decided to modernize mining. A constant and abundant supply of water will be brought from a distance of 15 miles and a monitor installed, with sluices and all the paraphernalia for preventing the loss of any stones.

Jade.

Burma.—This country continues to be the only one producing jadeite, the mines being situated in the Myitkyina district of upper Burma. The output in 1905 amounted to 122 metric tons, valued at $1,227,370, as against 171 metric tons in 1904. Canton is the chief center of trade in southern China and it is difficult to obtain jade in uncut form except from the Chinese. Jade is a favorite gem in China and is highly prized, especially the light green varieties. There are two distinct varieties included under the name. The first is known as nephrite and is a compact variety of hornblende. When the color is nearly white it is classed with tremolite; when a distinct green it is included under actinolite. The other is known as jadeite and is a silicate of aluminum and sodium.

Onyx.

Mexico.—For the exploitation and development of the large onyx deposits at Jimulco the French-Mexican Onyx Company was formed in Paris, with a capital of 2,550,000 fr. This company will also open up the onyx beds at Sombreretillo, Durango. It is said that the company has contracted to sell its entire product at 32c. per cubic foot.

Opal.

New South Wales.—At the White Cliffs opal fields there were found several parcels of high grade opal which realized more than $100 per oz. One specimen weighed 30 oz. and was won at a depth of 39 ft. from the surface. It is stated that the miners have organized a selling combination with headquarters in the United States and Europe to regulate the price.

Ruby.

Burma.—No new ruby-bearing locality was discovered in Burma in 1905.[1] The value of the Burmese stones that year was $140,000 or prac-

[1] *Indian Engineering*, Jan. 5, 1907.

tically the same as in 1904. Plague caused dulness in the Mandalay ruby market.

The Burma Ruby Mines, Ltd., made a comparatively small profit during 1905, distributing only £7475 in dividends, or at the rate of 5 per cent. During the year, 1,773,129 loads of earth were treated, at a cost of 17c. per load, as compared with 1,907,624 loads at 14c. per load during the previous year. The decrease in the quantity and the increase in the cost are accounted for by the fact that the Choungzone is nearly worked out. In the course of a few months it will be exhausted, and the company will then start operations on the adjoining Myntada mine, which is now being developed. The income of the company was £87,000 from sale of stones, and £12,595 from royalties received from native miners, the latter being one-half of that received two years ago. The great work in connection with the valley drainage tunnel is about half completed, and should be finished in 1908.

QUICKSILVER.

The production of quicksilver in the United States in 1906 was 28,293 flasks, valued at $1,157,184, against 30,705 flasks, worth $1,217,652, in 1905. California is still the mainstay of the industry. The Texas quicksilver district, about which such high hopes have been entertained, has not yet become an important contributor to the output, although from no lack of enterprise. Distance from railroads, and a variety of natural drawbacks are the retarding features. The old Sacramento mine of Mercur, Utah, is making a small output of the metal, in search of which the camp was first founded; development of the Black Butte, Ore., quicksilver mines continued through 1906, but no output of metal was recorded. An excellent reduction plant has been put up here, and a few experimental furnace runs have been made. A new cinnabar deposit in Yuma county, Arizona, is being developed by the Colonial Mining Company at a point 14 miles from Ehrenberg, but no furnace has yet been installed.

STATISTICS OF QUICKSILVER IN THE UNITED STATES.

Year.	Production.					Exports.			Imports.	
	Calif. (*a*) Flasks.	Texas. Flasks.	Others. Flasks.	Total. Metric Tons	Value.	Flasks	Metric Tons.	Value.	Pounds.	Value.
1896...	30,765	....	....	1,061	$1,075,449	19,944	692	$618,437	....	$2,037
1897....	26,648	....	....	919	993,445	13,173	475	394,549	45,539	20,147
1898....	31,092	(*b*)	153	1,077	1,194,746	12,830	445	440,587	81	51
1899....	29,454	261	....	1,025	1,416,790	16,518	573	609,586	131	83
1900....	26,317	1,700	233	974	1,279,436	10,702	353	425,812	2,616	1,051
1901....	26,720	2,932	75	1,031	1,382,305	11,219	389	475,609	1,441	789
1902....	29,552	5,252	....	1,208	1,515,714	13,247	459	575,099	*Nil.*	....
1903....	32,094	5,029	....	1,288	1,564,734	17,575	610	719,119	*Nil.*	....
1904....	28,876	5,336	700	(*c*)1,204	1 348,185	21,064	731	841,108	212	160
1905....	24,655	5,000	1,050	1,045	1,217,652	13,460	458	497,470	2,690	1,710
1906....	22,500	4,517	1,276	963	1,157,184	6,455	220	244,299	84	50

(*a*) Reported by the California State Mining Bureau. (*b*) Included in "Other States." (*c*) Estimated; the weight of the flask was changed from 76.5 ℔. to 75 ℔. within this year.

QUICKSILVER PRODUCTION OF THE WORLD.
(Metric tons.)

Year.	Austria.	Hungary.	Italy.	Mexico.	Russia.	Spain.	United States.	Total.
1896..................	564	....	186	218	491	1,524	1,036	4,019
1897..................	532	....	192	294	616	1,728	965	4,327
1898..................	491	....	173	353	362	1,691	1,058	4,128
1899..................	536	....	205	324	360	1,357	993	3,775
1900..................	510	32	260	124	304	1,095	983	3,308
1901..................	525	33	278	128	368	754	1,031	3,117
1902..................	511	45	259	191	416	1,425	1,208	4,055
1903..................	523	44	314	188	362	914	1,288	3,633
1904..................	536	45	357	(*e*) 190	393	1,020	1,192	3,733
1905..................	519	(*e*) 45	370	(*e*) 190	318	800	1,045	3,287
1906..................	526	(*e*) 45	418	(*e*) 200	210	1,242	963	3,604

(*e*) Estimated.

BRITISH IMPORTS AND EXPORTS OF QUICKSILVER. (a)

Year.	Exports Flasks.	Imports Flasks.	Price. Highest. £ s. d.	Price. Lowest. £ s. d.	Year.	Imports. Flasks.	Exports. Flasks.	Price. Highest. £ s. d.	Price. Lowest £ s. d.
1906.......	38,823	27,712	7 7 6	6 17 0	1901.....	35,341	26,863	9 2 6	8 17 6
1905.......	34,034	21,330	7 15 0	7 1 0	1900.....	32,725	25,869	9 12 6	9 2 6
1904.......	33,218	27,277	8 5 0	7 14 0	1899.....	51,696	32,239	9 12 6	7 15 0
1903.......	34,886	18,846	8 15 0	8 5 0	1898.....	54,563	34,014	7 15 0	6 16 0
1902.......	33,192	19,519	8 17 6	8 14 6	1897....	54,734	30,768	7 7 6	6 9 6

(a) Reported by Alex. S. Pickering, London.

Market.—The price of quicksilver in large lots at San Francisco for domestic delivery opened in 1906 at $39 per flask, but at the end of January rose to $39.50, at which figure it remained throughout the rest of the year. The price for foreign shipments ranged about $1.50 below the domestic price throughout the year until November, when it fell to $2 below. The price at New York remained steady at $41 throughout practically the entire year, except at its beginning and its close, when it was $1 lower. This steadiness in the American market is accounted for by the pooling of the output in California. The London price opened at £7 5s., rose at once to £7 7s. 6d. and remained there until midsummer, when it fell to £7 per flask for the remainder of the year.

QUOTATIONS FOR QUICKSILVER IN LARGE LOTS.

Month.	1903. New York.	1903. San Francisco. Domestic.	1903. San Francisco. Export.	1904. New York.	1904. San Francisco. Domestic.	1904. San Francisco. Export.	1905. New York.	1905. San Francisco. Domestic.	1905. San Francisco. Export.	1906. New York.	1906. San Francisco. Domestic.	1906. San Francisco. Export.
Jan...	$47.75	$45.50	$43.50	$45.75	$44.50	$41.50	$40.00	$40.42	$39.17	$40.25	$39.13	$37.63
Feb...	47.00	45.50	43.00	45.50	44.50	41.50	40.00	38.88	37.63	41.00	39.50	38.00
Mar...	47.00	45.00	42.50	45.40	44.50	41.50	38.95	38.15	36.90	41.00	39.50	38.00
Apr...	47.25	44.50	42.00	45.00	44.50	41.50	38.25	38.00	36.70	41.00	39.50	38.00
May..	47.50	44.50	42.00	44.94	44.19	41.81	38.38	38.25	37.00	41.00	39.50	38.00
June..	47.50	44.50	42.00	44.75	43.30	42.70	38.50	37.85	36.50	41.00	39.50	38.00
July..	47.50	44.50	42.00	43.81	43.50	41.94	41.25	39.00	37.75	41.00	39.50	38.00
Aug...	47.50	44.50	42.00	43.50	43.50	41.75	40.50	39.00	37.75	41.00	39.50	38.00
Sep...	47.50	44.75	42.00	41.40	42.45	40.85	40.00	39.00	37.75	41.00	39.50	38.00
Oct...	47.50	44.75	41.87	40.00	41.75	41.75	40.00	39.00	37.75	41.00	39.50	38.00
Nov..	46.75	44.00	41.50	40.00	41.75	41.75	40.00	39.00	37.75	40.75	39.50	37.50
Dec...	46.50	44.00	41.50	40.00	42.25	41.00	40.00	39.00	37.50	40.75	39.50	37.50
Year..	$47.27	$44.67	$42.15	$43.34	$43.39	$41.63	$39.65	$38.80	$37.52	$40.90	$39.47	$37.89

Quicksilver Mining in the United States.

Arizona.—Quicksilver deposits occur in Yuma county. The Colonial Mining Company is developing a mine at Cinnabar, 14 miles from Ehrenberg. A shaft, 180 ft. deep, has been sunk on a 6-ft. vein carrying 5 per cent. of cinnabar. Arrangements are being made to erect a reduction plant.

California.—(By Chas. G. Yale.)—California for over half a century was

the only section of the country which produced quicksilver, and it still yields the largest proportion of the product, but the industry is not as remunerative as it was formerly when there was no cyanide process and when hundreds of silver mines were worked where one is now. In 1906 the production decreased, and the domestic sales decreased in direct proportion to the decreased yield. There were in 1906 virtually no exports to China, and only a limited amount to Japan. This was a good thing for the California producers, because the quicksilver sold in the Orient brings less price than that sold in domestic market. It is sold abroad only as a matter of necessity, to dispose of the surplus product of this country, and these foreign sales reduce the general average price received by the California producers. The latter really did better in 1906 than in the previous year, though their product was materially less. Mexico, one of the California markets for quicksilver, has greatly decreased its annual demand because its own native product is preferred, and the Mexico mines are now producing at a good rate. This leaves a surplus to supply the increased requirements of the Eastern States.

The average prices for mercury throughout the United States in 1906 were $38.50 to $39.50 per flask of 75 lb., the lowest being naturally near the points of production in California and Texas. Sales of California quicksilver might have been made in China at an offered price of $30 per flask, but this offer was rejected. What very little was shipped to the Orient brought $31.50 per flask.

If the prices of silver are maintained, certain silver mines with lower-grade ores may resume operations, using the amalgamation process, which will create a demand for quicksilver. The higher-grade silver ores are now shipped to smelters, so no mercury is used in their reduction. Indeed, it is to the increased use of the cyanide process that the decreased demand for mercury is due, together with the present custom of shipping ores to smelters.

The New Idria mine in San Benito county, which is yielding 600 flasks per month and is supposed to be in a condition to maintain this rate of production for many years to come, is now the only California quicksilver mine paying dividends. The famous old New Almaden mine in Santa Clara county is now increasing its output, as is also the Socrates at Pine Flat, Sonoma county. The Napa Consolidated is yielding less by one-half than formerly. Some of the San Luis Obispo county mines have closed down, but others are working. One new mine in Stanislaus county recently started up, but very few new quicksilver properties in California are being put into operation.

The report of the Napa Consolidated Quicksilver Mining Company, for the year ended Dec. 31, 1906, showed the following statistics:

	1906	1905	1904		1906	1905	1904
Receipts	$47,327	$145,432	$164,858	Tons ore reduced	21,745	36,095	33,449
Op. expenses	92,780	149,423	162,464	Flasks quicksilver prod	2,130	4,145	4,400
Deficit	$45,453	$3,990	*$2,394	Drifts and tunnels run	3,840	5,955	6,757

*Net earnings.

Prospecting was carried on largely by boring with the Calyx drill. Forty-one holes were bored, some of which showed good colors and will be developed further later. As a whole, however, the results were unsatisfactory. During the latter part of the year, however, an improved grade of ore was developed in the tunnel levels.

The annual report of the New Idria Quicksilver Mining Company for the year ended Dec. 31, 1906, showed the following statistics:

	1906	1905	1904		1906	1905	1904
Sales	$253,119	$266,542	$324,324	Previous surplus	$141,927	$154,886	$136,185
Op. expenses	189,550	179,502	185,622	Surplus Dec. 31	$125,496	$141,927	$154,886
Net earnings	$63,568	$87,040	$138,701	Tons ore reduced	50,339	51,622	48,724
Dividends	80,000	100,000	120,000	Flasks produced	7,200	7,650	8,400
Balance	*$16,432	*$12,960	$18,701	Drifts run, ft	2,355	2,003	2,243

*Deficit.

The surface workings were good during the year. The ore is being gradually cut off in lifts or terraces, the intention being eventually to work by open cut to No. 3 level. The different levels below No. 1 are being worked steadily, and are largely in good ore. At No. 7 level the developments keep up well with the previous year, showing the orebody to be very large and of good grade, and consequently giving large reserves. Work on No. 10 tunnel was continued and in December, just two years after starting it, it was practically completed, having reached the bottom of the winze, coming down from No. 7, a distance of 3200 ft. Connection has been made and the work of drifting on the ore will soon be commenced. Between this level and the No. 7 level is a distance of about 350 ft. of virgin ground, for future stoping. No. 3 furnace has been practically completed, with the exception of a few details, which will soon be finished, and then will be put in commission. This will give three first-class furnaces, and add to producing capacity.

According to the *Mining and Scientific Press*, Jan. 5, 1907, during the last five months of 1906, 35,000 tons of ore were taken out of the Florence Mack mines in San Benito county, near San Carlos peak, which contained 51 to 54 lb. of quicksilver per ton. There is now enough material uncovered to run a 10-ton furnace regularly, but the owners have decided to keep on blocking out ore until next summer, when it will be possible to put in an adequate furnace and keep it going continuously.

The Socrates Quicksilver Company, of Sonoma Co., was reported to have opened a vein 22 ft. in width, assaying well in mercury. At the Eureka & Rattlesnake mines, Pine Flat, also in Sonoma county, a furnace was put in. The Phoenix mine, in Stanislaus county, was re-opened.

The Bonanza Quicksilver Mining Company did some work on its mines, 20 miles from Coalinga, Fresno county. A new mine was opened by Barton Bros., of Hanford, in San Benito Cañon, San Benito county.

Oregon.—The furnace of the Black Butte company was put in operation for an experimental run, but no commercial production was reported. The ore at Black Butte is very low in grade, but is said to be extensive in occurrence. It is found in a network of veins in andesite.

Texas.—Only four of the quicksilver mines in the Terlingua district produced any quicksilver in 1906: the Marfa & Mariposa Mining Company; the Lone Star Mining Company; the Chisos Mining Company; and the Texas Almaden Mining Company, otherwise known as the Dallas Mining Company. The last is at Big Bend and all the others are at Terlingua. The Lone Star is now closed down. The Chisos did not smelt its ore in its own furnace, although extensive additions to the plant are now under erection, including a 10-ton Scott furnace. This company has heretofore been hauling its ore to the works of the Colquitt-Tignor company, a distance of six miles. Two other former producers, the Terlingua Mining Company and the Big Bend Cinnabar Mining Company remained idle in 1906, while a third, the Colquitt-Tignor, it is rumored, will be reopened. The only mines active at this moment are thus the Marfa & Mariposa, and the Texas Almaden, of which the former is by far the larger producer.

The Scott furnace is the favorite for reducing the ore. Wood is the only fuel employed, and this is becoming exceedingly scarce. The upper Cretaceous shales of the district carry both oil and gas, and a fair grade of lubricating oil is not infrequently obtained with the mercury in the condensers. Coal of inferior quality is close at hand, and it has been suggested that this might be utilized for retort fuel through the medium of the gas producer.

The Texas Almaden Mining Company has recently completed a model reduction plant, containing a 20-ton Scott furnace, whose distinguishing feature is the use of extra hard and dense brick set with special care to reduce the width and permeability of the mortar joints, for lining the condenser. By this means the soakage of quicksilver was greatly reduced, and permitted the recovery of 60 flasks of mercury in less than one month after the retorts were charged with ore; this is a noteworthy accomplishment in mercury smelting. The condensers are also housed under well ventilated roofs, whereby they are kept cool, even during the hot weather

of that district. The company's mine is temporarily closed while a drainage hole is being drilled, but operations will shortly be resumed.

Utah.—The production of quicksilver in this State in 1906 was made by the Sacramento mine, at Mercur. It is reported that its reserves of high grade ore are now nearly exhausted, although there is a good deal of low grade ore left.

Washington.—The Washington Quicksilver Mining Company was reported developing a deposit of quicksilver on Camp creek, near Ellensburg, Kittitas county.

Quicksilver Mining in Foreign Countries.

Austria.—A new mercury deposit is said to have been discovered at Wiederschwing, in Carinthia (Austria), a little to the north of the district of Carniola, where similar deposits extend from Idria to Neumarkt. The new deposit is said to be extensive.

Canada.—A discovery of quicksilver ore, running from 1.5 to 7 per cent. mercury, is said to have been made on the west coast of Vancouver island. Cinnabar is also reported to have been found on Hardie mountain, near Kamloops.

Italy.—The mines of Monte Amiata, together with the metallurgical methods for the treatment of their ores, were described in a comprehensive paper in *Rassegna Mineraria*, Dec. 11, 1906.

Mexico.—The Bella Union mines, in Guerrero, were described in a paper by Juan D. Villarello, in *Memoires Societe Scientifique*, "Antonio Alzate" Vol. 23.

Russia.—The Nikitovsk quicksilver deposits, worked by A. Auerbach & Co., according to the *Torgovo Promysshlennaya Gazeta*, yielded 75,181 tons of cinnabar ore (462,564 lb. of quicksilver) in 1906, which compares with 82,767 tons of ore (696,436 lb. metal) in 1905, and 94,745 tons of ore (729,216 lb. metal) in 1904. This practically constituted the total production of Russia, as the Nikolaievsk mine at Daghistan yields comparatively little quicksilver.

Servia.—The quicksilver deposits at the Avala Berge, were described by H. Fischer, in *Zeit. f. praktische Geol.*, August, 1906.

Turkey.—According to *Rev. Min.*, Feb. 1, 1907, a new mercury mine in Karaburu, near Smyrna, Anatolia, recently commenced distillation of the mineral after installing modern Cermak-Spirek furnaces. Mining work was commenced in 1902 by Whittall & Co., of Constantinople, and the installation of Cermak-Spirek 12-ton furnaces was finished in May, 1906, since when everything has worked satisfactorily. The production has already attained 100 flasks per month.

The Metallurgy of Mercury.

The metallurgy of mercury was treated in an elaborate paper by Vincenzo Spirek, technical director of the Miniere del Siele e Cornacchino al Monte Amiata, read at the meeting of the International Congress of Applied Chemistry at Rome, Apr. 26–May 3, 1906. This paper was republished in a pamphlet of 26 pages. The paper is too long and detailed for satisfactory abstracting. Mr. Spirek is well known as a contributor to previous volumes of The Mineral Industry on the subject of the metallurgy of mercury.

Assay of Mercury Ore.

The following method of assaying mercury ores is given by Holloway in the *Analyst:* Mix the ore in a crucible with 10 grams of iron filings, through 60 mesh, and cover with 5 grams of iron filings through 30 mesh. Set the crucible in a hole in a tin plate or asbestos board in such a way that the bottom of the crucible may be heated by the flame, while the top remains cool. Cover the crucible with two pieces of clean silver foil which have been weighed, and which are of unequal size, the upper piece being the larger. Set on top of the silver covers a copper or metal vessel through which cold water may be passed in order to keep the plates cool. Heat for 20 min., allowing only the bottom of the crucible to become red hot. Cool the crucible, dry and weigh the covers. If tarry substances appear on the foil, they may be rinsed off with alcohol before drying. The amount of ore to be taken for assay depends upon its richness. For less than 1 per cent. of mercury use 2 grams; between 1 and 2 per cent. mercury, use 1 gram, and so on.

SALT.

The two sources of salt are deposits of natural rock salt and brines. Rock salt deposits are at present worked in New York, Pennsylvania, Ohio, Virginia, Michigan, Kansas, Louisiana and Oklahoma. The Californian deposits of rock salt are not now producing, having been flooded by the breaking of the Colorado river and the entrance of its waters into the Salton Sink. The production of this State now results only from the evaporation of sea water. Other States which obtain their salt from natural brines are Massachusetts, West Virginia, Ohio, Texas and Utah. Many of the rock salt deposits are not mined but are converted into artificial brines, which are pumped from the salt beds up to the surface through wells to be evaporated. All the deposits of salt are contaminated more or less by the chloride and the sulphate of other alkalies, and need to be purified by one or more crystallizing processes.

PRODUCTION OF SALT IN THE UNITED STATES. (*a*)
(In barrels of 280 lb.)

Year.	California.	Illinois.	Kansas.	Louisiana.	Michigan (*c*)	Nevada.	New York. (*c*)	Ohio, W. Virginia and Pa. (*b*)	Utah.	Other States.	Total Barrels.
1900..	621,857	(*d*)	2,233,878	(*d*)	7,210,621	(*d*)	7,897,071	1,669,156	249,128	987,631	20,869,342
1901..	601,659	99,700	2,087,791	451,430	7,729,641	13,781	7,286,320	1,385,257	334,484	569,092	20,566,661
1902..	682,680	90,009	2,158,486	399,163	8,131,781	14,829	8,523,389	2,318,579	417,501	1,112,824	23,849,221
1903..	629,701	(*d*)	1,555,934	568,936	4,297,542	(*d*)	8,170,648	3,043,135	212,995	489,238	18,968,089
1904..	821,557	(*d*)	2,161,819	1,095,850	5,425,904	(*d*)	8,600,656	3,030,829	253,829	639,558	22,030,002
1905..	664.099	(*d*)	2,098,585	1 055,186	9,492,173	(*d*)	8,359,121	2,728,709	177,342	1,390,907	25,966,122
1906..	806,788	(*d*)	2,198,837	1,179.528	9,936,802	11,249	8,978.630	3,436,840	262,212	1,361,494	28,172,380

(*a*) Statistics of the United States Geological Survey. (*b*) The production of Pennsylvania in 1906 is included in "Other States." (*c*) Includes brine used in manufacture of alkali. (*d*) Included in "Other States."

The course of the industry as regards culinary, packing, curing and dairying purposes is very regular, the production for those uses having remained at about the same figure for a number of years. The increasing importance of the chemical industry, which demands a large amount of salt, is responsible for practically all of the increase in the production which is to be noted in 1905 and 1906; this increase is most marked in Michigan. The accompanying tables give the statistics of production and the consumption of salt in the United States for a series of years, and the production of foreign countries.

PRODUCTION OF SALT IN FOREIGN COUNTRIES.
(In metric tons.)

Year.	1897	1898	1899	1900	1901	1902	1903	1904	1905	1906
Algeria......	23,222	21,300	17,378	18,325	18,518	27,263	26,329	18,563	27,000	(c)
Austria.....	331,084	341,959	342,059	330,277	333,238	310,807	359,014	369,877	343, 75	(c)
Canada.....	46,584	51,853	53,847	56,296	53,927	57,203	56,671	62,411	41,170	69,291
France.....	948,000	999,283	1,193,532	1,088,634	910,000	863,927	967,531	1,153,754	1,130,000	(c)
Germany...	1,306,684	1,370,341	1,432,181	1,514,027	1,563,811	1,583,458	1,693,935	1,701,654	1,777,557	1,867,952
Greece......	20,421	37,125	22,411	22,411	23,079	25,200	26,000	27,000	(c)	(c)
Hungary....	171,711	178,551	182,593	189,363	(a)211,321	174,882	183,327	187,620	195,410	(c)
India (d)....	937,888	1,043,828	977,240	1,021,426	1,120,187	1,056,899	908,911	1,188,900	1,425,010	(c)
Italy.......	31,526	29,745	28,842	367,255	435,187	458,497	488,506	464,326	437,699	(c)
Japan.......	621,731	646,719	390,433	669,694	659,118	620,820	657,489	(c)	(c)	(c)
Russia......	1,551,894	1,505,600	1,681,362	1,768,005	1,705,922	1,847,019	(c)	(c)	(c)	(c)
Spain.......	508,606	479,358	598,108	450,041	345,063	426,434	427,394	543,674	551,900	(c)
U. Kingdom.	1,933,949	1,908,723	1,945,531	1,873,601	1,812,180	1,924,273	1,917,184	1,921,899	1,920,149	1,996,593
U. States....	2,009,625	2,382,197	2,522,610	2,651,278	2,612,204	2,409,174	2,408,646	2,797,461	3,297,285	3,578,061

(a) Sales by the royal monopoly, including imports entered for consumption. (c) Statistics not yet published. (d) Does not include the untaxed output of certain native States.

CONSUMPTION OF SALT IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.		Imports.		Exports.		Consumption.	
	Amount.	Value.	Amount.	Value.	Amount.	Value.	Amount.	Value
1897...........	2,236,248	$4,920,020	209,025	$565,038	5,797	$52,320	2,439,476	$5,432,738
1898...........	2,465,769	6,212,554	185,530	588,653	8,640	63,624	2,642,659	6,737,583
1899...........	2,759,206	6,867,467	189,051	579,682	12,600	86,465	2,935,657	7,360,684
1900...........	2,921,708	6,944,603	199,909	634,307	7,511	65,410	3,114,106	7,513,500
1901...........	2,879,332	6,617,449	201,733	676,324	9,433	86,414	3,071,632	7,207,359
1902...........	3,338,892	5,668,636	184,764	647,554	5,094	55,432	3,518,562	6,260,758
1903...........	2,655,532	5,286,988	165,981	495,948	12,750	95,570	2,808,763	5,687,366
1904...........	3,084,200	6,021,222	166,140	467,754	13,964	113,625	3,236,376	6,375,351
1905...........	3,635,257	6,095,922	161,159	492,189	34,238	239,223	3,762,178	6,348,888
1906...........	3,944,133	6,658,350	170,505	502,583	33,988	274,627	4,080,650	6,886,306

Bibliography.

Reviews of the salt mining industry both in the United States and in foreign countries, together with statistics of production and imports will be found in every volume of THE MINERAL INDUSTRY. Special articles dealing with the technology of salt are as follows: In Vol. I, a 10-page article by F. E. Engelhardt, discussing the geographical distribution of salt deposits in the United States, with notes upon their size, character, and the methods of working them. Vol. II contains brief references to preparing salt from solutions by evaporation, by steam heat and by solar heat. Vol. V describes the salt deposits of Louisiana and the method of working them, and has also an 18-page article by R. Helmhacker, dealing with every phase of the salt industry in Austria.

Following are some of the articles and patents relating to salt, which have recently been published:

"The Remington Salt Co." C. S. Palmer. (*Eng. and Min. Journ.*, June 30, 1906.) Technical data respecting the production of salt at Ithaca, N. Y.

"Salt Deposits and the Salt Industry in Ohio." J. A. Bownocker.

(Geological Survey of Ohio, Bull. No. 8, June, 1906; 42 pp.) Monograph describing the deposits of salt in Ohio, with their locations, and a brief catalog of the salt-producing works.

"Die Gewinnung des Kochsalzes in der Türkei." C. Mayer. (*Chem.-Zeit.*, Nov. 10, 1906.) Describes the methods of extracting sodium chloride from sea water and brine wells in Asia Minor.

"Purifying Rock Salt." H. Tee, Liverpool. Purifying rock salt by melting it and forcing steam and hot air through it. Brit. pat. 6611 of 1905.

"Salt-Pan By-products." F. Bale, Droitwich. An improved process for treating the scale of salt pans by means of which products of value are extracted. Brit. pat. 10,722 of 1905.

SILICA.

Silicious materials are produced and marketed under a variety of names. Chief among the substances of this class are flint, quartz, diatomacious earth, pumice, novaculite and sand. A considerable part of the silica produced is made up into grindstones, whetstones, and similar articles for sharpening tools. White, milky quartz, free from iron, finds more or less application in packing the towers of sulphuric acid works. Silica in the form of sand is used extensively in the manufacture of silica brick and is the chief ingredient in the glass making industry. The amount of silica which is used in making sandpaper is very limited and is decreasing on account of the superiority of emery and abrasive cloths made from artificial substances. Considerable quartz is used in the stone cutting trade, especially among marble dealers, in cutting the blocks into slabs by means of gang saws.

In metallurgical operations quartz in the form of flint pebbles is becoming of considerable importance as a means of pulverizing ores in ball mills, also in grinding cement clinker. The largest amount of these flint pebbles is imported from La Somme district of France, where the picking of the pebbles from the Cretaceous beds is controlled by one large firm in Dieppe, having an agent in New York. The French pebbles are of the best quality and sell at \$8.50@\$11 per long ton at New York. Shipments are made in wooden barrels and burlap bags and the principal places of import are New York, Philadelphia, New Orleans and Galveston.

The annual output of lump quartz is about 20,000 tons. The import of French pebbles during 1904 and 1905 was between 7000 and 8000 long tons, brought in principally as ballast.

SILICON.

By F. J. Tone.

Pure silicon is one of the many new metallurgical products the manufacture of which has been made possible by the electric furnace. According to the estimate of F. W. Clarke of the Geological Survey, silicon oxide forms 58.3 per cent. of the contents of the solid crust of the earth, aluminum oxide 14.7 per cent., iron oxide 7.8 per cent., calcium oxide 5.3 per cent., and magnesium oxide 4.5 per cent. In other words elemental silicon constitutes 27.2 per cent. of known geological strata. Notwithstanding this wide occurrence and predominance of silicon in mineral formations, no earlier than the year 1900 metallic silicon was sold at 40c. per gram and was available only in quantities such as made it a laboratory product.

The process of manufacturing metallic silicon as carried out by the Carborundum Company at Niagara Falls employs electric furnaces of 1200 h.p. capacity, which produce several hundred pounds at each tapping.[1] A mixture of coke and sand is subjected to high temperature in an electric furnace especially designed to give close temperature regulation and to prevent volatilization of the reduced metal. Silicon belongs to a class of elements whose temperatures of reduction and volatilization are very close and for this reason its manufacture has heretofore offered great difficulties. The process is continuous, the charge mixture being fed in at the top of the furnace and the molten silicon tapped out from time to time in pigs of 400 to 500 ℔.

Silicon produced in the electric furnace is a crystalline body generally quite dense and of bright silver luster. Compared with the metals it is quite brittle and stands between 6 and 7 in the scale of hardness. The specific gravity is 2.34 and the melting point about 1430 deg. C. Various grades of metal are produced, running from 90 to 98 per cent. silicon, the purity depending for the most part on the purity of the crude materials used. The following are typical analyses:

Silicon	98.10%	95.30%
Iron	1.20	1.06
Aluminum	.60	1.54
Carbon	.10	2.10

Electrical Uses.—Silicon has important uses in the electrical industries.

[1] U. S. pats., 745, 122 and 842, 273, F. J. Tone.

Its resistance is about five times that of carbon and the temperature coefficient, which at normal temperatures is positive, changes at red heat and becomes negative. It may be cast into rods for various resistance purposes and its high melting point and resistance to oxidation, even at red heat, make it superior to other metals used for this purpose. It has been proposed by Prof. Elihu Thompson to employ granular silicon in electric heaters and cooking apparatus.[1] H. E. Heath has exhaustively investigated the electrical and chemical properties of silicon and has developed a thermo-electric couple comprising as one of its elements silicon and as the other copper or nickel.[2] This couple has the advantage of high electromotive force. With a rise in temperature of about 600 deg. C. the electromotive force of the couple amounts to one-quarter of a volt. This has an interesting application in thermometry.

A silicon wireless detector was developed during 1906 by G. W. Pickard.[3] A piece of silicon is pressed into good electrical contact between two conducting blocks as in the ordinary coherer. The energy of the oscillation set up in the receiving aerial is converted into heat at the junction of the silicon and the metal forming the conductor blocks by virtue of the high resistance of the former and the low resistance of the latter. The amount of heat developed by the high thermo-electromotive force and the consequent temperature rise is proportional to the square of the resistance. This detector is called a "thermo-electric regenerator detector" from the fact that thermal energy is converted or regenerated into a direct electric current, the detector performing the usual function of a very delicate relay. The advantages of the new detector are, that it fulfills all requirements of commercial wireless telegraphy as to sensitiveness, speed, stability, and freedom from delicate adjustments; that the sensitiveness is in no wise impaired by severe static discharges and is unaffected by changes in atmospheric temperature or humidity; and that it has no limited number of operative contact points as in the case of a crystalline mass which operates by the variation of resistance due to a heating of a small part of the mass.

Metallurgical Uses.—Since the high grade ferro-silicons have come into use by steel makers, the advantages of high silicon alloys have become well recognized. For many open-hearth heats it is desirable to introduce the alloy into the ladle in the most concentrated form possible in order to avoid chilling effects, and in such work pure silicon is being used to great advantage. Silicon is used widely in the manufacture of alloys for copper and bronzes, being a most important deoxidizing agent in the making of pure copper castings.

Silicon combines with oxygen with the evolution of great heat, the heat

[1]U. S. pat., 796, 684.
[2]U. S. pat., 824, 015.
[3]U. S. pat., 836, 531. *Scientific American*, March 16, 1907.

of formation of silica being 219,000 calories. It is proposed[1] to use powdered silicon and oxygen compounds for the generation of heat for repairing castings, welding and similar purposes in the same manner as aluminum is used in the alumino-thermic process. Mixtures of silicon and magnesium together with an oxygen compound are claimed to have advantages in similar thermic reactions due to the great fluidity of the slag which is formed as the result of the reaction.[2] T. Goldschmidt has recently proposed a mixture of calcium and silicon for the same purpose.[3]

Chemical Uses.—A highly important application of silicon promises to be the manufacture of silicon chemical ware. Silicon can be readily cast into forms, its melting point being about that of steel. Ordinary graphite crucibles may be used for this purpose and although it might be expected that the silicon and carbon of the crucible would unite to form carborundum, no such reaction takes place if the temperature is not excessive. Silicon completely resists the action of almost all acids. Sulphuric, hydrofluoric, nitric, and hydrochloric acids, alone have no effect upon it. When subjected to a mixture of hydrofluoric and nitric acids it is oxidized to silica. This property in conjunction with its ability to be cast in molds, makes its application in the chemical industries of great interest. For the concentration of sulphuric acid, evaporating vessels of silicon would be a considerable advance over the cast iron and platinum vessels now in use. Small silicon crucibles have been found of great utility in the laboratory in conducting acid fusions.

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[1] *Trans.* Am. Electrochem. Soc., VII, 246.
[2] Brit. pat., 3089 of 1906.
[3] French pat., 361, 197 of 1905.

SODIUM AND SODA SALTS.

By Reginald Meeks.

The production of metallic sodium in the United States is almost entirely for conversion into certain salts of soda, principally sodium cyanide. The annual production in the United States, Great Britain and Germany is approximately 3500 tons, divided nearly equally between the three countries. Of this amount about 1500 tons is used for sodium cyanide, 1500 tons for sodium peroxide and 500 tons is sold as metallic sodium. Practically all is produced by the Castner process, the patents for which are beginning to expire.

Manufacture of Metallic Sodium.

Castner Process.—The material used in this process is a pure form of caustic soda electrolyzed between electrodes of copper and nickel. The costs are: caustic soda, 5c.; power, 1 to 5c.; labor, 2.5c.; maintenance and fixed charges, 2c.; total, 10.5 to 14.5c. per lb. This method, the invention of the late H. Y. Castner, has been in successful operation at Niagara Falls, N. Y. for many years.

Ashcroft Process.[1]—This process is the invention of E. A. Ashcroft and is a combination of the Acker and Castner processes. Unlike the Castner process the material treated is common salt ($NaCl$) and it is claimed that metallic sodium may be produced more cheaply than with the older method. On a similar scale the cost is estimated as follows: common salt, 0.5c.; power, 1 to 5c.; labor, 1c.; maintenance and fixed charges, 2.5c.; total 5 to 9c. per lb.

The process consists of electrolyzing fused sodium chloride in a double electrolytic cell. In the first cell there is a molten lead cathode, which forms an alloy of lead and sodium; this is transferred to the other cell and is used as anode in a bath of fused caustic soda, yielding metallic sodium at the cathode. The caustic soda is not consumed and the by-product, chlorine, may be used in making many chlorine products. The power required in the Ashcroft process is the same as in the Castner process, but it is claimed that the electrical efficiency is higher than in the latter.

Sodium Transmission Lines. (By A. G. Betts.)[2]—The electrical con-

[1] *Electrochem. and Met. Ind.*, June, 1906.
[2] Abstract of an article in *Electrical World*, Nov. 10, 1906.

ductivity of sodium by weight is 115 and by volume, 31.4; copper has a conductivity of 37.5 by weight and 97.6 by volume. There are five metals (sodium, calcium, potassium, aluminum and magnesium) with a higher conductivity by weight than copper, but only one (silver) which compares better by volume. Of the common metals, sodium has the greatest conductivity per unit of weight. Comparing it with calcium, potassium, aluminum and magnesium, it can be the most cheaply prepared. The cost of making it by the Castner process is probably from 10 to 14c. per ℔. It is believed the Ashcroft process will reduce the cost to 7.5c. when mechanical difficulties are overcome. Five kilowatt-hours per pound is a fair estimate of the energy required and the cost of common salt is low. The by-product, chlorine, is also available for sale.

To use sodium as a conductor of electrical energy it is only necessary to enclose it in suitable iron or steel pipes, which should be of convenient lengths and provided with connections or joints. The accompanying table gives comparative data for sodium and copper transmission lines under equal current carrying-capacity.

COST OF SODIUM AND COPPER CONDUCTORS.

Size of wrought iron pipe	0.5 in.	1 in.	1.5 in.	2 in.	3 in.	4 in.	5 in.	6 in.
Current carried by iron (800 amp. per sq.in. copper) amperes	29.9	59.2	96	128	269	380	518	670
Lb. sodium per ft	0.128	0.36	0.86	1.41	3.10	5.35	8.40	12.14
Current carried by sodium (same basis) amp.	79	222	526	864	1908	3270	5152	7460
Total current, iron and sodium, amp	109	281	622	992	2177	3650	5670	8130
Cost of pipes per ft	2.5c.	4.175c	. 6.75c.	9.0c.	17.85c.	26.67c.	36.25c.	46.9c.
Cost of sodium per ft. @ 7½c. per ℔	0.96c.	2.7c.	6.45c.	10.58c.	23.25c.	40.13c.	63.0c.	91.05c.
Total cost per ft	3.46c.	6.87c.	13.3c.	19.58c.	41.1c.	66.8c.	99.25c.	$1.38
Cost of equivalent copper @ 16c. per ℔	8.38c.	21.6c.	48c.	76.3c.	$1.67	$2.81	$4.37	$6.21

This table was arrived at by calculation only; some of the results have been verified by experiments with a 130 ft. transmission line using 1.5 in. wrought iron pipe and 120 ℔. of metallic sodium.

I built an experimental line early in 1906, using standard wrought-iron pipe of various lengths up to 17 ft. A line built on so small a scale would naturally cost more than a long commercial transmission line. The pipes were joined, and at each end an elbow was loosely connected and two short vertical lengths were attached. The whole was then heated to above the melting point of sodium. The molten sodium was poured in until the metal had risen in the vertical pipes and above, the pipes being filled. This was to allow for contraction during cooling. Copper connections were inserted at each end and a current of 500 amperes was passed through the line showing a resistance of 0.00001 ohm per foot. Ten months' exposure to weather showed no apparent change in the conductor except rusting which could probably have been avoided by using a better quality of paint.

There are limitations to the use of sodium conductors. Heavy work

above ground will not permit of their use and they should not be placed near wooden or non-fireproof buildings. In case of fire near the pipes, the heat would tend to melt the sodium and if water were brought in contact with it the result would be disastrous. Transmission lines through the country could be constructed without meeting with these difficulties.

Nitrate of Soda in 1906.

The year 1906 was very prosperous for producers and importers but the consumers were obliged to pay high prices. In spite of fairly good supplies the market was so well supported and manipulated that a high level of prices was easily maintained and the year closed with spot deliveries for 96 per cent. grade bringing \$2.55 per 100 lb. f.o.b. New York. The lowest figure was \$2.20, in January; the highest, \$2.625, in October; the average for the year was \$2.378.

NITRATE OF SODA STATISTICS. (a)
(In tons of 2240 lb.)

Year.	Shipments from South America.	Consumed in Europe.	Consumed in United States.	Consumed in World.	Stocks in Europe.	Visible Supply at close of 1906.
1899	1,373,000	1,140,000	160,000	1,330,000	236,000	741,000
1900	1,429,000	1,126,000	175,000	1,324,000	221,000	794,000
1901	1,238,000	1,154,000	192,000	1,364,000	243,000	617,000
1902	1,360,000	1,028,000	214,000	1,259,000	263,000	660,000
1903	1,435,000	1,127,000	265,000	1,412,000	155,000	654,000
1904	1,476,000	1,131,000	275,000	1,447,000	162,000	672,000
1905	1,623,000	1,190,000	308,000	1,547,000	183,000	674,000
1906	1,719,000	1,243,000	358,000	1,641,000	190,000	749,000

(a) Statistics of W. Montgomery & Co., London.

IMPORTS OF SODIUM NITRATE INTO THE UNITED STATES.
(In tons of 2240 lb.)

Year.	Quantity.	Value.	Value per ton.	Year.	Quantity.	Value.	Value per ton.
1897	94,965	\$2,810,187	\$29.59	1902	205,245	\$5,996,205	\$29.21
1898	147,495	2,298,240	15.58	1903	272,947	8,700,806	31.88
1899	146,492	3,486,313	23.80	1904	228,012	9,333,613	40.93
1900	182,108	4,935,520	27.10	1905	321,231	11,206,548	34.89
1901	208,679	5,999,098	28.75	1906	372,222	14,115,206	37.92

Conditions which governed the markets for nitrate were: (1) pronounced scarcity of labor in the Chilean fields; (2) an ever increasing demand for the salt both in the United States and in Europe; (3) the earthquake at Valparaiso, which while not directly affecting the nitrate fields had a disturbing tendency and was made an excuse to elevate prices; (4) the rebuilding of Valparaiso attracted laborers, at higher prices, from the nitrate mines; (5) delays in loading vessels caused by a strike, followed by a lockout, of launchmen and loaders. These conditions

were made use of to raise the price of the salt and orders have been booked far ahead at the higher figures.

During 1906 ten new *oficinas*, having a combined estimated output of 285,000 tons per annum, entered the list of producers. These did not start operations until late in the year and their initial production had little effect on the total shipments from Chile, which exceeded those of 1905 by about 96,000 tons.

Agreement of Producers.—The Nitrate Propaganda Association met in March, 1907, at the close of its agreement, and drew up a new agreement, the particulars of which have not yet been published. In the first year of the combination, which expired March 31, 1906, according to the *Chemical Trade Journal* of London, the export quota was fixed at 31,091,095, Spanish quintals (101.41 lb.) which was successively raised until, upon the renewal of the agreement, the figure had reached 49,563,-606 quintals, although the consumption in the preceding year had not reached 35,000,000 quintals.

In the hands of the "combination" the price of nitrate of soda has steadily risen. In 1900, when negotiations were opened, the price was 4s. 6d. per quintal, f.o.b. Chile; in 1901 the price rose to 6s. 4½d.; the next year the average price was 6s. 6d.; in 1903 it was 6s. 8½d.; in 1904, 7s. 5d.; in 1905, 7s. 7⅛d.; and in 1906, immediately after the renewal of agreement, the price was 7s. 8d. for prompt delivery. Later in the year the price advanced, first to 8s., then to 8s. 6d. and 9s., and finally, in October, to 9s. 7½d. per quintal f.o.b. Chilean ports. At these high figures, many contracts were entered into for delivery two or more years forward.

Synthetic Nitrate. (By Alfred J. Lotka.)[1]—This process is based on the observation that the electric-disk discharge, produced by establishing an electric arc in a strong magnetic field, possesses to a peculiarly high degree the power of causing the combination of a mixture of oxygen and nitrogen. Such a disk-flame, fed by an alternating current, is inclosed within a flat furnace, built of refractory clay, with a copper mantle, a vigorous current of air being maintained through it. The furnace is placed between the poles of a powerful direct-current electromagnet; the copper electrodes are so close together that, but for the diffusing action of the magnetic field, they would be short-circuited. These electrodes are hollow and water-cooled; owing to this precaution, they have a long life. The furnaces are used at the present on a practical scale at Notodden, Norway; they consume, under normal conditions, 500 kw. (at 5000 volts); but this figure has occasionally been raised to 700 kw. without injury. The immense disk of flame in these furnaces is more

[1] *Eng. and Min. Journ.* May 4, 1905.

than $6\frac{1}{2}$ ft. in diameter, and probably represents the most extended electric discharge ever maintained for any considerable time.

It is estimated that the waterfalls (with an aggregate capacity of about 350,000 h.p.) controlled by the company which is exploiting the Birkeland and Eyde process, can furnish electric energy at a cost of about \$3 per h.p.-year. The yield of nitric acid obtained regularly hitherto has been about 500–600 kg. of HNO_3 per kw.

The gases issuing from the furnace contain only about 2 per cent. of nitric oxide. First they are cooled, the heat being utilized for concentrating the nitrate lyes finally obtained, and the cooled gases are then passed through reaction towers lined with acid-proof slabs. Here the combination of nitric oxide with the excess of oxygen is carried to its limit. The gases next enter granite absorbing towers filled with quartz, over which water is kept trickling from above. The nitric acid flowing out at the bottom is returned to the top, being circulated in this way until it reaches a concentration of 50 per cent. The residual gas, too dilute in oxides of nitrogen for further absorption by water, is passed through towers fed with milk of lime, and through a chamber packed with solid lime. The calcium nitrate thus obtained is subsequently treated with nitric acid, to obtain calcium nitrate solution and to liberate nitrous vapors. The latter are returned to the granite absorption towers.

At Notodden, the dilute nitric acid obtained is converted into calcium nitrate by neutralizing with limestone. The solution so obtained is united with that from the lime towers, and evaporated, the fused calcium nitrate being allowed to solidify in iron drums.

The plant at Notodden makes about 1500 kg. (3300 lb.) of nitric acid in the form of nitrate per day. Calcium nitrate promises to be in demand as raw material for various chemical processes. It is considered to be an excellent fertilizer, its freedom from sodium and chlorine giving it an advantage over Chile saltpeter.

The Markets for Soda Salts in 1906.[1]

The market for soda salts in 1906 was satisfactory; prices held firmly and the demand continued active throughout the year. Bicarbonate was sold on a basis of \$1.30 in bulk and \$1.50 in kegs, f.o.b. works, which was the same as in 1905. Some trouble was experienced with deliveries, due to poor transportation facilities. The demand for caustic soda was good and prices advanced 10c., to \$1.85 @ 1.90 for the 60 per cent. grade, according to quantity and terms of sale. Salt cake was active and prices held steady at 65c. per 100 lb. for bulk. Final arrangements were made with the glassmakers early in October for their next fire requirements and large orders were booked for future delivery. Prices

[1] *Oil, Paint and Drug Reporter*, Feb. 11, 1907.

for sal soda advanced from 70c. to 80 @ 85c. in October when supplies of American grades diminished. The foreign product held firmly at 85 @ 90c. all the year. Bleaching powder held firmly at $1.25 @ 1.30 per 100 ℔. until October, when the price advanced 5c., to $1.30, for carloads and for small lots $1.50 was asked. There was no change at the close of the year. Chlorate of soda opened at $8\frac{3}{8}$ @10c. in January, advanced to $8\frac{3}{4}$c. the next month and in March receded to $8\frac{1}{4}$ @ $9\frac{1}{2}$c., which prices continued until October. Contracts for 1907 were established at $8\frac{1}{2}$c., a gain of $\frac{1}{4}$c. over last year, and large sales were made at that figure for 1907 delivery. Prices remained steady for the remainder of the year.

STRONTIUM SULPHATE.

Strontium sulphate occurs in its natural form as the mineral celestite which contains 66.4 per cent. strontium oxide and 43.6 per cent. sulphuric anhydride. The mineral has but very few uses and its value is slight. Its principal use is probably in the process of refining beet sugar. It is a source of strontium nitrate which is used in making red fire and is also used to a limited extent as a pigment in the same manner as barytes.

The mineral is not treated in this country, but the small production which is made is shipped to Germany to be worked up there. The low price of the mineral in Germany makes it almost impossible to mine and export the mineral with profit to that country and those shipments which have been made were mostly experimental ones.

The last record of mining strontium sulphate in the United States is mentioned in the report of the Texas Geological Survey and the "Mineral Resources of the United States." According to these reports the deposit of celestite in Texas is located in the Mt. Bonnell and Mt. Barker districts and is owned by R. C. Walker, who in 1904 produced one carload of 17 tons of this mineral which was exported in Germany. No other deposits of strontium mineral were developed in the United States during 1904 nor was any mineral mined from any other United States deposit in that year or since.

In Vol. VI of THE MINERAL INDUSTRY will be found notes on the mineralogical characteristics of celestite with a brief account of preparing strontium hydrate from the mineral and its subsequent use in the process of sugar refining.

SULPHUR AND PYRITE.

The importance of American sulphur mining has been increasing rapidly in recent years, and this is especially true for the year 1906, when the aggressive policy of the Union Sulphur Company of Louisiana in threatening to invade European markets resulted in the winding-up of the Anglo-Sicilian Sulphur Company and the taking over of the Sicilian sulphur industry by the Italian Government. The major part of the American production of sulphur is mined in Louisiana. Other producing States are Utah, Nevada, Wyoming and California. The production from this region has never been very considerable, being only in the neighborhood of 4000 tons per annum, but in 1906 much more activity was manifested.

For commercial reasons, the Union Sulphur Company is not willing to report figures of its output, and so it is impossible to arrive at an accurate figure for the sulphur production of the United States. Consequently the output in 1905 and 1906 has had to be estimated upon the basis of an increased consumption of sulphur, the known imports of brimstone and the known production of pyrites. It is believed, however, that the figure for sulphur production, which is thus arrived at, is a close approximation to the actual. The accompanying table gives the statistics of sulphur and pyrite production, exports, imports and consumption for a series of years.

CONSUMPTION OF SULPHUR IN THE UNITED STATES.
(In tons of 2240 lb.)

Source.	1901	1902	1903	1904	1905	1906
Sulphur—Domestic production	6,866	7,443	35,098	193,492	215,000	285,000
Imports	175,310	176,951	190,931	130,421	84,579	74,440
Total	182,176	184,394	226,029	323,913	299,579	359,440
Exports	207	1,253	967	2,493	1,713	14,419
Consumption	181,969	183,141	225,062	321,420	297,866	345,021
(*a*) Sulphur contents at 98 per cent	178,330	179,478	220,560	314,992	291,909	338,121
Pyrite—Domestic production	234,825	228,198	199,387	173,221	224,980	225,045
Imports	403,706	440,363	427,319	413,585	515,722	598,078
Total	638,531	668,561	626,706	586,806	740,702	823,123
Exports	....	3,060	1,330	....	....	....
Consumption	638,531	665,501	625,376	586,806	740,702	823,133
Sulphur in domestic at 44 per cent	103,323	104,071	87,730	76,217	98,991	99,020
Sulphur in foreign at 47 per cent	189,742	205,532	200,215	194,385	242,389	281,097
Total sulphur content	189,742	309,603	287,945	270,602	341,380	380,117
Grand total sulphur consumption	471,395	489,081	508,505	585,594	633,289	718,338

(*a*) Includes crude and refined sulphur.

The statistics of pyrites production include a small amount of pyrrhotite, produced in Virginia, and also a small amount of marcasite produced in Wisconsin. Pyrite concentrates from silver and gold ores are not included in this statement of production. The output of pyrite for 1906 was 225,045 long tons, valued at $767,866, against 224,980 long tons, valued at $752,-936 in 1905. It will be noticed that the output increased but little in 1906, which was due to the suspension of operations of several producers in New York.

WORLD'S PRODUCTION OF SULPHUR. (*a*)
(In metric tons.)

Year.	Austria. (*d*)	France. (*c*)	Hungary	Germany	Greece.	Italy. (*b*)	Japan.	Spain.	United States.
1895	830	4,213	102	2,061	1,480	370,766	15,557	2,231	1,676
1896	643	9,720	138	2,263	1,540	426,353	12,540	1,800	3,861
1897	530	10,723	112	2,317	358	496,658	12,013	(*b*)3,500	1,717
1898	496	9,818	93	1,954	135	502,351	10,339	3,100	2,770
1899	555	11,744	116	1,663	1,150	563,697	10,241	1,100	1,590
1900	862	11,551	123	1,445	891	544,119	14,439	750	4,630
1901	4,911	6,836	137	963	2,336	563,096	16,548	610	6,977
1902	3,721	8,021	105	487	1,391	510,333	18,287	450	7,565
1903	4,475	7,375	135	219	1,266	497,615	22,914	1,680	35,660
1904	8,407	5,447	143	209	569	519,243	(*e*)	4,039	196,588
1905	11.788	4,637	135	205	(*e*)	530,510	(*e*)	3,815	218,440
1906	(*e*)	(*e*)	(*e*)	178	(*e*)	(*e*)	(*e*)	(*e*)	289,560

(*a*) From the official reports of the respective governments. The sulphur recovered as a by-product by the Chance-Claus process in the United Kingdom, amounting to between 20,000 and 30,000 long tons annually, is not included. (*b*) Crude. (*c*) Crude mineral; limestone impregnated with sulphur. (*d*) Crude rock. (*e*) Not yet reported.

Industrial Conditions.

Conditions during 1906 changed materially. The domestic production of sulphur increased considerably, and whereas American consumers formerly depended chiefly upon the importations from Sicily, they now are supplied by the domestic output. Importations during the last few months of 1906 showed a large decrease in comparison with the previous year.

IMPORTS OF SULPHUR INTO THE UNITED STATES.
(In tons of 2240 lb.)

Kind.	1903		1904		1905		1906	
	Amount.	Value.	Amount.	Value.	Amount.	Value.	Amount.	Value.
Crude	188,888	$3,649,756	128,885	$2,463,779	83,201	$1,522,005	72,603	$1,282,873
Flowers	1,854	52,680	1,332	39,133	572	16,037	1,099	29,565
Refined	160	3,746	163	4,373	79	19,960	709	17,928
Precipitated	29	3,508	41	5,403	27	3,352	28	3,224
Total	190,931	$3,709,690	130,421	$2,512,688	84,579	$1,561,354	74,439	$1,333,590

The price of first-grade Louisiana sulphur at New York, Boston and Portland fell from $22.15, during the first seven months of the year, to $20 for the remaining five months. The price at Philadelphia or Baltimore ranged about 50c. per long ton higher. This difference is due only to the fact that Sicilian sulphur costs more to deliver at Baltimore and Philadel-

phia than at the first mentioned cities, and the Union Sulphur Company sees no object in quoting a different price. Roll sulphur sold for $2 per 100 lb.; flour sulphur at $2.20; and flowers at $2.40 per 100 lb. during the year.

No noteworthy change occurred in the consumption of sulphur. It is still consumed chiefly in the wood-pulp business, but even therein is meeting with increased competition by pyrites. It has been estimated by F. J. Falding that the use of suitable pyrites for wood-pulp manufacture in New England would lead to an economy of 75c. to $1 per ton of pulp, according to whether the plant is on the seaboard or inland. It is also stated that sulphur will not be used more largely in the manufacture of sulphuric acid until it is reduced in price to $12 @ 14 per ton, according to the location of the works.

Toward the end of 1906, the Union Sulphur Company had great difficulty in shipping its product. All deliveries along the Atlantic seaboard are made by boat from New Orleans, to which point the sulphur has to be transported by the Southern Pacific Railway; shipments in other directions are made by all-rail routes. The shortage of cars, which was so prevalent, as well as of sea-going barges, made it impossible for the company to insure prompt delivery.

Sulphur in the United States.

Colorado.—In Gunnison county, near the town of Vulcan, there are sulphur deposits owned by the Vulcan-Good Hope-Mammoth Company and it is reported that they are to be exploited again in 1907. The mineral is said to run 75 per cent. sulphur.

Louisiana.—The sole operator in this State is the Union Sulphur Company, which produces upward of 750 tons of sulphur per day by the Frasch process, whereby hot water under pressure is forced through pipes sunk through the earth to the sulphur deposit, melting the sulphur which, in a liquid form, is forced up to the surface by compressed air. This method of mining has been fully described in previous volumes of The Mineral Industry. The company now owns 4500 acres of land in one tract on which there are 41 wells, each one having pierced the sulphur deposit.

Utah.—The Utah Sulphur Company of Salt Lake City owns a deposit at Sulphurdale, a small mining camp about 20 miles north of Beaver. The deposits are locally known as the Cove Creek sulphur beds and have supplied local markets for about 30 years. An area of several acres has been exploited, but the lateral extent and the depth of the deposit is unknown. Some of the sulphur occurs in cylindrical masses 10 or 15 ft. in diameter and having a rude radial structure, as if they had been formed about a central vent extending indefinitely downward into beds of tuff. The min-

eral varies in richness from a trace to practically pure sulphur and is refined in iron retorts wherein the sulphur is melted out by heat. The product from the retort contains 99.71 per cent. sulphur.

Wyoming.—L. W. Trumbull has described[1] the only deposit of sulphur now being developed in Wyoming, which is on the Stinking Water river, Big Horn county, three miles above Cody. The sulphur here occurs as pockets or bunches, in some places widening out so that it appears bedded. The sulphur-bearing area is, roughly, a mile square, and is recognized by the heavy deposit of calcareous tufa forming a broad mesa upon either side of the river, 100 to 200 ft. above the river level. This deposit is now being worked by the Big Horn Sulphur Company, of Cody. During 1906 this company erected a mill, and is now placing pulverized brimstone upon the market. At present all the mineral is taken from open cuts. Drilling is done by hand. The crude mineral is carried to the mill in tram cars, and is emptied at the mill into receiving bins, from which it is drawn as needed direct into the retort cars. These cars are built of heavy sheet steel, the side doors being hinged at the top. To permit the easy dumping of the clinker, after the sulphur is extracted, an inverted V reaches from the bottom nearly to the top of the car, thus practically making two long pockets. The sheet metal is perforated to permit of easy access of the steam and egress of the molten sulphur. Instead of being rectangular in cross-section, the cars are cylindrical and made to clear the inside of the 4.5 ft. steel cylinder used as a retort. This retort is a cylindrical steel shell 16 ft. long and 4.5 ft. inside diameter. This shell is placed horizontally with a casting in the bottom to serve as track for the retort cars to run upon.

The molten sulphur runs from the rear end of the retort through troughs to bins constructed of rough 1-in. lumber. After a bin has been filled enough to make a block of the desired hight (about 6 ft.) the boards are knocked off and the sulphur broken up to go to the crusher. The breaking up of this mass is done by means of picks and bars quite readily, as the block is formed in thin layers which break apart much more readily than does a homogeneous mass. When broken fine enough the sulphur is fed to a small jaw crusher, from which it goes to the pulverizer where it is reduced to 100-mesh. This product is marketed as flowers of sulphur. Although not so pure as the sublimed flowers of sulphur, it is pure enough for most purposes, being 99 per cent. fine, or a little more. The mill has a capacity of about 10 tons per day, but is so built that its capacity can be readily doubled in case the demand warrants. It is not expected that this sulphur will be shipped to points outside the mountain States, as the local demand will probably be sufficient. The principal uses for sulphur in the Western States are the making of sheep dip and fruit-tree spray.

[1] *Mines and Minerals*, Feb., 1907

Sulphur in Sicily.

The sulphur industry in Sicily underwent a complete change in conditions during 1906. Under a law passed by the Italian Parliament, the organization of the *Consorzio Obbligatorio*, which had been under discussion for some time, was completed, and the entire sulphur industry was placed in its hands. The *Consorzio* has full powers, and the sulphur producers are required to sell their output to it at prices to be fixed at regular intervals. No sales can be made and no sulphur exported without its permission. At the outset considerable difficulties were found in carrying out the plans. Producers were not satisfied and there was much controversy over the rival claims of Catania and Girgenti to be centers of the operations of the trust. Strikes and riots were incidents of the introduction of the new régime, and were so serious that military intervention was required. A partial settlement was effected, however, though matters are not yet running smoothly enough to prevent dissatisfaction.

The most important incident in connection with the change was the passing of the Anglo-Sicilian Sulphur Company. This company was organized about 10 years ago, and as a trust, or combination, succeeded in controlling the trade, stopping the sharp competition which had previously existed, and maintaining a high level of prices. Under the *Consorzio* the company necessarily ceased to be a factor in the trade, and was forced out of business. In regulating production and sales it had accumulated large stocks—over 400,000 tons—of brimstone, and these it was forced to sell to the *Consorzio.* It was estimated that the accumulated stocks had cost the Anglo-Sicilian company not far from $6,000,000 and that the loss incurred was not far from $1,500,000. The terms finally agreed upon were that the *Consorzio* purchase the stocks at the rate of 59 lire per ton; payment to be made in bonds guaranteed by the Italian Government, bearing interest at the rate of 3⅝ per cent., and redeemable in 12 annual drawings. In this way the shareholders in the Anglo-Sicilian company will eventually receive back practically the whole of the nominal capital.

The primary cause of all these changes was the appearance on the market of Louisiana sulphur in large quantities. For a long time the Italians were skeptical with regard to the possibility of American sulphur, and doubted the claims of the Union Sulphur Company, exploiting the Frasch process; but as their sales to America dropped year by year, dwindling to an insignificant amount in 1905, they could no longer doubt the seriousness of the new competition, especially in view of the reports of the technical men, including Signor Baldacci, whom they sent to the United States to investigate the conditions. Even then it was necessary, before they could be fully convinced, for the American producer to threaten competition in foreign markets.

After the organization of the *Consorzio* it was understood that negotiations had been opened with the Union Sulphur Company, the American producer, for a division of business and the prevention of competition. Whatever the nature of the proposed agreements, the negotiations during 1906 were not successful. Since the close of the year they have been renewed. In April, 1907, the *Consorzio* stopped all exports of sulphur to the United States; and in May, Mr. Frasch, as representative of the Union Company, started for Europe. At the time of writing, however, nothing definite is known as to the state of the negotiations; but it appears possible that some agreement may be reached.

TOTAL EXPORTS OF SULPHUR FROM SICILY, 1900–1906. (*a*)
(In tons of 1030 kg.)

Country.	1900	1901	1902	1903	1904	1905	1906
Austria	21,594	18,842	19,086	17,926	23,374	25,111	22,756
Belgium	9,721	7,471	12,323	15,233	13,627	14,442	13,940
France	103,647	74,394	67,249	74,372	103,040	96,170	67,536
Germany	28,702	23,448	25,906	32,553	31,613	28,319	34,967
Greece and Turkey	19,647	21,702	20,548	22,133	25,376	25,069	26,560
Holland	18,595	10,848	8,648	5,157	8,122	4,425	5,539
Italy	101,073	74,516	45,603	45,572	79,619	99,633	79,519
Portugal	10,937	11,335	10,614	14,064	8,373	13,196	12,302
Spain	6,187	2,979	2,249	4,099	4,063	2,478	3,120
Scandinavia (*c*)	22,681	24,486	24,918	28,292	20,120	18,288	21,608
Russia	22,090	15,110	17,295	15,068	15,141	16,673	16,181
United Kingdom	23,973	22,468	25,477	19,210	18,108	18,847	20,883
United States	162,505	144,817	168,919	155,996	100,000	70,332	41,283
Other Countries (*b*)	6,810	9,484	18,484	25,833	25,167	23,277	21,238
Totals	558,162	462,299	467,319	475,508	475,745	456,260	387,432
Stock in Sicily at end of year	221,204	302,410	339,113	361,220	396,541	462,437	525,115

(*a*) In 1900 and 1901 by A. S. Malcolmson, New York; for following years, by Emil Fog & Sons, Messina. (*b*) Mainly South Africa, Northern Africa, Australia and the East Indies. (*c*) Including Norway, Sweden and Denmark.

So far as the Anglo-Sicilian Sulphur Company is concerned, it is out of business, and its liquidation involves considerable losses for those who

SHIPMENTS OF SULPHUR FROM SICILY TO THE UNITED STATES.
(In tons of 1030 kg.)

Port.	1901		1902		1903		1904		1905		1906	
	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.
New York	72,104	19,631	76,383	26,842	70,800	21,201	41,429	10,547	26,782	16,270	18,106	13,009
Philadelphia	2,300	9,595	3,500	10,399	4,910	8,500	1,325	5,825	800	1,848	554	200
Baltimore	7,550	2,900	9,065	2,400	10,900	2,000	3,370	1,400	.....	....	21	....
Boston	3,497	3,200	2,204	2,300	5,508	2,450	11,397	1,749	5,477	1,009	959	1,750
Portland, Me.	21,990	.....	26,328	.....	23,855	.....	23,638	.....	17,882	.....	4,993	....
Other ports(*a*)	650	1,400	8,498	1,000	5,872	.....	.....	.....	....	.....	1,419	272
Totals	108,091	36,726	125,978	42,941	121,845	34,151	81,159	19,521	50,941	19,127	26,052	15,231

(*a*) Norfolk, Mobile, New Orleans, Savannah, San Francisco, Bangor, Portland, Ore., St. Louis and Canada.

invested in its stock. Although the management and the methods of finance of the company were sound and conservative in all other respects, the directors made a serious mistake in sinking their profits in accumulating

stocks of sulphur. As things turned out, they did not receive for these stocks more than three-quarters of what they paid for them. Although their policy was formulated at a time when there appeared to be no possible outside competition from America or elsewhere, it was a gross error of judgment to ignore the possibility of such competition, especially in view of the fact that the Louisiana deposits were known to be large even before the Anglo-Sicilian Company was organized. Evidently the repeated failures in the early attempts to work them clouded the judgment of the directors of the English company, and when it was announced that the Frasch process was a success they continued to disregard the future possibilities.

The whole affair is one of the most remarkable occurrences in industrial history, and in so far as the Anglo-Sicilian company is concerned, it might be cited as the sad fate of a trust; at least it is instructive in showing how an apparently secure trust may meet with disaster. American consumers have had to pay, since 1896, an advance of $3 to $7 per ton on their sulphur; but while it may be no consolation to them, at all events they will be interested to know that the trust, which has paid dividends to be sure, did not finally bag all of the profit. The extraordinary developments in connection with this business are by no means at an end, inasmuch as the entrance of the Italian Government into a compact by which it undertakes to regulate the output of private producers introduces into industrial economics an experiment which will be watched with interest. In effect it becomes the principal in dealing with the Union Sulphur Company, the middleman having been eliminated.

The future of the industry apparently rests on the making of some agreement between the Sicilian and the American producers. The *Consorzio* evidently intends to maintain high prices as far as possible. Should such agreements be made as will eliminate competition in the brimstone market, the only regulation of prices will be found in the increased production and use of pyrites.

Pyrites.

The increase in the demand for pyrites stimulated the development of new deposits and the reopening of some old mines, but the results of this work did not show up to any great extent in the total production in 1906, which increased only slightly. In Canada developments are in progress which will open some large veins of high-grade mineral.

Markets.—The year 1906 opened quiet, with prices at 10c. per unit for Spanish lump ore washed, 48 to 52 per cent. sulphur, and 10½c. for domestic furnace-size, averaging 45 per cent. sulphur per unit, all f.o.b. shipping ports. In June the market showed decided activity, which continued throughout the year. The 1907 output of the domestic mines was all sold

up before the close of the year. Prices began to move upward in June, when lump and furnace size, Spanish and domestic, were advanced ½c. per unit, and toward the close of the year most of the shippers demanded an additional advance of ½c. per unit.

PRODUCTION, IMPORTS AND CONSUMPTION OF PYRITE IN THE UNITED STATES. (a)
(In tons of 2240 lb.)

Year.	Production.		Imports. (b)		Consumption.	
1897	133,368	$404,699	259,546	$847,419	392,914	$1,252,118
1898	191,160	589,329	171,879	544,165	363,039	1,133,494
1899	178,408	583,323	310,008	1,074,855	488,416	1,658,178
1900	201,317	684,478	322,484	1,055,121	523,801	1,739,599
1901	234,825	1,024,449	403,706	1,415,149	638,531	2,439,598
1902	228,198	971,796	440,363	1,650,852	668,561	1,622,648
1903	199,387	787,579	425,989	1,628,600	625,376	2,416,179
1904	173,221	669,124	413,585	1,533,564	586,806	2,202,688
1905	224,980	752,936	515,722	1,780,800	740,702	2,533,736
1906	225,045	767,866	598,078	2,148,559	823,123	2,916,425

(a) These statistics do not include the auriferous pyrite used for the manufacture of sulphuric acid in Colorado. (b) Net imports, less re-exports of 3060 tons in 1902 and 1330 tons in 1903.

WORLD'S PRODUCTION OF PYRITES.
(In metric tons.)

Year.	Belgium.	Canada.	England.	France.	Germany.	Hungary.
1896...	2,560	30,580	10,177	282,064	129,168	52,697
1897...	1,828	35,291	10,752	303,488	133,302	44,454
1898...	147	29,223	12,302	310,972	136,849	58,079
1899...	283	25,112	12,426	318,832	144,623	79,519
1900...	400	36,308	12,484	305,073	169,447	87,000
1901...	560	31,982	10,405	307,447	157,433	93,907
1902...	710	32,304	9,315	318,235	165,225	106,490
1903...	720	30,822	9,794	322,118	170,867	96,619
1904...	1,075	29,980	10,452	271,454	174,782	97,148
1905...	976	29,713	12,381	265,000	185,368	106,848
1906...	(b)	35,927	11,318	(b)	196,571	(b)

Year.	Italy. (a)	Japan.	Norway. (c)	Portugal. (d)	Spain.	United States.
1896...	45,728	(b)	60,507	207,440	100,000	111,031
1897...	58,320	7,626	94,484	276,738	100,000	133,502
1898...	67,191	8,726	89,763	302,686	70,265	194,219
1899...	76,538	8,376	95,636	347,234	107,386	181,263
1900...	71,616	16,166	98,945	402,870	34,638	204,538
1901...	89,376	17,589	101,894	443,397	33,953	238,582
1902...	93,177	18,580	121,247	413,714	145,173	231,849
1903...	101,455	16,149	129,939	376,177	155,739	202,577
1904...	112,004	(b)	133 603	463,731	161,841	175,992
1905...	167,667	(b)	162,012	(b)	179,079	228,580
1906...	(b)	(b)	(b)	(b)	(b)	228,646

(a) Cupriferous in part. (b) Reports not yet available. (c) Both iron and copper pyrites. (d) Copper pyrite.

Alabama.—In recent years a growing industry has materialized in the mining of pyrite. The shipping mines are confined to a small area in Clay county, although it is known that there are other deposits that could be brought into use. The industry now centers around Pyriton. A branch of the Louisville & Nashville railroad affords transportation facilities from Stockdale. Some of the pyrites contain copper which can be econom-

ically extracted after the mineral has been burned. The copper content appears to increase with the depth of the deposits.

California.—The output of this State was curtailed in 1906 through the shutting down of the plant of the Realty Syndicate. This was done to enable the company to construct sulphuric acid works at Melrose. Upon its completion, which is expected to be in February, 1907, the handling of the pyrites will be resumed.

New York.—The increasing demand for pyrite and the advance in price for the product in New York created a great deal of activity in the industry in St. Lawrence county and considerable prospecting went on during 1906 in the vicinity of Gouveneur and Hermon, but no new discoveries were opened up to any appreciable extent. The old Stella mine, located about 12 mines from Gouverneur was reopened by the St. Lawrence Pyrites Company. A new up-to-date concentrating plant was erected, going into operation in 1907. Extensive tracts of land and mining rights have been secured by this company, and prospects for future production are promising. The Adirondack mine at Gouverneur was not active during 1906 because of difficulties among the stockholders. Amasa Corbin, a former president of the company, finally acquired the entire property and had the mine pumped out preparatory to renewing operations.

Virginia.—This State is the largest producer of pyrites in this country. In 1906 operations were carried out on a very extensive scale at the mine near Mineral, Louisa county, and near Dumfries, in Prince William county. The mines operating in 1906 in this State were the Arminius, those north of Mineral, in Louisa county, and the Cabin Branch mine near Dumfries, in Prince William county. A new producer is the Austin Run Mining Company, which is operating a deposit between the Cabin Branch and the Arminius mines. Shipments will begin in the spring of 1907. The ore is stated to run 48 to 52 per cent. sulphur and to be free from arsenic and phosphorus.

Production and Consumption of Sulphuric Acid in the United States.

By W. R. Ingalls.

In this article all references to tons mean 2000 lb., unless the long ton (2240 lb.) is expressly stated. Sulphuric acid is chiefly made in three degrees of strength, viz. 50, 60 and 66 deg. B., these being dilutions of pure sulphuric acid with different proportions of water.

Acid of 66°, U. S. Beaumé scale contains	93.500%	H_2SO_4
" " 60°, " " " " "	77.605%	"
" " 50°, " " " " "	62.177%	"

1 part of 66° acid	=1.20	parts of 60° acid
1 " " 60° "	=1.25	" " 50° "
1 " " 50° "	=0.80	" " 60° "
1 " " 60° "	=0.83	" " 66° "
1 " " 66° "	=1.50	" " 50° "

Production.

The only official statistics of sulphuric acid production in the United States are those of the Census. The reports for 1890, 1900, 1904, are as follows:

Grade	1890 (a)	1900 (b)	1904 (c)
			(d)
50° B........	504,932 tons	953,439 tons	....
60° B........	10,190 "	17,012 "	30,658
66° B........	177,267 "	377,279 "	325,582
Total......	692,389 "	1,347,730 "	(d)

(a) 105 works. (b) 127 works. (c) Statistics reported in Census for 1905. (d) Not reported.

Previous to the Eleventh Census (1890) the production was not reported according to grades, wherefore the figures are without commercial value.

The acid production of the United States in 1900, as reported by the Census, was equivalent to 1,540,623 tons of 50 deg. B; or 1,232,500 tons of 60 deg. B; or 1,022,975 tons of 66 deg. B; or 956,482 tons of pure acid. The Census figures do not appear to include the acid made by zinc smelters, and as a by-product in the treatment of gold and silver-bearing pyrites at certain metallurgical works in the West, the quantity of which in 1900 was reported by THE MINERAL INDUSTRY as 85,000 tons, basis 66 deg. B; equivalent to 102,000 tons of 60 deg. B, or 127,500 tons of 50 deg. B.

No statistics of sulphuric acid production since 1900 have been compiled, except the Census report for 1905. It is difficult to obtain these figures by direct reports from the producers, and the only available method of approximating the annual acid production is by computation from the quantity of brimstone and pyrites produced, imported and used, with allowance for the by-product acid. Unfortunately this method is rendered difficult by the consumption of brimstone for other purposes than sulphuric acid, and the absence of any statistics as to stocks of brimstone and pyrites, whereby the actual consumption could be determined.

The production of domestic pyrites in 1900 was 229,169 short tons. The imports of foreign pyrites were 361,182 short tons. Disregarding a possible difference in stocks between beginning and end of the year, the consumption of pyrites in the United States in 1900 may be estimated at 590,351 short tons, practically all of which was used for the manufacture of sulphuric acid. According to the statistics of the Twelfth Census the average product of 39 works burning pyrites was 206 parts of 50 deg. B acid per 100 parts of pyrites burned. The make of acid from pyrites was therefore probably about $590{,}351 \times 2.06 = 1{,}216{,}123$ tons of 50 deg. B $= 972{,}898$ tons of 60 deg. B = 807,506 tons of 66 deg. B.

The domestic production of brimstone in 1900 was 3525 short tons; the imports were 186,844 short tons; approximate consumption, 190,369 short tons. These figures refer to crude brimstone, which may be assumed to average 98 per cent. sulphur.

According to the Census statistics, the production of refined sulphur used in the manufacture of gunpowder, blasting powder, and lucifer matches, and for wool dipping, etc., was 13,000 tons, which would correspond to 13,266 tons of brimstone. Deducting this from the total consumption would leave 177,103 tons for the manufacture of sulphuric and sulphurous acids.

Total brimstone consumption	190,369 tons
Refined for various uses	13,266 "
Used for acids	177,103 "

The total production of sulphuric acid, basis 50 deg. B, from brimstone and pyrites was 1,540,623 tons. The make from pyrites estimated above, was 1,216,123 tons. This leaves 324,500 tons to come from brimstone.

According to the Census statistics, the average product of 20 works burning brimstone was 402 parts of acid of 50 deg. B per 100 parts of brimstone burned. It may be estimated roughly that the manufacture of one ton of 50 deg. B acid required 0.25 ton of brimstone, and the manufacture of 324,500 tons of acid required 81,125 tons of brimstone. There was, therefore, 177,103—81,125 = 95,978 tons of brimstone available for the manufacture of sulphurous acid, used in making sulphite pulp in the paper trade.

The manufacturers of sulphite pulp in the United States have not yet employed pyrites, except experimentally.[1] It is highly probable that they will begin to do so in the near future, but until they do, the production of sulphuric acid may be closely approximated by computation, as described above, from the statistics of consumption of brimstone and pyrites.

1901: The domestic consumption of pyrites was 715,155 tons; of brimstone, 203,865 tons. The production of pyrites acid must therefore have been $715{,}155 \times 2.06 = 1{,}473{,}219$ tons, basis 50 deg. Estimating a 10 per cent. increase in the consumption of brimstone for the manufacture of refined sulphur, and a 20 per cent. increase in the consumption for sulphite pulp manufacture, the consumption of brimstone for those purposes would have been $14{,}592 + 115{,}174 = 129{,}766$ tons, leaving 74,039 tons for the manufacture of sulphuric acid, the quantity of which would have been $4 \times 74{,}039 = 296{,}156$ tons, basis 50 deg. B.

1902: The domestic consumption of pyrites was 748,788 tons; of brimstone, 205,118 tons. The production of pyrites acid must therefore have

[1] This report was written in 1905.

been 748,788×2.06=1,542,503 tons, basis 50 deg. B. Estimating the same increase in the consumption of brimstone for refined sulphur as in the previous year, viz. 1326 tons, and 10 per cent. increase in the consumption for paper making, the total for those purposes would have been 15,916+126,691=142,609, leaving 62,509 tons for sulphuric acid, the quantity of which would have been 4 × 62,509 = 250,036 tons, basis 50 deg. B.

There were in operation in 1902 sulphite mills with aggregate capacity for making 2750 tons of pulp per day, or 2750×300=825,000 tons per annum. The manufacture of a ton of pulp requires an average of 0.15 ton of sulphur. On this basis, their consumption of sulphur would have been 123,750 tons; equivalent to 126,275 tons of brimstone. It is known that the sulphite mills were running full time in 1902. This estimate may be considered, therefore, to be a satisfactory check on the previous estimate, the figures being 126,275 vs. 126,691 tons.

1903: The domestic consumption of pyrites was 701,911 tons; of brimstone, 256,870 tons. The production of pyrites acid must therefore have been 701,911×2.06=1,445,937 tons, basis 50 deg. B. Estimating the same increase in the consumption of brimstone for refined sulphur as in the previous year, viz. 1326 tons, and 15 per cent. increase in the consumption for paper making, the total for those purposes would have been 17,244+145,694=162,936, leaving 93,932 tons for sulphuric acid, the quantity of which would have been 4×93,942=375,728, tons, basis 50 deg. B.

It is doubtful if there was really such an increase in the production of brimstone acid in 1903 as the statistics would indicate. The increased production of brimstone in the United States was due to Louisiana, the output of which became important toward the end of the year and probably went to a large extent into stocks which were consumed in 1904.

The increase in the manufacture of sulphite pulp is known to have been large in 1903 and probably is fairly stated at 15 per cent. over the previous year.

The statistics of pyrites and brimstone hereinbefore quoted are as stated by THE MINERAL INDUSTRY, except those for 1900, which are reported by the U. S. Geological Survey. The *Engineering and Mining Journal* reported the brimstone production of the United States as 39,310 long tons in 1903, and 191,250 long tons in 1904, the production in each year being chiefly from Louisiana. These statistics were not based on direct reports from the Union Sulphur Company, but were estimates based on substantial data, and were considered to be close approximations to the actual production.

The statistics of consumption of pyrites and brimstone in the United States for 1900-1903 are presented in the accompanying table:

CONSUMPTION OF SULPHUR IN THE UNITED STATES.

Pyrites.				
Year.	1900	1901	1902	1903
Production.........	204,615	234,825	228,198	199,387
Imports...........	322,484	403,706	440,363	427,319
Total long tons.....	527,099	638,531	668,561	626,706
Total short tons....	590,351	715,155	748,788	701,911
Brimstone.				
Year.	1900	1901	1902	1903
Production.........	3,147	6,866	7,443	39,310
Imports...........	166,825	175,310	176,951	191,005
Total............	169,972	182,176	184,394	230,315
Exports...........	540	207	1,253	967
Consumption, long tons........	169,432	181,969	183,141	229,348
Consumption, short tons........	189,764	203,805	205,118	256,870

PRODUCTION OF SULPHURIC ACID IN THE UNITED STATES.
Distributed according to source and computed on basis of 50 deg. B. Tons of 2000 pounds.

Year.	Pyrites.	Brimstone.	Blende	Other.	Total.
1900....	1,216,123	324,500	75,000	52,500	1,668,123
1901....	1,473,219	296,156	80,000	67,000	1,916,539
1902....	1,542,503	250,036	110,000	70,000	1,972,539
1903....	1,445,937	375,728	125,000	75,000	2,021,665

The chief sources of error in the above statistics are: (1) The uncertainty as to the actual consumption of brimstone for other purposes than acid making. (2) The assumption that production plus imports, minus exports represents the consumption of brimstone and pyrites, differences in stocks being disregarded. (3) The increase in yield of acid per ton of pyrites or brimstone with improvements in plants, but this is evidently a comparatively unimportant source of error.

Of the 127 works in operation in 1900, the number burning brimstone only was 31, pyrites only 79, while 17 used both brimstone and pyrites. The average product of 20 works burning brimstone was 402 parts of 50 deg. B acid per 100 parts of brimstone burned. The average product of 39 works burning pyrites was 206 parts of 50 deg. B acid per 100 parts of pyrites burned. One part of sulphur, burned, should theoretically make 3.0625 parts of pure acid, or 4.6 parts of acid of 50 deg. B. An efficiency in the process of 98 per cent. is attained in the best practice. In the burning of 97 units of sulphur in brimstone, with 98 per cent. recovery in the

chamber process, the yield should be 4.37 parts of acid of 50 deg. B; in the burning of 45 units of sulphur in pyrites, the yield of 50 deg. B. acid should be 2.05 parts. The Census figures show therefore a rather high efficiency of the brimstone plants.

As a new development in the sulphuric acid business, the Tennessee Copper Company has recently erected a plant for making acid from the gases of its smelting furnaces at Copperhill, Tenn., which are now operated on the pyritic principle. It has been determined that these gases average about 6 per cent. in sulphur dioxide, and when the plant is in full operation their quantity will enable the production of upward of 700,000 tons of sulphuric acid of 50 deg. B per annum.

Consumption.

The total production of sulphuric acid in 1900, reduced to its content of the pure acid, was 1,034,659 tons. The principal uses of this acid were as follows:

Manufacture of Superphosphate.—The average of the entire country showed in 1900 a consumption of 2000 lb. of acid of 50 deg. B per 2000 lb. of phosphate rock decomposed. The consumption of phosphate rock being reported annually in the "Mineral Resources of the United States," the annual consumption of acid for this purpose can be closely approximated. The consumption in 1900 was 815,855 tons of 50 deg. B, or 507,462 tons of pure acid, according to the Census.

Nitric Acid.—The production of 31,237 tons of nitric acid required 47,348 tons of sulphuric of 60 deg. B, or 36,742 tons of pure acid. (Census.)

Muriatic Acid.—The production of 58,424 tons of muriatic acid required 39,000 tons of sulphuric of 60 deg. B, or 30,264 tons of pure acid. (Census.)

Mixed Acids.—The production was 31,635 tons, which may be estimated as 60 per cent. pure sulphuric, or 18,981 tons. (Census.)

Alum and Aluminum Sulphate.—The consumption of acid for these purposes was 61,424 tons of 60 deg. B, or 47,665 tons of pure acid. (Census.)

Copper Sulphate.—The production, according to THE MINERAL INDUSTRY, was 39,109 tons, or 24,834 tons of anhydrous $CuSO_4$, requiring 15,644 tons of sulphuric acid. In making one ton of blue vitriol, 0.4 ton of pure sulphuric acid is required.

Ammonium Sulphate. The production, according to THE MINERAL INDUSTRY, was 63,800 tons, which would have required 47,850 tons of pure acid, one ton of the sulphate requiring about 0.75 ton of pure acid.

Copperas.—The production of copperas as an original product was 14,866 tons, requiring 5203 tons of pure sulphuric acid, one ton of copperas requiring 0.35 ton of pure acid. (Census.)

Chrome Products.—The consumption of chrome ore was 19,647 short tons. Deducting 2500 tons for the manufacture of brick, etc., the amount

available for the manufacture of bichromates, etc., was 17,147 tons which would have required 6859 tons of pure acid.

Epsom Salt.—The production of magnesium sulphate (epsom salt) was 4620 tons, requiring 1848 tons of pure acid, one ton of epsom salt requiring 0.4 ton of acid. (Census.)

Summary.—The principal uses of acid in 1900, reduced to the basis of pure acid, were as follows, in tons:

1. Superphosphate	507,462
2. Petroleum refining	196,141
3. Pickling iron and steel	70,000
4. Nitric acid	36,742
5. Muriatic acid	30,264
6. Mixed acids	18,981
7. Alum and aluminum sulphate	47,665
8. Blue vitriol	15,644
9. Ammonium sulphate	47,850
10. Copperas	5,203
11. Chrome products	6,859
12. Epsom salt	1,848
13. Other purposes	50,000
Total	1,034,659

The above statistics are presented only as a very rough approximation. They indicate that about 68 per cent. of all the acid is used in the manufacture of fertilizers and in petroleum refining; about 7 per cent. in pickling iron and steel preparatory to galvanizing and tinning; about 8 per cent. for the manufacture of other acids; about 12 per cent. in the manufacture of six heavy chemical products; and about 5 per cent. for miscellaneous purposes, including the manufacture of carbonic acid by decomposition of marble dust, the manufacture of glucose, bleaching barytes, manufacture of blanc fixe, zinc sulphate, boric acid, and numerous other minor chemicals.

I arrive at the consumption by taking the figures for items 1, 4, 5, 6, 7, 8, 9, 10, 11, and 12 as previously computed, adding to them an estimate based on private information for item 3, and an estimate for item 13 which is purely a surmise, the difference between this total and the reported figure for acid production giving what was presumably used in refining petroleum. According to Samuel P. Sadtler, in THE MINERAL INDUSTRY Vol. IV, p. 505, the whole charge of acid needed for an average distillate is about 6 lb. to the barrel of oil. Assuming that the entire production of petroleum, 63,620,529 bbl. in 1900, were refined, which was not the case, the consumption of acid on the above basis would have been about 190,000 tons (of 66 deg. B). However, we know that a part of the sludge acid is refined and reconcentrated for further use; and a part is sold for use in the manufacture of fertilizers.

It would be futile, for obvious reasons, to attempt an itemization of the sulphuric acid consumption year by year subsequent to 1900. A study of the statistics of consumption of certain materials in connection with which sulphuric acid is employed, however, will be indicative of the

acid consumption in some important channels. The available statistics, taken from the reports of the U. S. Geological Survey, except where otherwise noted, are given in the accompanying table:

CONSUMPTION OF CERTAIN MINERALS IN THE UNITED STATES.

Article.	1900	1901	1902	1903
A. Phosphate rock.	1,170,094	1,062,371	1,048,446	1,028,085
B. Petroleum......	63,620,529	69,389,194	88,766,916	100,461,337
C. Spelter..........	99,399	141,697	132,682	154,381
D. Tin............	35,000	37,280	42,520	41,566
E. Bauxite.........	34,540	40,564	48,285	70,533
F. Chrome ore......	19,647	22,525	44,318	25,682
G. Blue vitriol......	39,109	39,000	34,380	21,562
H. Ammonium Sulphate...........	63,933	66,138	71,649	....

The production of phosphate rock is from The Mineral Industry; consumption being computed by adding imports to production and deducting exports. Note discrepancy between these figures for 1900 and those of the Census. The imports of tin and production of blue vitriol and ammonium sulphate are also as reported by The Mineral Industry; the figures for ammonium sulphate are not, in my opinion, thoroughly to be relied upon. The production of petroleum is stated in barrels; of all other substances, in short tons.

A. In the acidification of phosphate rock about 2000 lb. of acid of 50 deg. B are required per 2000 lb. of rock, on the average.

B. In the refining of petroleum about 6 lb. of acid of 66 deg. B is required per barrel of petroleum refined.

C. About 50 per cent. of the domestic consumption of spelter is employed in the galvanizing of iron and steel. The surface of the iron or steel is necessarily first cleaned by use of sulphuric acid. The zinc being applied in a coating of not widely varying thickness, the quantity of spelter uses bears a roughly regular relation to the surface covered. The consumption of acid bearing a similar relation, there is an approximately regular proportion between the quantities of spelter and acid used. This is about 0.45 ton of acid of 60 deg. B. per ton of spelter.

D. The same notes apply as to spelter, but I am unable to state the average proportion between acid and tin consumption.

E. A portion of the bauxite used in the United States is for the production of aluminum and aluminum hydrate, but the major portion is employed in the manufacture of alum and aluminum sulphate, in which a large quantity of acid is required.

F. In 1900, about 2500 tons of chrome ore was employed for making brick, furnace linings, etc; the remainder for the manufacture of potassium bichromate, and similar products, in which sulphuric acid is required.

The consumption for chrome brick, etc., has probably increased largely since 1900.

G. The manufacture of one pound of blue vitriol requires about 0.4 lb. of pure sulphuric acid (100 per cent.).

H. The manufacture of ammonium sulphate requires about 0.75 ton of pure acid per ton of the sulphate. This is a large and growing use for sulphuric acid.

The statistics of mineral and chemical production presented above indicate that the increase in the production of acid from 1,668,123 tons in 1900 to 2,021,665 tons in 1903, basis 50 deg. B, has been absorbed chiefly in the refining of petroleum. There has also been an important increase in the consumption for pickling iron and steel, and in the manufacture of some chemical products, especially alum and aluminum sulphate, and ammonium sulphate. If the statistics were available, it would probably appear that there has been a generally increasing consumption for the manufacture of heavy chemicals and other acids. Evidently there has been no increase in the manufacture of fertilizers; which is rather surprising.

Consumers.—At the present time, the largest consumers of sulphuric acid in the United States are (1) the Virginia-Carolina Chemical Company; (2) American Agricultural Chemical Company; (3) Standard Oil Company; (4) U. S. Steel Corporation; (5) General Chemical Company; (6) Grasselli Chemical Company; (7) Pennsylvania Salt Mfg. Company. It would appear probable that those concerns consume upward of 75 per cent. of all the sulphuric acid produced in the United States.

TALC AND SOAPSTONE.

By Reginald Meeks.

During the latter half of 1906 supplies of good grades of white talc fell off considerably and there was a decided scarcity of available foreign or domestic grades. This was due largely to curtailment of production at the mines caused by floods and freshets which placed producers behind in their deliveries. The shortage of prime white, high-grade, domestic talc forced dealers to fill their orders with imported goods and considerable attention was paid to soapstone to supply the needs of purchasers. White talc for spot delivery received a handsome premium over nominal quotations throughout a large portion of the year.

New York continued to produce the larger portion of the talc in the United States. Other producing States were New Jersey, Pennsylvania, Virginia, North Carolina, Vermont, Massachusetts and California. Arkansas will, in 1907, resume production through the efforts of the Arkansas Soapstone and Refractories Manufacturing Company of Little Rock.

STATISTICS OF TALC AND SOAPSTONE IN THE UNITED STATES. (*a*)
(In tons of 2000 lb.)

Year.	Production						Imports.		
	Fibrous Talc.			Talc and Soapstone. (*b*)			Sh. Tons.	Value.	Value Per Ton.
	Sh. Tons.	Value.	Per Ton.	Sh. Tons.	Value.	Per Ton.			
1896	51,816	$256,080	$4.94	21,448	$207,085	$9.66	1,950	$18,693	$9.60
1897	52,836	283,685	5.37	27,068	259,948	9.60	779	8,423	10.54
1898	54,807	285,759	5.21	27,974	237,280	8.48	445	5,526	10.70
1899	57,120	272,595	4.77	26,682	241,267	9.04	254	3,534	13.91
1900	45,000	236,250	5.25	26,726	249,777	9.35	79	1,070	13.50
1901	69,200	483,600	6.99	28,643	424,888	14.83	2,386	27,015	11.32
1902	71 100	615,350	8.65	26,854	525,157	19.36	2,859	35,336	12.36
1903	60.230	421,600	7.00	26,671	418,460	15.69	1,791	19,677	10.99
1904	64,005	507,400	7.93	27,184	433,331	15.94	3,268	36,370	11.13
1905	67,000	519,250	7.75	40,134	637,062	15.87	4,000	48,225	12.06
1906	64,200	541,600	8.43	....	....	....	5,643	67,818	12.02

(*a*) Statistics for 1902 and subsequent years, are as reported by the United States Geological Survey. (*b*) The value of these products has not much significance owing to the diverse conditions of the material reported.

Georgia.—There are only two producing mines in this State, viz, the Cohutta Talc Company of Spring Place, and a new company, the Fort Mountain Talc Company, which is planning to build a factory at Chatsworth, Ga.

North Carolina.—The Croaton company is arranging to develop the Fair Promise talc mine, near Glendon, Moore county. A plant to prepare 50 tons per day for market is to be erected. F. M. Peck, of Hartford, Conn.,

is president, and J. C. Stockton is superintendent. Other companies organized in North Carolina during 1906 were as follows: North Carolina Talc and Manufacturing Company, Hewett; American Talc Company, Glendon; Nauthala Marble and Talc Company, Kinsey; Southern Mineral Company, Tomotla; Glendon Manufacturing and Mining Company, Glendon; and Cherokee Iron, Marble and Talc Company, Murphy. Most of these companies are either simply developing or else are only small producers.

Virginia.—Within recent years the soapstone industry in Virginia has assumed large proportions. During 1906 production of soapstone and talc was made in four counties, the principal operations being confined to Albemarle and Nelson. Beginning with the most southwesterly operation in Nelson county, the following companies were operating during 1906: The Piedmont Soapstone Company; the Phoenix Soapstone Company; the American Soapstone Company; the National Soapstone Company; the Plumbers Soapstone Company of America; the Virginia Soapstone Company; the Old Dominion Soapstone Corporation; and the Albemarle Soapstone Company. In Amelia county, about four miles north of Jetersville, a station on the Southern Railway, extensive developments were in progress by a Philadelphia corporation during the summer. In the vicinity of Herndon station, on the Southern Railway, in Fairfax county, a good grade of talc was mined.

Fibrous Talc in New York.

By D. H. Newland.

The St. Lawrence county mines reported an output in 1906 of 64,200 short tons, valued at $541,600. The yield was somewhat less than in 1905, when the total was 67,000 short tons, but the value showed a notable increase due to the higher range of prices. The average selling price for the year, on the basis of car load lots, was $8.43 per ton, as compared with $7.75 per ton in 1905. The annual production of talc in New York, and value for the period 1897–1906 are given in a preceding table.

A further consolidation of the mining and milling interests in the talc district was effected during 1906 by the International Pulp Company, the largest producer. Since its organization in 1893 this company has exercised a controlling influence over the industry, though it has not been without active competition from rival concerns. At the time it was formed the properties of the St. Lawrence Pulp Company, the Natural Dam Pulp Company, the Agalite Fibre Company and Adirondack Pulp Company were acquired and in the following year those of the Asbestos Pulp Company were also secured. The company thus came into possession of all of the then active interests except the United States Talc Company, which was taken over in August, 1906. The latter owned a mine west of Talcville and a mill near Dodgeville. The mill has been recently enlarged and will

partially restore the impairment of productive facilities with which the International company met during 1906 in the loss by fire of its Hailesboro mill. That mill was the largest and perhaps the best equipped of any in the district. It had a capacity of 75 tons per day. The company also owns three other mills, situated at Talcville near its mines.

The Union Talc Company and the Ontario Talc Company have been the only producers in recent years aside from those mentioned. It has been reported that the latter has also come under the control of the International Pulp Company through purchase of the entire capital stock. The company owns three mines and a mill near Fullerville in the central part of the district. The Union Talc Company has been the largest independent operator since 1900. It was organized as a consolidation of the Columbian Talc Company, the American Pulp Company, and Keller Bros., and came into possession of three mines and an equal number of mills. The American mill near Fowler owned by this company was burned in December, 1905.

A new company was incorporated late in 1906 to mine and manufacture talc in the St. Lawrence county district. According to present plans, this company proposes to work the Freeman mine which has been under lease to the International Pulp Company. Water power on the Oswegatchie river near Dodgeville has been secured and it is planned to erect a mill of 75 tons daily capacity.

Talc in Foreign Countries.

Canada.—All the talc produced in this country in 1906 came from the Henderson mine, near Madoc, Hastings county, Ontario. There were 1234 short tons shipped from this mine in 1906 to the United States. This constitutes the whole of the Canadian export of this mineral.

France.—France is an important market for talc, in block or in powder. Good white talc may be purchased at from \$9 to \$11 per metric ton, and a better quality is obtainable at \$15 per ton. It is used largely for burners of acetylene gas jets. A contract with an American firm was closed, during 1906, for 1000 tons at \$40 per ton for this and similar purposes. France may now be considered the first European talc country, having overtaken Italy, which was long the largest producer. The talc deposits of Luzech, France, are situated almost equally distant from Bordeaux and Marseilles. The deposits are large, of excellent quality, free from mica, and of a pure white or blueish-white color, much appreciated by consumers, wherefore the demand for this grade of talc is strongly increasing. The beds of talc are situated at the contact of the massive St. Barthelémy granites and the old schists, about 7 km. from Luzech, at an altitude of from 1500 to 1800 m. The deposits are operated as quarries. The thickness of the beds is variable, but quarry No. 3 has already been proved to a depth of 56 m. Four qualities of talc are produced from these quarries. The "extra qual-

ity" consists of blocks of steatite, averaging 200 to 250 cm. on each side. The "first quality" is small pieces of steatite which are perfectly white and almost translucent. This quality is pulverized and used in the pharmaceutical trade for so-called talcum powder. The "second quality" comprises pieces of talc more or less cloudy in color, being contaminated by organic matter or a little silicate or oxide or iron. The "third quality" is yellow, due to decomposition of adjacent pyrite beds; this grade is pasty and pulverulent and its use is limited.

Italy.—In the Italian Alps there are deposits of talc at Pinerolo near Turin, which are the oldest known. For a long time they have been the chief producers, but are now on the decline. These Italian mines formerly produced about 10,000 metric tons annually, but of late their production has fallen to about 7000 tons.

TANTALUM.

Tantalum has been known for a century, but was isolated in pure form only as recently as 1903, when Moissan reduced it in the electric furnace. His first samples were so impregnated with carbon as to mask the wonderful physical characters of the pure metal. The demand for the metal for filaments of a new incandescent electric lamp, known as the "tantalum lamp," and the later utilization of it for the manufacture of writing pens and other purposes, have directed special attention to its metallurgy and the investigation of its wonderful properties.

Pure tantalum was first obtained by Werner von Bolton, chief chemist for Siemens & Halske, of Berlin. Since tantalum has an affinity for nearly all known elements (being especially liable to react with them at high temperature), it occurred to Bolton to try the electric furnace in a vacuum. He found it impossible to use carbon electrodes, because of the liability of forming tantalum carbide, and therefore employed metal electrodes, and successfully, especially when made of tantalum.

The method of reduction consists in first forming the impure metal into rods, or other suitable form, by pressure or by pounding in a crucible of refractory material. Such a crucible may consist of oxide of magnesium or thorium; it is lined on the inner side with a conductive coating, as of metallic tantalum. The contents of the crucible are connected to one pole, preferably the anode of a continuous current. A rod of tantalum metal (of sufficient size to prevent its being melted in the arc) serves as the cathode. A large cathode may be made of silver; but if too small, it will melt.

The crucible containing impure tantalum is enclosed in a casing with the electrodes, and an air-pump is attached to produce a partial vacuum. The casing has a spy-hole of glass, and provision is made for shifting the position of the cathode without destroying the vacuum. In operation, the cathode is moved over the surface of the mass of impure tantalum, so that the electric arc passes successively through all parts of the mass, melting the metal and expelling the oxygen or oxide. In this manner large lumps of pure tantalum are obtained, homogeneous and free from blisters.

According to Ivar Nordenskjold, the melting point of pure tantalum is 2250-2300 deg. C.; specific heat 0.0365; atomic heat 6.64, agreeing well with Dulong and Petit's rule. Specific gravity of cast metal 16.64; this

is half as much again as the old values, which must have been determined from impure material. Coefficient of linear expansion 0.0000079, between 0 and 50 deg. Specific electrical resistance 0.165 ohm in a wire 1 m. (39.37 in.) long and having 1 sq. mm. (0.00155 sq. in.) area of cross section, with an average rise of 0.3 per cent. per degree of temperature between 0 and 100 deg. C., and 0.26 per cent. between 0 and 350 deg. Tensile strength of a wire 1 mm. (0.04 in.) in diameter, 93 kg. per sq. mm. (59 tons per sq. in.); elongation 1 to 2 per cent.; modulus of elasticity of a wire 0.08 mm. (0.003 in.) in diameter, 19,000 kg. per sq. mm. (12,000 tons per sq. in.). At a red heat tantalum can readily be hammered out in a sheet, which under repeated heatings and hammerings acquires a hardness but little below that of the diamond.

When being mechanically worked, the metal (in particular when it contains a small quantity of carbon or other hardening medium) readily assumes so great a degree of hardness that the further working is rendered impossible and it must then be carefully reheated or annealed in order to be rendered soft again. In this annealing process care must be taken that the temperature does not rise too high, as otherwise the material is more easily attacked by the oxygen of the atmosphere. In making pens from tantalum, the metal, having been smelted so as to obtain in the first instance a lump of irregular form, which is in a soft condition, is rolled into thin sheets, by which process the tantalum assumes a much greater degree of hardness than before and it may therefore require to be repeatedly annealed between the several stages of rolling. After the final stage of rolling, the thin sheets of tantalum may be found to be of the right hardness for forming the pens; if too hard they may be again annealed to a certain extent, of if the pens are required to have a greater degree of hardness than can be attained by merely working the metal, the tantalum may have a small quantity of a hardening substance, such as carbon, added to it. This may either be effected in the first instance when the metal is smelted, a very small quantity, such as 0.1 per cent. of carbon, being sufficient for the purpose, or the prepared sheets of tantalum metal may be packed in carbon powder and subjected to considerable heat for some time.

The minerals columbite and tantalite occur in the United States, being most abundant in the Black Hills, South Dakota. Other occurrences are known in Connecticut and North Carolina. Except for a small amount used in experimental work and in investigations, all of the limited production of tantalum minerals is shipped to Germany. The minerals are marketed on the basis of the content of tantalic acid ($Ta_2 O_5$). The limited market is now so greatly overstocked that it would be difficult to dispose of any supply of mineral at present. In 1905, about 70 tons of tantalite, valued at £8925, were exported from Australia.

TIN.

By W. R. Ingalls.

There was a small production of pig tin in the United States in 1906, made experimentally in the Black Hills. There was also a small production of tin ore in the Carolinas, the Black Hills and Alaska. However, tin mining in the United States continues to be insignificant. It is probable, as was remarked last year, that some of the deposits known to exist in the United States could be worked profitably as low-grade enterprises, if they were approached in the proper spirit and with broad plans for development, but owing to the numerous failures that are on record, capital is timid about undertaking the exploitation of these mines. Efforts were made in 1906 to interest some of the large mining syndicates in taking hold of the mines in the Carolinas, but these efforts were quite unsuccessful.

The statistics show a further decrease in the tin production of the Straits Settlements in 1906. On the other hand, there was a considerable revival of mining in Cornwall and Australia, stimulated by the high price for the metal, and the tin production of Bolivia forged largely ahead. The production of Tasmania showed a decrease, the Mount Bischoff mine evidently being near to exhaustion. The unfavorable reports from this company caused a great fall in the value of its shares.

THE PRINCIPAL TIN SUPPLIES OF THE WORLD. (a)

(In tons of 2240 lb.)

	1898	1899	1900	1901	1902	1903	1904	1905	1906
English production	4,684	4,013	4,268	4,566	4,392	4,282	4,132	4,468	4,920
Straits to Europe and America	43,350	44,460	46,058	50,339	51,831	52,212	57,566	56,847	57,138
Straits to India and China	2,551	1,484	1,785	2,655	1,882	3,123	3,261	1,700	1,300
Australia to Europe and America	2,420	3,337	3,235	3,345	3,199	4,934	4,846	5,028	6,888
Banka sales in Holland	9,038	9,066	12,631	14,978	14,978	15,070	11,363	9,960	9,300
Billiton sales in Java and Holland	5,342	5,057	5,882	4,387	3,897	3,650	3,215	2,715	1,950
Bolivian arrivals on Continent	1,000	813	1,900	9,670	10,150	9,790	11,867	12,500	14,700
Bolivian arrivals in England	3,464	3,940	5,065						
Totals in long tons	71,849	72,170	80,824	89,940	90,329	93,061	96,250	93,218	96,196
Totals in metric tons	72,999	73,325	82,117	91,379	91,774	94,550	97,790	94,709	97,735

(a) Compiled from commercial reports. In addition to the tin supplies noted in the above table there is an uncertain quantity exported from China. In 1905 and 1906 this appears to have amounted to 4000–5000 tons per annum. There is also a small production in Germany.

IMPORTS OF TIN INTO THE UNITED STATES.

Year.	Pounds.	Value.	Year.	Pounds.	Value.	Year.	Pounds.	Value.
1898...	62,748,399	$8,770,221	1901...	74,560,487	$19,024,761	1904...	83,168,657	$22,356,896
1899...	71,248,407	16,748,107	1902...	85,043,353	21,263,337	1905...	89,227,698	26,316,023
1900...	69,989,502	19,458,586	1903...	83,133,847	22,265,367	1906...	101,027,188	37,446,508

Tin Mining in the United States.

Alaska.—A small amount of tin concentrate was produced in Alaska but the mines of this region have still failed to materialize in importance, although considerable confidence is still displayed that eventually they will become producers of consequence. The United States-Alaska Tin Mining Company claims to have developed a well-defined lode of high grade tin ore, at Cape mountain, where a three-stamp mill was erected in 1906. Some development work has also been done on Ear mountain, on Lost river and near Potato mountain.

Carolinas.—Small quantities of tin ore were produced in 1906 by Capt. Samuel Ross, of Gaffney, S. C., L. U. Campbell, of Gaffney, S. C., and Capt. Foster, of the Blue Ridge Tin Company, King's Mountain, N. C. Their product was entirely placer tin, which was shipped to Cornwall and Hamburg. The product assayed from 66 to 70 per cent. tin. The freight rate to Cornwall was $5 per ton, while $40 per ton was charged for smelting.

South Dakota.—Some work was done in 1906 in the Gertie mine by E. C. Johnson, who smelted experimentally some small lots of black tin. Developments also were made in the Clara Belle mine near Oreville, where at a depth of 265 ft. a vein of tin ore 4 ft. wide was discovered. The ore was reported to assay about 2.5 per cent. metallic tin. The Tinton Tin Mining Company, of Tinton, did no work, operations having been suspended because of difficulties among the stockholders.

Tin Mining in Foreign Countries.

Australia. (By F. S. Mance.)—Operations on all the tinfields were characterized by great activity in 1906 and a largely augmented production is recorded. The mines of Tasmania furnished an output of 4473 tons of ore, valued at £557,266, during the year, which exceeds that for 1905 by 581 tons, and £194,596 in value. The Briseis and New Brothers Home No. 1 mines were favored with a plentiful supply of water, and were thus enabled to deal with large quantities of drift, the output of tin ore averaging 110 tons a month. The high price ruling for tin permitted the Mt. Bischoff company to conduct operations profitably on ore of very low grade, but during the month of September the value of the ore decreased to such an extent that the output of concentrates fell from 100 to 70 tons per month. Attention has accordingly been directed toward effecting further economies in working, and in securing improved recoveries by the tin-dressing plant. It is thus hoped that a monthly output of 60 tons of concentrate can still be maintained. Consideration is also to be given to the question of treatment of the available pyritic ores. During the last 30 years this mine has distributed £2,079,000 to shareholders, or at the rate or £173 5s. per share.

New South Wales also shows decidedly improved returns, the output for

the year being valued at £255,744, and exceeding that of any other year since 1888. The feature of the industry is the success which has attended the operations of the dredges in the Inverell-Tingha district, which were installed to treat the stanniferous gravels, mostly worked over by ordinary methods in former years. During 1906 there were 27 plants in operation and they secured 943 tons of ore valued at £110,582. The yield obtained in the month of September by the three principal dredges averaged 2.5 lb. of tin ore per cu. yd. of material treated, valued at 32.2d., which, after deducting all expenses, gave a profit of 22.6d. per cu. yd. Other plants are being erected on similar areas in the State, and there are indications that many years of profitable work are ahead.

The State of Queensland contributed tin ore to the value of £490,283 during 1906, which is £192,829 in excess of the value of the yield for 1905. The mines within the confines of what is known as the Walsh & Tinaroo field yielded most satisfactorily, and the developments at the lower levels have been especially encouraging. The principal contributing mines were the Vulcan, Stannary Hills, and Arbouin. At the 900-ft. level in the Vulcan mine an orebody was exposed which assays 10 per cent. tin. The dredges operating at Stanthorpe, on the New South Wales border, also secured very good returns.

The Greenbushes tin field of Western Australia produced 778 tons of ore in 1906, against 638 in 1905 and 534 in 1904. At Moolyella, also in Western Australia, about 400 men are working on alluvial tin deposits, and several lodes have also been located.

In Queensland the Smith's Creek Tin Company is reported to be developing a valuable tin mine. In the main shaft, at a depth of 30 ft., the vein is 6 ft. wide, and averages 4 per cent. tin. A new milling plant is being erected on the mine. The Stannary Hills tin mines continue to develop well. In the Ivanhoe mine the vein is said to be 15 ft. wide, and assaying 4 per cent. tin. The Herberton mine, situated seven miles north of Hot Springs, Queensland, is being actively developed. Both tin and wolfram mining are being actively carried on in this district. The Dalziel tin mine, also in Queensland, owned by an English company, is developing a large ore deposit.

The output of the Mt. Bischoff mine, Tasmania, dropped from 100 tons of tin ore per month to 70 tons, owing to exhaustion of the richer ore of the mine. This unexpected disclosure, which was attended by a great decline in the value of the shares, excited much adverse comment upon the management of the company. The situation at the mine is outlined in the following report by the general manager, H. W. F. Kayser, under date of Oct. 8, 1906: "After having thoroughly examined the different

workings, in accordance with instructions from the directors, I have come to the conclusion that it will be unwise, in fact almost impossible, to continue a high-fixed output in future. Speaking generally, the greater part of the crushing material is low in grade, and the higher grade ores still in sight have to be handled very judiciously to maintain anything like a regular output, and to extend the life of the mine. The No. 3 bench giving out so suddenly was unexpected, and the only explanation I can make is that the last slip from the higher part of the face forced all the good material out both east and west, leaving only the poor dirt behind. This, with the loss of the brown face, quickly affected the output, and, although there is still some good ore on the No. 3 bench in the solid face going west, and in all probability underfoot also at the north alluvial, and some nice patches here and there in other portions of the workings, I do not think it will be advisable to attempt to maintain, at least for the present, a higher output than about 60 tons per month. This may be increased if the underground works improve. So far, however, the deep ground has been disappointing.

"The pyritic ore on hand, and such surface deposits as are known to exist, may help matters to a certain extent as soon as we are ready to treat this material. With this in view, the erection of a 10-head battery will be started immediately at the old Happy Valley site, but unless new pyritic supplies are discovered the plant will not be kept constantly at work. The pyritic deposit met with in the rise from the main tunnel to the brown face may prove a valuable factor in this respect, and prospecting works are now under way to prove the extent of this formation. The porphyry dikes supply a large portion of the ore, but they are mostly low in grade, and we have to depend on better material to maintain the output; and, as the porphyry is considerably intermixed with pyrites of no value, which has to be picked out, the working expenses have increased by not being able to carry on blasting and stone-breaking work in a wholesale way. This is not the first time anxiety has been felt for the future, but matters have always righted themselves, and, although the mine is not young, there is still hope of improvement."

In other parts of the west coast of Tasmania there are signs of revival in tin mining. The Dundas district, where pyritic orebodies occur containing rich tin ore, is receiving more attention, and the alluvial that is now being secured will encourage serious work on the solid rock. The Mount Lyell company completed the purchase of the properties acquired under option at the Blue Tier, Zeehan. A track has been cut to the top of Australian Hill, previously inaccessible, and a diamond drill is at work about that portion of the formation. A good deal of testing by jumping long drill holes into the formation has been done, in addition to the trenching work on the Blue Tier Lyell section, and tin-bearing rock has been

discovered in places where its occurrence previously was unknown. A stanniferous discovery is reported from Waratah, Tasmania, where a 12-ft. lode is said to have been opened.

(By James B. Lewis.)—The Anchor tin mine in Tasmania is able to work profitably on a yield of 5 ℔. black tin (tin oxide) per ton, or 3.5 ℔. metallic tin (as the ore assays about 70 per cent.) The accompanying table gives a brief analysis of the working costs for the year ending June 30, 1905. The figures represent pence and decimals of a penny.

WORKING COST, ANCHOR MINE.

	Labor.	Stores and Renewals.	Explosives.	Totals.		Labor.	Stores and Renewals.	Explosives.	Totals.
Quarrying	10.58				Stripping and developing	.50	.01	.03	.54
Maintenance of plant	.55	.27	2.16	13.56	Mine Office	.35			.35
Trucking	1.90	.07		1.97	Sampling	.26			.26
Crushers	1.18	.38		1.56	Sundries	.28	.15		.43
Battery, running	2.20				Freight				.66
Battery, tin dressing	2.03	2.43		8.50	Horse feed				.45
Battery, repairing	1.84								
Power (maintenance races)	.89			.89	Total	22.56	3.31	2.19	29.17

Quantity of ore treated for 12 mo., 118,634 tons. Average assay, 69.1 %. Average number of stamps crushing, 74.25 (out of 100). Stone per stamp crushed in 24 hours., 4.98 tons. Cost per ton at the mine, 2s. 5.17d.

A matter which materially affected the returns was drought which caused a shortage of water for seven months, including an entire stoppage for one month, and which reduced the average of the stamps to three-fourths full power, and the total stone crushed by 30,000 tons, and increased the cost per ton materially. Before the drought we had succeeded in getting costs for one month to 2s. 1.5d. per ton; for six months to 2s. 3.25d. We opened the present year (1906) with an unprecedented drought, which has again thrown us back, but with sufficient water power we can keep down to 2s. 2d. and with fair luck get below 2 shillings.

It is to observed that the figures stated are only costs at the mine, and include nothing for management, office, interest, or smelting expenses, nearly all of which are fairly constant, whether we crush much or little stone. On an output of 140,000 tons per annum this would add about 8¾d. per ton. Nor is anything allowed for depreciation of plant. Wages average 7s. 6d. for a day of eight hours, and the unit of measurement is the ton of 2240 ℔.

We have excellent facilities for cheap work. Water power only is used. The stone is quarried in open faces. It is good ground for boring (which is done by hand), and for shooting, only occasionally coming out in big blocks requiring much "popping." It is seldom very hard, is easily reduced by the stone crushers, and freely gives up its tin in the battery, where it crushes well. In fact, at the present time, in one shed, with stamps of about 800 ℔., we are putting through over six ton sper

stamp in 24 hours. We lose about 20 per cent. of the tin in the stone, not because the battery makes slime, but because it exists originally in that state in the rock.

Many of our methods are crude and our plant is susceptible of much improvement. In fine weather when work can be done in the open to the best advantage we have not enough water power, and in the winter when there is plenty of water, our output is reduced by clogging crushers, self-feeders, etc., and the men cannot work to advantage. We are gradually improving our methods and our plant, and a scheme to regulate the water supply is under consideration.

We are working on an enormous deposit, the known outcrops covering many square miles, and these pass under the country granite to an unknown extent (we have stripped it to a depth of 40 ft.). The actual extent of these tin formations would possibly run into hundreds of square miles in area, and to unknown depth. So far we have made the only systematic attempt to work them. The Mt. Lyell company has secured a very large area and is prospecting a portion of it. Other attempts at working are fitful and futile, the reason being the uneven grade of stone. Small patches of payable stone occur in large masses of poor or almost barren stone, and no satisfactory method of prospecting it has yet been devised. In the early days of the Anchor company the stone, from the information available, was supposed to average 1 per cent. It does not actually average one-quarter of that.

Banka.—The tin mining industry of the Federated Malay States and the Dutch East Indies has been suffering for rather a long time from labor troubles, especially shortage of labor. A commissioner was recently sent by the Netherlands East India Government to study the conditions in Banka. According to his report, the mines need about 5000 more coolies, which will bring the total number up to 16,000. Each coolie is reckoned to produce 12 piculs of tin per year. The authorities are using pressure on the coolies in the island to cause them to go to the mines in preference to agricultural work, and are offering labor brokers a bounty of $90 per head, and moreover are arranging both to feed the coolies better and to raise the rate of wages. The immigration of coolies into the Federated Malay States is decreasing, and fears are expressed that Chinese labor will also become scarce there. In the Malay States the general policy has been recently to secure coolies to work on the tribute system, which they prefer. The average output in the Malay States in 1905 was 4.09 piculs per coolie. The tin production of Banka in 1906 amounted to 190,134 piculs.

Bolivia.—In tin mining in 1906 the most noteworthy feature was the great increase in the production of tin in this Republic, and the prospect that it will be much further increased, when the new railway developments

(which are now in progress) and the influx of Chilean and other foreign capital which is being directed to the exploitation of these tin resources have borne their natural fruits. It was fully realized during 1906 that the tin deposits of Bolivia are of great importance and it is believed that they may make that country the leading tin producer of the world in the not distant future. The production in 1905 was checked by the continuance of the long drought. During the last rainy season the precipitation was copious, and operations were thereby enabled to be prosecuted vigorously.

In addition to the existing railways from Oruro to Antofagasta, and La Paz via Lake Titicaca to Mollendo on the Peruvian coast, a Chilean company is already busy with a far shorter route of access, La Paz to Arica, passing Viacha on the Titicaca line and the copper district of Coro Coro. On July 4, 1906, the President of Bolivia, Dr. Ismael Montes, in the presence of the United States Minister, inaugurated at Oruro the construction of the first of the network of railways contracted for by Speyer & Co. of New York,—namely, Oruro to Viacha, Cochabamba, and Potosi. Two other lines to be built by this firm are Potosi-Tupiza, linking up with the Argentine railway now between Jujuy and Tupiza to form a new Transandine route from the Pacific to Buenos Ayres, and La Paz to Puerto Pando on the Ben, an affluent of the Amazonian waterway.

The tin production of Bolivia has been restricted not only from inability to bring in machinery to extract the mineral, but also from lack of transportation facilities to remove the finished product. Mines are hand-picking rich tin ore, crushing by hand with old-fashioned rocking stones, hand-jigging the coarse and washing the fines in square buddles, kieving the black tin, and accumulating stocks of finished barilla for llamas, when available, to carry it away to the nearest railway station.

According to M. Falansier, director of the French Legation in Bolivia, small miners disdain to mine ore of less than 10 per cent., and treat it in quite a primitive fashion by crushing with a large stone and washing by hand. Modern works, on the contrary, (among which are those of Mr. Soux, at Potosi, Huayuna-Potosi & Melluni, near La Paz (both French), those of Oruro etc.,) treat 5 per cent. ore, and reserve lower grade for the future. These works produce concentrates of as much as 70 per cent. metallic, and ship nothing below 50 per cent.

An Australian prospector, in a recent issue of the Melbourne *Argus*, says: "I visited the recently discovered Kimsi Cruz tin-mining region, situated at an elevation of from 15,000 ft. to 17,500 ft. above sea level. The country rock is altered slate, with occasional bands or seams of porphyry, conforming to the strata. The tin ore is a mixture of cassiterite and iron. The highest mines—those, say above 16,000 ft.—are richest, the tin contents, in beds up to 1.5 m. in thickness, running up to an average

of 10 per cent. metal, whereas beds or lodes in lower country are worthless. One mine, the Monte Blanco, was recently sold to a Chilean syndicate for £170,000 cash, a high price for an undeveloped mine, and, in consequence, owners are asking about 10 times the value of any holding showing any ore. Climatic conditions are severe. Only Indians born in the locality can work, all imported men being unable to do more than just move about, on account of the rarefied air. Although the district is well in the tropics snowstorms are of frequent occurrence, both in summer and winter. The native miners are now receiving 5 to 6 bolivianos a day (about 9s.), and are very scarce at that. The mine owners, therefore, are endeavoring to secure Government sanction to import Japanese miners. Bolivian laws are notoriously bad, and in consequence every mine of value has a crop of law suits."

Among the new Chilean mining companies near Oruro and La Paz is the one owning the rich Llallagua mine, and a newer one called Monte Blanca, owning mines in the Quinsa Cruz region, with a capital of £280,000, of which £220,000 formed the purchase price. The lodes there are said to be wide and rich. Concentration machinery is being installed on rather a large scale, and, when once running, this company should make a large output.

Other concentrating mills being erected at present in the south of Bolivia are two at Chocaya belonging, one to the new Chilean company, Oploca, and the other on property of Aramayo, Francke & Co., each to employ gas engines, ball mills, stamps, jiggers, Wilfley tables, etc., and to produce about 25 tons daily. Another of about the same output, with gas engines, rolls, Buss tables, etc., is being erected at Santo Domingo.

The Chorolque mine is able to produce 200 tons of black tin a month. It has small steam engines (used now as makeshifts only), one 60-h.p. Deutz gas engine, two 20-h.p. petroleum engines, ball mills, jigs, classifiers, Wilfley, Ferraris and Acme tables and is about to install high-speed rolls.

Another well known tin-silver mine has recently changed hands, an American syndicate having taken over the Chilean company owning old mines in Oruro, called Sasson de la Virgen, and the rather large lixiviation and concentrating works of Machacamarca, which has a branch railway line.

The Bolivian export duties on tin ore and bismuth were fixed by decree, Jan. 1, 1906, as follows: Tin ores—If the quotation for Straits tin should be inferior to £100, the quintal of 100 ℔. Spanish shall pay 1.00 Bs.; from £100 to £110, 1.00 Bs.; £110 to £120, 1.15 Bs.; £120 to £130, 1.30 Bs.; £130 to £140, 1.45 Bs.; £140 to £150, 1.60 Bs.; £150 to £160, 1.75 Bs.; £160 to £170, 2.00 Bs.; £170 to £180, 2.25 Bs.; £180 to £190, 2.60 Bs.; £190 to £200, 3 Bs.; £200 and above, 3.50 Bs. Tin bars—If the quotation for Straits tin should be inferior to £100, the quintal of 100 ℔.

Spanish shall pay 1.50 Bs; from £100 to £110, 1.60 Bs.; £110 to £120, 1.75 Bs.; £120 to £130, 1.90 Bs.; £130 to £140, 2.10 Bs.; £140 to £150, 2.30 Bs.; £150 to £160, 2.50 Bs.; £160 to £170, 2.80 Bs.; £170 to £180, 3.10 Bs.; £180 to £190, 3.40 Bs.; £190 to £200, 3.80 Bs.; £200 and above, 4.20 Bs. In accordance with the quotation for Straits tin in Europe, the Minister of Finance will fix fortnightly and with strict observance of the preceding tariff the rate of duty which the ores and bars shall pay, which rates shall remain invariable during the 15 days, notwithstanding that appreciable fluctuations in the quotation for Straits tin may take place in the meantime. Companies working tin mines are exempt from the contribution of 3 per cent. on their net profit hitherto levied. The boliviano is nominally equivalent to 38.48c., or 1s. 7d.

According to Señor Ventura Farfán, superintendent of Bolivian customs, the exportation of tin in 1906 was 638,486 quintals (quintal, 220.46 lb.), or 29,370 metric tons, of concentrate assaying 60 per cent. tin, or, say, 17,622 metric tons of metal, which was the entire quantity produced. The total duties of exportation exceeded the sum of 1,561,740 bolivianos.

The departments of the Republic which produced tin were Potosi, Oruro, La Paz, and Cochabamba, in the following proportion: The Department of Potosi, which comprises Uyuni, Tupiza, Colquechaca, and a part of the exportation through the custom-house at Oruro, 15,674,808 kg.; Department of Oruro, 11,910,477 kg.; La Paz, 1,748,191 kg.; and Cochabamba, 36,892 kg. The production for the year 1905 was 601,948 quintals, or 27,689,621 kg., with a commercial value of 26,205,141 bolivianos.

EXPORTATION OF TIN FROM BOLIVIA.
(In metric tons.)

Year.	Barrilla. Tons.	Metallic Tin. (a) Tons.	Year.	Barrilla Tons.	Metallic Tin. (a) Tons.
1897.....	3,691	2,235	1902....	17,340	10,404
1898.....	4,327	2,596	1903....	21,785	13,071
1899.....	9,134	5,480	1904....	20,369	12,221
1900.....	15,088	9,053	1905....	27,690	16,614
1901.....	21,573	12,943	1906....	29,370	17,622

(a) Tin content of the barilla (black tin concentrate), computing the latter at 60 per cent. metallic tin.

Canada.—The Geological Survey of Canada has announced that tinstone in a solid vein has been discovered by John Keddy at Lake Ramsay, three miles west of New Ross, in Lunenburg county, Nova Scotia. Samples assayed 78 per cent. tin. The ore is found in semi-crystallized forms, disseminated through a vein of decomposed kaolinized pegmatite in granite. The vein has so far been opened only 12 ft. Tin ore has been reported by the Geological Survey to be found at several places in Canada. It was found in New Brunswick on the Pokiok river, in Compton county, Quebec, and also at Buckingham, in Labelle county; in British Columbia in a

3-ft. vein of pegmatite cutting granite near Osoyoos lake, also in Cariboo and Boundary Creek districts, and in the Yukon in several tributaries of the Klondike river; but, so far, most frequently in Bonanza, Hunker, and Sulphur creeks, where it occurs as stream tin in smooth round pebbles up to 2 in. in diameter, which remain with the gold in the sluice boxes. The discovery of tinstone in the Laurentian rocks in Greenland has also been reported.

China.—It has been known that there is a considerable production of tin in China, which does not appear in the commercial statistics for the world. Some light is thrown on this subject by two recent British consular reports.

Mr. Wilkinson, British consul general at Mengtzu, reports: The past year (1905) is the record year for tin. Although owing to the enormous increase in the cost of labor, due to the requirements of the railway, mine owners of Kochiu had to reduce their staff by one-half; nevertheless the mines exported 74,972 piculs (89,252 cwt.), valued at £514,034. The whole of the tin is shipped to Hong-Kong in half slabs measuring 22 in. in length by 9 to 10 in. in breadth and $\frac{3}{4}$ to 1 in. in thickness. Each half slab (the original "pig" is always cut into two, chiefly for convenience of mule carriage, but also to show the quality) weighs 55 catties (73 lb.). Its market value in Mengtzu before payment of export duty is, roughly, £6 per picul of 133$\frac{1}{3}$ lb.

Mr. Carlisle, in his report for 1905, says: The export of tin is an incident of the transit trade. The tin comes from the mines of Ko Tiu, near Mengtse, in southern Yunnan. These mines are exploited by Chinese who bring the tin into Mengtse, whence it is sent down to Manhao on the Red river and thence through Tonkin to Hong Kong. The total amount of the export during the year was 4578 tons, valued at £553,915, and of this value only £2231 refers to Indo-Chinese tin (mostly ore), the bulk of the remainder being the Yunnanese product.

A tin mine in the extreme north of the province of Kwangsi, China, is operated on a small scale, its product being exported through the port of Wuchow. Another tin mine is to be opened in the prefecture of Wuchow.

According to F. Browne, in *Chem. News*, 1906, LXXXXV, 3-4, from 4000 to 6000 tons of tin in the form of flat, rectangular plates, 2 ft.x1 ft. 1 in., arrive annually at Hong Kong from the mines of Yunnan. It is refined at the native refineries, three qualities being produced, of the following composition: I. 99.5-99.8 Sn; 0.1-0.4 Pb; 0-0.1 Sb. II., 98.6-99.3 Sn; 0.1-07 Pb; 0.02-0.3 Sb; III., 95 Sn; 4 Pb; 0.02-0.3 Sb. About 70 per cent. of the tin exported from Hong Kong is stated to be of No. 2 quality. In the refining process the crude metal is first melted and skimmed, and the separated tin poured into molds. The dross is cast into bars

of about 10 ℔. in weight, and heated with charcoal in a wind furnace, the resulting metal being melted and skimmed as before. The residue from the furnace consists chiefly of metallic iron, together with some tin, lead, silica, sulphur, and charcoal, and small quantities of arsenic and antimony. It is sold at $10 per picul.

Congo Free State.—According to the engineers of the Tanganyika Concessions, Ltd., a tin belt extends in the Congo Free State from the Busanga mine on the Lualaba river, a distance of about 110 miles, and in this belt many locations have been made of both alluvial and lode tin. It is considered that this will be a tin field of great commercial importance. Eight and a half tons of bar tin were made in 1905 by the crudest methods, and nine tons were made at Busanga during 1906. The mines are said to be well located for cheap working when adequate means for transportation are provided.

Germany.—Owing to the increased value of tin the old mines in the Erzgebirge, both in Saxony and across the border in Bohemia, are being carefully examined by experts sent out by the Saxon and Austrian government, and by private individuals. A company is now cleaning out the abandoned workings of the old Ehrenfriedensdorf mine, and expects to have it in operation with modern machinery in 1907. German engineers are also examining the abandoned tin mines at Bockau, Sofia, Geyer, Seiffen, Schoenfeld, Rittergrued, etc. The Austrian government is showing equal interest in the possible reopening of the almost forgotten tin mines on the Bohemian side of the mountains, and has experts investigating the old workings at Platten, Baerringen, Neudeck, and Joachimsthal.

Great Britain. (By Edward Walker).—The production of tin in Cornwall in 1906 does not show any substantial increase over 1905. This statement is based on the records of the tin ticketings and not on the official returns from each mine, which are not yet available; but this method of comparison is a very good index of the actual state of things. Most of the tin ore produced in Cornwall is sold by public auction at Redruth every fortnight. These ores are bid for by the five custom smelters in the neighborhood, and little, or none, goes to buyers outside the duchy. A certain amount of the product of the mines and stream works is sold by private contract. Perhaps from one-tenth to one-fifth of the output is disposed of in this way..

The total ore sold at the fortnightly ticketings during 1906 amounted to 6006 long tons, against 5796 tons in 1905. There were 27 ticketings in 1906, so that the increase is more apparent than real. The total value of the ore sold during 1906 was £630,067, as compared with £480,628 during 1905, which reflects the result of the increase in the price of the metal. The above mentioned values are the prices actually paid by the smelters and represent the "market value." Most of the ores are very good con-

centrates and some of these represent 70 per cent. metallic tin. Others are second-class concentrates, while others fetch lower prices on account of the presence of arsenic, copper or wolfram. Perhaps nine-tenths of the concentrates will average 65 per cent. tin, and the remaining tenth, including poor quality of concentrates and stream tin, will range from 40 to 65 per cent.

At the ticketings during 1905 the ores amounted to 5796 tons, while the official returns of the actual output were 7201 tons, which was estimated to contain 4467⅓ tons of metal. It may be a surprise to many to find that the amount of ore sold at the ticketings in 1906 was not greater, so much having been said recently as to the revival of Cornish mining. The present high price of tin being due solely to the shortage of supplies from the East Indies, Cornishmen and London capitalists do not see their way clear to reopen old mines precipitately. They are going slowly and carefully, reorganizing the methods of mining and dressing at some of the producing mines, and reopening others that paid in earlier days. But as yet the output has not been increased materially by these new developments. Another cause for delay in the revival is the scarcity of labor. During the years 1895–1900, while the yearly output of ore was dropping from 13,000 tons to less than 7000 tons, both skilled miners and laborers left for other parts to seek work, and it is not easy to draw them back again, nor to find efficient substitutes. Though Cornishmen are returning in disgust from South Africa, their number is not sufficient to flood the labor market in their own country.

The usual method in dressing the tin ores is to reduce in Cornish stamps and to recover the tin by buddling and by passing over frames. The first concentration takes out the greater part of the tin and other metallic constituents. These concentrates are calcined to drive off sulphur and arsenic, and to reduce the compounds of the other metals to a form which will permit them to go into the tailings at the second buddling operation. In this way by means of the two buddlings with an intermediate calcination very high-class concentrates are obtained.

The tailings from the first concentration are passed continuously over varying forms of buddles and subsequently go over a series of "frames." The tailings in all cases eventually find their way into the rivers and streams. Other people work the streams over again and make a profitable living by the use of the "frame." In all mining camps we are accustomed to seeing people making profits out of the tailings discarded by the big companies, and even in Tasmania, where the dressing plant is near perfection, the Chinaman handles the tailings with profit. In Cornwall it must be confessed that the losses are considerable, and that it is not the frugal Chinaman but white men who make profits out of the tailings.

The Red river, which flows down from the high ground on the south

of Camborne, through Dolcoath and adjoining estates, is full of stream tin works at which the waters containing the tailings from the various mines are passed over inclined tables. About 10 per cent. of the whole Cornish output is due to this work.

The greatest opening for improvement seems to lie in more accurate study of extractions. There does not appear to be any systematic sampling of the ore or of the products at the several points of the dressing process, and the managers seem to have no definite idea of the amounts lost. Not that there is any careless or prodigal waste, but I think that considerable improvements could be made in the extraction if modern tables of all sorts were studied systematically.

There are five smelters in Cornwall: Bolitho's, at Chyandour, Penzance, called the Consolidated Tin Smelting Company; Williams, Harvey & Co., at Hayle; the Cornish Tin Smelting Company, at Redruth, belonging to the Lanyons; the Redruth Tin Smelting Company, at Redruth, owned by the Bains; and the Penpoll, at Devoran, belonging to Mr. Strauss. Up to within the last few years, Bolitho's and Lanyon's have been the most important. Recently, however, Williams, Harvey & Co. have made up leeway and now occupy a leading position. This development has been due to the influence of Dr. Richard Pearce (once of Colorado), who has put money and brains into the concern.

All these five smelters could not make a living out of Cornish ores, and they import large quantities from foreign parts, especially from Bolivia and Australasia. Probably the Penpoll lives entirely on foreign ores. There are a number of Cornish companies financed by smelters and other local people operating tin mines in all parts of the world. The produce of some of these is brought to Cornwall for smelting, but that of others, especially on the Straits, is treated on the spot. The bulk of the imported ores is, however, bought in the open market. It should be added that no Cornish ores are ever exported.

Malay States.—The output of tin for the Federated Malay States during 1906 was 48,616 tons of 2240 lb. against 50,991 tons in 1905. The imports of tin into these States during 1906 was 8078 tons and in 1905 7628 tons, mostly from Siam and the Dutch East Indies. The exports from the Straits Settlements, including the ports of Singapore and Penang, are shown in the accompanying statement.

There was a decrease in the total exports during 1906, as compared with 1905, of 260 tons, and a falling off in the output of 2373 tons. The total exports for 1906 exceed the output and imports by 2201 tons. Seventy per cent. of the gross weight of ore is estimated to represent the amount of block tin which the ore will produce.

It is largely to the failure of the Malay States to maintain their production that the recent great rise in the value of tin has been due. Foo

Choo Choon, member of the Perak State Council, who is one of the leading tin magnates of the States, was interviewed recently respecting the tin prospects of the States, and expressed the opinion that owing to the

PRODUCTION OF TIN IN THE MALAY FEDERATED STATES.
(In piculs of 133⅓ lb.)

	1900	1901	1902	1903	1904	1905	1906
Perak	355,590	385,060	405,870	436,296	443,507	446,781	435,909
Selangor	269,490	302,570	278,360	284,592	300,413	289,867	268,624
Negri Sembilan	82,320	75,230	73,520	85,461	84,849	85,133	77,766
Pahang	15,700	26,310	23,120	25,317	27,469	34,879	34,488
Total in piculs	723,100	789,170	780,870	831,666	856,238	856,660	816,787
Total in metric tons	43,123	47,713	47,211	50,254	51,790	51,793	49,859

scarcity of tin the present price would maintain for some time to come. The surface mines at Perak have been practically worked out and deep mining is now necessary. Deep mines, however, mean big capital. All the principal mines in the States are equipped with modern machinery. He thought it possible to get tin in payable quantities from the old Chinese workings or lampans. In Taipeng there are a lot of abandoned mines. If the Government would give them to other people who had labor-saving machines, he thought it would pay to work them again.

EXPORTS OF TIN FROM STRAITS SETTLEMENTS.
(In tons of 2240 lb.)

Countries.	1906	1905	Countries.	1906	1905
United States	15,008	16,879	Netherlands India	160	95
England	34,434	31,512	Philippines	1	
Continent of Europe	7,437	8,492	Siam	7	1
China	433	483			
India	856	1,014	Total	58,897	59,157
Japan	561	661			

The tin smelting industry of the Straits Settlements is largely in the hands of the Straits Trading Company, Ltd., of Singapore. The report of this company for the half-year ended Sept. 30, 1906, is as follows: After making ample provision for bad and doubtful debts and writing off depreciation, the net profits amount to $532,365.91, to which has to be added the balance of $91,378.30 brought forward from last account, making a total of $623,744.21 available Sept. 30, 1906. The directors recommended (1) that a dividend of $1 and a bonus of 50 cents per share be paid to shareholders, absorbing $375,000; (2) that $100,000 be added to the reserve fund, which will then stand at $1,000,000; (3) that $50,000 be added to the fund for equalizing dividends, bringing it up to $250,000; (4) that $20,000 be transferred to employees' bonus ac-

count; (5) that the balance of $78,744.21 be carried forward to a new account.

For some considerable time past, scientific and practical experiments have been undertaken at the works, having in view the further profitable treatment of the large slag and other refuse heaps, accumulated since the formation of the company. The results have been eminently satisfactory, and have contributed appreciably to the profits. The new and expensive slag plant installation is now working very efficiently, and, with the reserve stock of slag, will continue to prove a handsome source of revenue for years to come.

The tin smelters of the Colony are protected by an export duty. On June 1, 1906, a new regulation was put into effect, replacing the 10 per cent. *ad valorem* duty imposed Nov. 1, 1904, and practically restoring the scale established in November, 1903. Under the new schedule, when the price of tin is $37-$38 per picul the duty is $11.50 per bhara. For every additional $1 per picul in the price of tin, $0.50 per bhara is added. The duty is reckoned on the price telegraphed daily from Singapore. On tin won from rock such that the tin cannot be obtained therefrom without picking, blasting, or crushing the duty is payable subject to such rebate, if any, as the resident may in particular cases approve on the joint recommendation of the Senior Warden of Mines and the Government Geologist of the Federated Malay States; but no such rebate shall exceed 50 per cent. of the duty prescribed. Tin ore exported under such guarantees as the resident may require that it will be smelted in the Colony of the Straits Settlements or in the United Kingdom is subject to 70 per cent. of the duty on tin. Tin ore exported without such guarantees is subject to 70 per cent. of the duty on tin, plus an additional duty of $30 per picul.

Nigeria.—In 1904-1905, John Parkinson, and H. L. Huddart, engaged on a mineral survey for the British Colonial Office, discovered deposits bearing tinstone while making a traverse of the Uwet district in Southern Nigeria. The district examined lies between Awai Iku, on the north, the Ekpri-Ibami on the south, and along the route passing through Akwa-Ibami, Uyanga, Oyja-Ekankpa, Oyja-Nkorimba, and Ofunapa. The deposits met with along this route were panned for tinstone. Eighty-seven concentrates were thus obtained. Of this series of concentrates, only those from the Akwa-Ibama district are sufficiently rich in tinstone to be of importance from a commercial standpoint.

The impurities in the tinstone concentrates of Akwa-Ibami consist of columbite, garnet, ilmenite, and tourmaline, with a small amount of quartz and occasionally magnetite. Quartz can be eliminated by washing; the other impurities, can be fairly well separated by electro-magnetic methods. In this way, the tin-bearing deposits of Akwa-Ibami

could be made to yield a product containing 75 per cent. of metallic tin. The proportion of cassiterite in the stream beds in the district around Akwa-Ibami is 2 to 5.2 ℔. per ton, the average being 3 ℔. per ton. The concentrates contain on the average about 70 per cent. of metallic tin.

A mineral survey in Northern Nigeria was made by J. D. Falconer and D. R. Horne. The most important tin deposits are said to be those of Bautchi, which were examined by Mr. Nicolaus on behalf of the Niger Company, the following being the substance of the report: The tinstone that is found is alluvial in coarse and fine grains. It can be secured very pure by simply washing the sands and gravel, the resulting product, having a value of £73 per ton without smelting. The black tin contains no impurities whatever detrimental to its smelting. The average value of a ton of river gravel is, by the samples, £1 5s. 6d. The richest of the stanniferous gravels and sands in and near the river and its tributaries extended a distance of about 11 miles, which was the only area worked for stream tin by the natives. The fine tin was traced for a distance of about 14 miles, making altogether a length of about 25 miles of river commercially workable for tin.

In the Bautchi tinfield all the tinstone so far obtained has been taken from the stream beds. Here the Haussa laborers, in gangs of three or four, wade into the river tributaries, creeks or gullies at or near shallow rapids, and scoop the gravel into large calabashes. When sufficient gravel has been collected it is washed, and the tinstone so obtained smelted locally. A gang of four laborers will wash sufficient gravel to yield from 80 ℔. to 1 cwt. of clean tinstone per day. The average yield of tinstone so far obtained in Bautchi works out at 35.66 ℔. per ton of gravel. The local market price for tinstone is about £15 15s. per ton. The source of Northern Nigeria tinstone is supposed to be a stockwork formation at the base of the Gura mountains.

Cassiterite has been found in the stream beds of other districts. It is probable, in the opinion of the officers of the Colonial Office, that as soon as suitable transport is provided, Northern Nigeria will become an important tin-producing country. The deposits in the Bautchi province are being worked by the Niger Company, and the metal smelted on the spot. The first consignment of tin from the Protectorate has recently reached Great Britain.

Sir Frederick Lugard, in his report on Northern Nigeria, dated Nov. 27, 1906, states: The Niger Company, which had held prospecting licenses in the Bautchi province, applied in 1905 for mining licenses over selected areas, at Naraguta. This involved a careful survey, more especially since an uncertainty existed as to whether one of the areas lay within its district or partially within the district included in a license granted to Messrs. Rickard & Co. The licenses were finally granted in

February, 1906. By March "practically the whole of the machinery was in position and working well." The water supply caused some initial difficulty, but this is only in the driest month of the year, and can be surmounted by dams. The chairman of the company, in his annual report to the shareholders, states that "the developments this year have been very good indeed, and leave no doubt as to the future." It is anticipated that an immediate output of one ton a day of black tin can be realized, and the company claims that it has demonstrated that tin can be successfully worked. Messrs. Rickard & Co. intend shortly to commence prospecting in their district. The most urgent need for the development of this industry is a practicable and direct route to the Benue river at Loko. Steps have already been taken by the Government to open up and safeguard this route.

Russia.—According to the *Viestnik Finanzoff*, the tin question is a serious one for Russia. With the final stoppage of the Pitkäranta tin smelting works, production has ceased. "Therefore, serious attention should be paid to the tin deposits of the Transbaikal region on the Osion river, which by now have been sufficiently prospected, and in both quantity and quality are hardly to be beaten by any known tin deposits."

Siam.—Tin is found in the valley of the Nam Sak river, and in various places in Northern Siam, but all the deposits of importance are derived from, and lie adjacent to, the great line of granite upheaval which forms the boundary range between Central Siam and Tenasserim. The Siamese territory is rich in deposits, especially in the possessions in the Malay Peninsula. Tin is being worked on the east coast at Ratburi, Bangtaphan, Langsuan, Chaiya, Bandon, Lakon, Jalar, Rangeh, Rahman, Kelantan, and Tringannu. On the West Coast it is being worked at Kra, Renong, Takuapar Panga, Takuatung, Puket, Trang, Setul, Perlis, and Kedah. The annual production is estimated to be about 5000 tons.

According to the Penang *Gazette*, there is a great field for the expansion of the tin mining industry in the Siamese possessions in the Malay Peninsula, and considerable activity in prospecting on the part of European capitalists has lately been shown. At present, Puket island on the West Coast has a considerable mining industry, as well as Kedah, Kelantan, Takuapar, and Renong. On the east coast Lakon Sri Tammarat, and Jalar are the chief centers. The most promising districts for future developments are in Kedah, Rahman, Jalar, Takuatung, and Renong.

South Africa.—According to *South African Mines*, Jan. 26, 1907, despite the discredit which was brought on South Africa's prospects as a tin-producing country by the collapse of Vlaklaagte, the Transvaal, Swaziland, and the Cape Colony are each producing tin ore, and South Africa has reason to hope that before long tin ore will be one of her valuable exports. After a long delay the mill of the Bushveld Tin Mines has started crush-

ing, though it may not be practicable to run the full mill of 10 stamps yet awhile. Recent developments on the properties of the Transvaal Consolidated Land Company near Potgietersrust have been of an encouraging nature. The company has made a profit in developing the stanniferous deposits and is about to ship a further 50 tons of cassiterite. Development has proved the existence of seven or eight sausage-shaped lenses about 9 ft. long by 3½ ft. wide. These lenses have been sunk on to a depth of 116 ft. and show no signs of pinching out at that depth. The lenses have been proved over a distance of about three miles. In the Eckstein tin field, the T. C. L. Company has already produced about 95 tons of undressed tin ore. The first parcel despatched was sold in England for £48 per ton and the company had on hand at the end of the year a further 190 tons of similar quality and 350 tons of low grade ore.

In Swaziland tin mining is being carried on in at least two mines. The principal one is under the control of H. Eckstein & Co., and is reported to be opening up in a satisfactory manner. At Kuil's river, a new pumping plant will soon be in working order. Hitherto the average quantity of ore shipped has been 12 tons per month, but with adequate pumping machinery in operation, the shipments will be much larger. A 150-h.p. engine is employed, and 200 Wragg machines are in course of erection, which will take one ton a day over each machine. The recovery is expected to be 10 lb. per ton. There are about 230 boys at work operating on washings, which are estimated to be 15 ft. in depth. At the proposed new scale of operation there is calculated to be 30 years' working. Already 213 tons have been exported and 12 more are waiting. Sufficient has been done to establish confidence in the enterprise, and when it is taken into consideration that on account of the floury fineness of most of the alluvial tin only about one-third has been secured in the washing—the remainder waiting more thorough treatment—it is permissible to conclude that the property is of considerable value.

Spain.—Renewed attention is being paid to some of the Spanish deposits. A group named the Sultana, situated in the Province of Orense, has been acquired by a syndicate of London, Paris and Manchester men and floated as the Arnoya Mining Company, Ltd. According to Mr. Wickersheimer, a French engineer, the lodes are extensive, and considerable work has been done on them already. The tin occurs as oxide, in conjunction with arsenical pyrites, but it is stated that the amount of arsenic is relatively small. The estimates of the extent and richness of the orebodies are somewhat general in character. One lode is said to contain 300,000 tons averaging 5 per cent. of tin oxide. It is stated also that the coarse granite and quartz forming the vein matter are very friable, and can be mined without explosives.

Notes on the Metallurgy of Tin.

Tin Smelting at Launceston, Tasmania.—According to J. D. Millen[1], the ores received at the Mt. Bischoff works average about 70 per cent. tin, and as a rule are fairly pure. The plant consists of reverberatory furnaces, the ore being charged into them through side doors, the charge consisting of 50 cwt. of ore and about 10 cwt. of small coal, more or less, according to varying circumstances. After charging, all the doors are luted up. The time required to reduce the charge is eight hours, during which it is subjected to several rabblings. When the smelting is complete, the slags are skimmed off and reserved for further treatment, and the metal is tapped into a brick-lined receptable, and allowed to cool somewhat. Another charge is then thrown in, and the operation repeated.

The metal, after cooling, is ladled into a large kettle, where it is refined by a kind of poling, billets of green wood being sunk under the surface. The oxide or dross, which rises to the surface, is skimmed off. Dip samples are continually taken, and when the metal is sufficiently refined it is ladled into molds holding about 80 lb. The refined metal assays 99.89 per cent. tin. The slags from the ore furnace are broken up, and mixed with small coal, lime, and, if necessary, iron, and again smelted.

A few small parcels of ore, which is pyritic in nature, are received. These require a thorough roasting before the ordinary process of reduction is proceeded with. Further extensive alterations to the present works are under consideration, and when these are complete the character of the ores that the plant will be capable of treating will be much more varied.

Tin Smelting at Irvinebank, Queensland.—The works of the Irvinebank Tin Mining Company comprise 40 stamps, 30 being retained for the use of the Vulcan mine and 10 for public custom. There are nine slime tables, five Wheeler pans, a Phœnix weir concentrating table, besides the Luhrigs and vanners. There are two reverberatory furnaces, one of four tons, the other of six tons capacity. From the drier at the mill the ore is conveyed to a calciner, where the sulphur is burned off, and the bismuth which is generally associated with tin in this district is volatilized and caught in a chamber. From the calciner the metal passes to a leaching vat, where any traces of copper are leached out. The black tin is mixed with charcoal in certain proportions, moistened to prevent loss by dusting and let down into the furnace on a thin bed of coal. It takes nine hours to smelt a charge. When ready the furnace is tapped, the molten metal running into the fore hearth. The slag overflows into a drain prepared for it. The tin flows into a collecting

[1]Abstract from the Annual Report of the Zeehan School of Mines and Metallurgy.

pot, where the remaining slag is skimmed off. From the collector it runs into the refiner, where it is poled. When finished it is ladled into molds, which form ingots of 86 lb. weight.

Assay of Tin.

Determination of Tin and Tungsten.—E. Donath (*Zeit. f. angew. Chem.*, 1906, XIX, 473-474) gives the following method for the analytical separation of tungsten and tin: The mixed tungstic and stannic oxides are heated in a covered porcelain crucible with zinc dust or zinc filings, for a quarter of an hour, and after cooling the mass is heated with 1:2 hydrochloric acid till no more hydrogen is evolved, and all the metallic tin is, therefore, dissolved. To the cooled liquid, potassium chlorate is cautiously added till the blue color has entirely disappeared, and the tungsten is entirely in the form of trioxide. The liquid is now diluted with at least 1.5 times its volume of water, and allowed to stand for 24 hours, when the whole of the tungstic oxide is deposited. It is filtered off, washed first with water containing nitric acid, and finally with a hot dilute solution of ammonium nitrate, dried, ignited, and weighed. In the filtrate the tin is precipitated as sulphide.

Angenot recommends (*Zeit. f. angew. Chem.*, 1906, XIX, 140) that ores containing these two elements should be finely pulverized and intimately mixed with eight times their weight of Na_2O_2 in an iron crucible. Heat up carefully until the mass is fluid, and maintain this condition for about 15 minutes, moving the crucible somewhat to stir the contents. Cool, take up with water (if Pb is present pass in CO_2) dilute to 250 cc., and then filter off two lots of 100 cc. each. One lot (for W) is run into a mixture of 15 cc. conc. HNO_3 with 45 cc. conc. HCl, evaporated in a porcelain dish until "dust dry." The residue is treated with a solution containing 100 gm. NH_4Cl and 100 gm. conc. HCl per liter. The insoluble portion is dissolved in warm ammonia; run this solution again into a solution of 15 cc. HNO_3 with 45 cc. HCl as before, and again evaporate, and treat as before. The insoluble portion after washing is pure WO_3. To the other lot (for Sn) add 40 cc. conc. HCl which precipitates WO_3 and SnO_2, then add 2 to 3 gms. pure Zn. Allow to stand at 50 to 60 deg. for an hour. Sn will have passed into the form of $SnCl_2$, the major part of the blue W_2O_5 being in an insoluble form. Filter, and in the filtrate precipitate Sn by H_2S. The remainder of the treatment needs no further directions. To insure avoidance of loss of Sn as metal in the residue, dissolve off the W_2O_5 from the filter by warm NH_4OH. If any metallic particles remain, dissolve in HCl and add to the main solution.

Separation of Antimony and Tin.—According to A. Czerwek (*Zeit. anal. Chem.*, Aug. 31, 1906, p. 829) tin may be separated from antimony by precipitation as the double compound of stannic acid and phosphoric acid,

but it is necessary that chlorine ions should be absent. For the analysis of an alloy, 0.5 gram is covered with a mixture of 15 cc. of nitric acid (sp. gr. 1.42), 15 cc. of water, and about 6 grams of tartaric acid, the temperature being 40 to 50 deg. C. When solution is complete, the liquid is heated to boiling and 5-30 drops of 45 per cent. phosphoric acid (sp. gr. 1.3) are added, according to the amount of tin present. After diluting with about 300 cc. of boiling water, and allowing to stand on the water-bath for 15 minutes, the solution is decanted on to a filter, and the precipitate washed with water containing ammonium nitrate. The precipitate is dissolved in warm ammonium sulphate, and then, after cooling and diluting largely, the tin is again precipitated by adding sulphuric acid. The mixture is allowed to stand some time on the water-bath, and the greenish gray precipitate is then filtered off, and washed as before. The dried precipitate is added to the filter ash in a crucible, oxidized with nitric acid (sp. gr. 1.42), dried on the water-bath, ignited and weighed as tin dioxide. The filtrate from the tin is neutralized with ammonia, and then warmed with ammonium sulphide, and acidified with acetic acid. The precipitated antimony sulphide is dissolved in ammonium sulphide, evaporated to dryness, and oxidized to antimony tetroxide in the usual way, with fuming nitric acid. Analyses are given of alloys containing antimony and tin only, and also of those containing lead, copper and zinc in addition. In dealing with these latter, tin is precipitated as before, and ammonium sulphide separates the other metals from the filtrate. In all cases in which antimony and tin have been obtained free from other metals by solution in ammonium sulphide, electrolytic precipitation has been found satisfactory as a preliminary to their separation.

Commercial Tin.—Victor (*Chem. Ztg.*, XXIX., 179) gives the following method: Take 20 grams if the metal is 99 per cent. or 10 grams. if it is 98 per cent. or less. Dissolve in 100 cc. HCl (of sp. gr. 1.124) in a liter Erlenmeyer flask with the aid of moderate heat. When nearly cold add $KClO_3$ in small amount at a time, until the foreign metals are dissolved. Then expel the Cl by long boiling. Meantime dissolve 30 grams $H_2C_4H_4O_6$ in water, supersaturate it with NH_4OH, add to the solution of the sample, and render the whole mixture *slightly* alkaline with NH_4OH. Then pour into it water at 40 deg. C. which has been saturated with H_2S, until no further precipitation occurs, filter and wash with H_2S water. Digest the washed precipitate for some time with Na_2S solution, to remove all Sn, filter and dissolve in HNO_3. Evaporate down to fumes with H_2SO_4, filter off $PbSO_4$, dissolve same in $NH_4C_2H_3O_2$, add H_2SO_4 and evaporate again to fumes, etc., to obtain pure $PbSO_4$ for weighing. In the filtrate from the Pb separate Bi and Fe by NH_4OH and $(NH_4)_2CO_3$ and in the filtrate from that determine Cu.

For Sb dissolve 10 grams in HCl as before, oxidize with $KClO_3$, boil out Cl, and dilute largely. Place in a flask provided with a Bunsen valve, add an iron nail and a small amount of ferrum reductum (iron "by hydrogen"). The metallic Sb is precipitated after about 45 minutes' action if the solution is kept moderately warm. After washing off the nail, filter, dissolve in $HCl + KClO_3$, boil out Cl, neutralize with excess of NaOH, add Na_2S and KCy and electrolyze.

In block tins containing 96 per cent. or less, the Sn is determined volumetrically by titration with strongly acid Fe_2Cl_6 solution. The indicator is made by dissolving 10 grams KI in 10 cc. of water, and adding 10 grams HI (sp. gr. 1.5) containing 3.3 grams Cu_2I_2. The preparation should stand for some days before using. A few pieces of metallic copper should be introduced, and the mixture should be kept in the dark. When used, it should be clear and colorless. The starch added to the tin solution at the time of titration, will react with the I of this indicator as soon as the Fe_2Cl_6 is in excess. Ten drops of the indicator are used at a time. The Fe_2Cl_6 solution is made with 275 grams of the sublimed salt, dissolved in 250 cc. conc. HCl, and diluted to 10 liters. 1 cc. should equal 0.01 gram Sn.

To carry out the process dissolve 5 grams of the sample with the aid of $KClO_3$ and dilute to 500 cc. Put 50 cc. into a flask with a Bunsen valve, add a little ferrum reductum, and warm 20 minutes on the water-bath. A portion of the Fe should remain undissolved when the reduction of Sn is complete. Filter through asbestos on which a little ferrum reductum has been strewn, and wash with diluted HCl, add to the filtrate 100 cc. conc. HCl, 10 drops of indicator and 1 cc. starch solution. Near the end of the titration add a few drops of the I indicator again to be certain of the end reaction. Reduction by Al, as preliminary to titration as above, is preferable, as it avoids filtration. The reduction should be effected in a flask through which a current of CO_2 is passed. After the Al has reduced all the Sn to metal, 50 cc. HCl are added and the metal again brought into solution, when the titration can be performed as described.

The Tin Markets in 1906.

New York.—More than ever before in its history, tin was used as a football by speculators in the London market, as well as in New York, during 1906. It was comparatively easy to create a bull or bear position as the available supplies were closely concentrated in a few hands. As the production, at least so far as the East Indies were concerned, did not increase, while the consumption showed remarkable expansion, no stocks of consequence were accumulated during any part of the year. This fact assisted materially in making manipulations at the different exchanges easy.

The consequence was an excited and exaggeratedly high market when the bulls had the upper hand, and a rapid decline in prices when the bears were victorious.

On going in detail into the market situations as the year developed them, we find that at the beginning of 1906 spot tin was selling at 36c. per lb. It fluctuated between this and 37c. per lb. until toward the end of March, following closely quotations at London. In April a shortage of spot supplies became acute and a corner proved so effective that toward the middle of May owners of small available stocks were able to drive prices up to 48c. and 50c. per lb. During that time a large quantity of Chinese tin was brought into this market and consumers who had never before dared to use anything but prime Straits of Malacca tin were only too glad to have an opportunity to replenish their supplies by acquiring some of this lower-grade Chinese tin.

Fortunately this corner did not last very long, but was followed during the last week of May by a slump which brought prices down by about £30@35 in London and from 6@8c. in this market. The market held steady at this level during June and the beginning of July, at which time spot tin was selling for from 39@39¼c. per lb. July again witnessed a rather bad break in London and simultaneously in this market prices touched 36c. Prices righted themselves during August, and retained a level of 40½@41½c. per lb. until the end of September. During October the bull party in the London market had everything its own way and prices held around 43@43½c. When, notwithstanding the firm and high market, large orders from American consumers failed to materialize, and it was even found that one of the largest consumers of tin in this country was reselling some of its purchases in the London market, the influence on that market was rather depressing. Prices settled gradually into a lower level and closed the end of the year at 42c.

Developments in the different producing centers of tin during 1906 showed that no increase need be expected in the Straits Settlements. While some of the mines there are petering out entirely, a great many others are running on leaner ore and prospecting work in new directions is not being done by the Chinese. Production in the Dutch colonies is continuing at the regular rate. There should be a large increase in the output of Bolivian tin, and as soon as the necessary railroads have been built in that country an increased tonnage ought to be available for export. The high prices for tin have also had a stimulating effect on tin mining in Wales and also in some parts of Saxony, where, however, the production at best is not large.

So long as the output of this important metal is, comparatively speaking, limited and handled to a large extent through London brokers, the disturbing element of wild speculation cannot be eliminated.

AVERAGE MONTHLY PRICES OF TIN PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1896	13.02	13.44	13.30	13.34	13.54	13.59	13.63	13.49	13.15	12.94	13.09	12.96	13.29
1897	13.44	13.59	13.43	13.34	13.44	13.77	13.89	13.80	13.98	13.88	13.79	13.71	13.67
1898	13.87	14.08	14.38	14.60	14.52	15.22	15.60	16.23	16.03	17.42	18.20	18.30	15.70
1899	22.48	24.20	23.82	24.98	25.76	25.85	29.63	31.53	32.74	31.99	28.51	25.88	25.12
1900	27.07	30.58	32.90	30.90	29.37	30.50	33.10	31.28	29.42	28.54	28.25	26.94	29.90
1901	26.51	26.68	26.03	25.93	27.12	28.60	27.85	26.78	25.31	26.62	26.67	24.36	26.74
1902	23.54	24.07	26.32	27.77	29.85	29.36	28.38	28.23	26.60	26.07	25.68	25.68	26.79
1903	28.23	29.43	30.15	29.81	29.51	28.34	27.68	28.29	26.77	25.92	25.42	27.41	28.09
1904	28.85	28.09	28.32	28.13	27.72	26.32	26.57	27.01	27.78	28.60	29.18	2.9292	27.99
1905	29.325	29.262	29.523	30.525	30.049	30.329	31.760	32.866	32.095	32.481	33.443	35.835	31.358
1906	36.390	36.403	36.662	38.900	43.313	39.260	37.275	40.606	40.516	42.852	42.906	42.750	39.819

London.—Opening at £161 12s. 6d. for spot warrants, and £161 7s. 6d. for three months', the market was momentarily depressed by realizations for bull account. But buying orders predominated, chiefly for American consumption, and offers were readily absorbed. The American demand continued unabated, and speculation favored the rise—particularly when it was known that the Banka sale had realized £169 2s. 6d. The closing days of the month brought forth a volume of demand which frustrated all bear tactics, and prices rose rapidly to £166 5s. for spot, and £165 7s. 6d. for three months', closing strong.

February revealed a reduction of 650 tons in the London stocks, notwithstanding large shipments from the Straits, and the price of spot warrants rose sharply to £167 2s. 6d. This was followed by a slackening in the American demand, with some pressure of bear sales. Advices from the East indicated shortage of labor and of material, and a renewal of American demand sufficed to absorb all parcels offered. On the other hand, the Welsh demand fell off, and confidence was disturbed by political complications, and the closing days of the month were comparatively uneventful, with spot warrants £165 10s., and three months' £163 15s.

March opened with a temporary depression. Prices drifted down, but this was the signal for renewed activity and an almost uninterrupted advance till the end of the month. The closing prices were £169 for spot warrants and £166 for three months'.

April found abundant evidence of serious shrinkage in stocks, East and West, resulting in large dealings and a general anxiety to cover all commitments. Prices advanced rapidly, American buyers in particular being quite ready to pay top prices. This continued throughout the month, which will be memorable for its volume of trade and for the record prices paid. Some realizations helped to ease the situation, and to encourage the bears, whose operations lowered the price to £177, whereat the month closed, the backwardation being thus widened to £6 10s.

May brought forth the most remarkable development the market had ever known. There was at first some hesitation, but this was followed by

bold buying, chiefly on behalf of consumers. Prices advanced by leaps and bounds. By the middle of the month spot warrants had advanced from £183 7s. 6d. to £215, and three months' from £177 15s. to £205. At this juncture there was some aggressive bear selling, followed by a general slump in values. By May 21 spot warrants had fallen back to £184, and three months' to £175. The advent of new orders from consumers initiated a recovery to £190 15s. for cash, and £188 15s. for three months'. Fluctuations continued bewildering in their breadth and frequency. The Banka sale on May 29 went at the unexpectedly low average of £181 2s. 6d. The month closed with an easy tendency, £181 15s. for spot warrants, and £181 10s. for three months'.

June began with an easing off in prices. The noticeable feature was the cessation of American demand. Bull accounts in many cases were liquidated by disappointed holders, and some parcels arriving in London had to be warehoused. A further fall was the natural result, but an improvement set in later. The market was irregular and subject to frequent bear attacks. Further decline was arrested by the firmness of Eastern sellers, and by improved demand for August delivery. The close of the month brought values up to £177 5s. for prompt, £175 15s. for three months', and £177 15s. for August dates.

July was chiefly noticeable for the premium commanded by warrants for August dates throughout the month. The opening days saw a steady decline in values. There were sharp fluctuations. The periodical Banka sale realized the equivalent of £175 for delivery in London, which price was too inflated for the London market to follow. A notable feature of the month was the disappearance of the backwardation on three months' warrants; it reached its maximum of £3 15s., July 2, gradually dwindled until July 17, when the price was uniform for cash and for three months', revived and reached 12s. 6d., July 19, again dwindled and was wiped out on July 30, when, after a long interval, a contango was established. Closing prices were £170 12s. 6d. for cash, and £170 17s. 6d. for three months'.

August gave promise of renewed American demand. The situation was rendered more acute by numerous orders from consumers, with the result that the first four days witnessed an advance. A sharp reaction followed, bringing down the cash price, partly induced by selling pressure on the part of Chinese dealers who had missed their opportunity and had taken alarm. Trade requirements were, however, too manifest to allow of any lengthy depression, and a gradual recovery ensued which, with few checks, lasted till the end of the month. New York orders were heavy and urgent, and a large general consumption was assured. The contango on three months' warrants varied within £1 per ton, but had disappeared by the end of the month when the price was uniform at £184 15s.

September opened with a slight set-back, followed by quick recovery

prompted by large and urgent American demand. A large business was done in English refined sorts. Toward the end of the month a steady advance ensued. The periodical Dutch sale fetched the relatively high equivalent of £188 15s., and was followed by renewed consumptive inquiry. Closing values were £191 for cash warrants and £190 7s. 6d. for three months'.

October opened with a sudden advance. This proved a temptation for bears, whose operations brought about a fall, followed by subsequent recovery. Smaller fluctuations followed with a rise to £197. By Oct. 25 the demand from America had become spasmodic, and the market was hardly prepared for the surprise which came from that quarter on Oct. 29, when brokers received instructions to sell quantities for early shipment from the United States at below current value. This was generally regarded as a crude stratagem, seeing that there were abundant indications of American stocks requiring replenishment rather than depletion. It served, however, to disturb the market with wild fluctuations. The panic was of short duration, and the month closed with a firm tendency, cash warrants commanding £192 12s. 6d. and three months' £194 5s.

November found the market ready for a sharp advance which, within three days, raised prices about £5 per ton. This, however, was too sudden to be sound. A sudden drop of between £3 and £4 served to clear the situation. With small stocks and active consumption a gradual improvement set in. Final quotations were £196 15s. for prompt cash, and $197 for three months'.

December opened with an initial advance, promptly followed by a determined onslaught by bears and consequent free selling from the East. This quickly gave way to a recovery. Thereafter the market steadied, and prices moved within narrow limits. Toward the close bear selling was again aggressive. Realizations of near prompts further tended to depress prices, and transformed the recent backwardation of 15s. into a contango of the same figure. The underlying strength of the market was manifested at the close, when prices stood at £196 and £196 15s., respectively, for spot and three months'.

TUNGSTEN.

By Reginald Meeks.

In recent years tungsten mining has received considerable impetus from the introduction of "high-speed" steel and the growth of the industry has been rapid, especially in the United States. The demand for tungsten ore began in 1900; in that year \$2@3.50 per unit was paid for concentrate containing 60 per cent. WO_3. Since then the production in the United States has steadily increased and the price advanced to \$9.50 per unit in 1906.

PRODUCTION OF TUNGSTEN CONCENTRATE IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Production.	Value.	Average per Ton.	Year.	Production.	Value.	Average per Ton.
1900	46	\$11,040	\$240	1904	740	\$184,000	\$249
1901	179	27,720	155	1905	834	257,463	308
1902	184	33,112	180	1906	1,096	443,150	404
1903	292	43,639	149				

(a) Statistics reported by the U. S. Geological Survey, except for 1905 and 1906.

Arizona.—The Euclid Mining Company operated wolframite mines in the Whetstone mountains during 1906 and discoveries were also reported on the property of the Buffalo-Arizona Company near Phoenix.

California.—The property of the Minnehaha Mining and Milling Company at Caliente, Kern county, was originally worked for silver and lead but a ledge of scheelite, parallel to the first deposit, proved of more value. It was reported that the company had contracts for 10 tons per month. The workings are at a depth of 200 ft. and ore is found containing 50 per cent. tungstic acid. The Atolia Mining Company, Randsburg, Kern county, produces high-grade concentrate, which is shipped to Germany. This company operates mines in Arizona, Nevada and California.

Colorado.—The production of tungsten concentrate in Colorado in 1906 was 823 tons valued at \$352,150 against 720 tons worth \$216,000 in 1905. One of the features of the year was the entrance of two of the largest consumers into the field as producers. The Primos Chemical Company, Primos, Delaware county, Penn. through its subsidiary company the Stein & Boericke Mining Company bought the property of the Great Western Exploration and Reduction Company and will erect sampling works at its mill in Gordon gulch. The Firth-Sterling Steel Company is interested in the Boulder county mines through the Wolf Tongue company. These

two companies, together with the Colorado Tungsten Corporation and the Henry E. Wood's ore testing works, are the principal producers of Colorado.

(By William A. Greenawalt.) The tungsten deposits of Boulder county, Colo., which furnish more than 80 per cent. of the tungsten of the United States, were discovered more than 25 years ago, but were not developed until about five years ago. The possibilities of the industry were slow in being recognized, and it was not until Morris J. Jones, of the Great Western Exploration Company, developed a mine and built a mill, that it was firmly established.

The tungsten district of Boulder county is eight miles long and three miles wide, but good prospects have been found outside of this area. The town of Nederland is the center. Good tungsten ore has been found as far south as Rollinsville, as far west as Eldora, as far east as Sugar Loaf and Magnolia, and as far north as Ward, where the ore occurs as hübnerite.

All of the tungsten ore about Nederland occurs as wolframite (Fe, Mn) WO_4, which contains when pure, about 76.5 per cent. tungstic acid (WO_3), 9.5 per cent. ferrous oxide, and 14 per cent. manganous oxide. The concentrate, as produced at the mills in Boulder county, varies from 60 to 70 per cent. tungstic acid. A characteristic analysis of Nederland ore is: WO_3, 63.20; FeO, 20.36; MnO, 1.10; SiO_2, 15.00; P, trace; S, 0.50; gold, trace; silver, 2.4 oz. The iron is high and the manganese low, compared with the theoretical composition of wolframite.

The tungsten ore invariably occurs in connection with enormous andesite dikes which traverse the country in roughly parallel lines. The prevailing country rock is schist and granite. Many of the orebodies do not outcrop, but are found usually 10 to 60 ft. below the surface. Prospecting is easy on account of the peculiar and very pronounced black tungsten stain which pervades the joints and cleavage planes of the veins where tungsten occurs. As the orebody is approached these stains take the form of stringers of wolframite in the dike rock.

It has been demonstrated that the orebodies are reasonably permanent. While the oreshoots are not strictly continuous, no failure of the ore at depth has yet been recorded. The deepest shaft in the district is 400 ft. and several are almost 300 ft. In one of the mines the best ore was encountered at the lowest level, and in all the working mines the ore is practically as good at the bottom as it was nearer the surface.

The treatment of the ore at the mines consists of simple concentration, as follows: (1) Crushing in rock breakers; (2) reducing in stamps or rolls to 10 or 20 mesh; (3) hydraulic classification; (4) concentrating on tables; (5) treatment on Frue vanners or slime tables. In this way 70 to 80 per cent. of the mineral in the ore is saved as concentrate, containing 60 to 70 per cent. tungstic acid. The great difficulty in concentration lies in excessive sliming. With improvements in the method of concentration, it is

probable that the average extraction will soon be increased to 80 or 85 per cent.

There are at present operating in the district one 20-stamp mill, three 10-stamp mills and several 5-stamp mills. All of these are operated in connection with the larger mines, but most of them also buy custom ore. At one of the mills a milling charge of $4 per ton of ore is made, and the concentrates are purchased on the basis of $9.50 per unit for concentrates containing 60 per cent. or more, and $8.50 for concentrates containing between 50 and 60 per cent. tungstic acid. Concentrates containing 50, 60 and 70 per cent. tungstic acid are therefore worth $425, $570, and $665 per ton, respectively. At another mill the ore is bought outright, the price varying from $10 per ton for 3 per cent. ore to $570 per ton for 60 per cent. ore. Ore containing less than 2 per cent. tungstic acid is profitably handled when the mine and mill are controlled by the same interests.

At the Ebony mine at Beaver creek, a tunnel is being driven to encounter the ore at a depth of 400 ft. Recently the "Rogers patent," representing more than 2000 acres of tungsten mineral land, taken up under agricultural patents before the advent of tungsten mining, has been acquired by German interests and will be thoroughly exploited.

Montana.—Promising deposits of scheelite occur at Jardine and are operated by the Kimberly Montana Gold Mining Company. The mineral occurs in bluish white quartz. Sorted, first class ore, is said to run 64 per cent. WO_3. Concentrate from the second class ore contains 70 per cent. tungstic acid.

South Dakota.—The American Tungsten Company is operating a deposit of tungsten ore near Hill City, Pennington county. Associated with the tungsten ore is columbite and tourmaline in white quartz. The veins are mostly enclosed in schist, but in one case the walls are granite. A two-compartment shaft has been sunk 100 ft. and at this depth little or no change is noticeable in the character or value of the ore.

The Aarondale Wolfram Company owns eight claims in the same district and expected to operate late in 1906. A rich strike was reported to have been made at Keystone, Pennington county, where ore (said to run from 35 to 40 per cent. tungstic acid) is found in a well defined ledge which crops out at the surface.

Washington.—The Krupp Gun Company of Essen, Germany, purchased five claims in the Deer Trail district and organized the Tungsten Mine Company to acquire claims in this and other districts. In the Cascade mountains in Okanogan county, discoveries of wolframite were reported and claims were taken by the Tungsten Consolidated Mining and Milling Company. Deposits of wolframite in the Cedar cañon district, Stevens county, 20 miles from Davenport, Lincoln county, were exploited by the

Tungsten Mining Company. An electric power and light plant and a concentrating plant were installed.

Wyoming.—A discovery of tungsten ore is reported to have been made in May, 1906, near Shoshone, in Fremont county, Wyo. The vein is said to be 14 in. wide and of considerable length.

Tungsten Mining in Foreign Countries.

Australia.—Fully 75 per cent. of the Australian tungsten ore is produced in Queensland. In 1906 the output was 710 long tons valued at £56,913. Most of the ore comes from the Kooboora, Bamford and Wolfram Camp groups in the Herberton-Chillagoe district. Occasionally scheelite is found, but the ore is usually wolframite, frequently associated with molybdenite and bismuth. The principal mine is the Nevill of the Kooboora group. Here the ore occurs in chloritic rock instead of in the massive white quartz usual to other mines. The output from this mine in 1905 was 279 tons. The ore from this district is first hand-picked, and then crushed in stamp-mills and jigged.

New South Wales in 1906 produced 132 long tons of wolframite, valued at £9057, and 109 tons of scheelite, valued at £7647. This compares with 86 tons (£7361) of wolframite and 138 tons (£10,122) of scheelite in 1905. Hillgrove is the center of the district.

In the Northern Territory a deposit of wolframite was reported to have been discovered 42 miles from Pine Creek and 200 miles from Palmerston. The ore is said to be associated with copper and the deposit is described as one of considerable importance. In 1906 Tasmania produced 20 long tons of wolframite valued at $7325. There was also a small production in New Zealand.

Portugal.—(By Wilhelm Preus).—Among the most important tungsten mines on the Iberian peninsula are the Panasquiera mines in the province of Beira Baixa, Portugal. The ore deposits consist of quartz beds in Cambrian schist, varying from 0.10 m. to 0.60 m. in thickness, containing wolframite, cassiterite, oxide of iron, iron pyrites, mispickel and mica.

The deposits are well defined. Those in the Panasqueira group are almost horizontal, dipping 5 to 10 deg. to the south; in the Cabeco Piao group they dip 35 deg. to the north. The mineralization is erratic; although faults appear to have had some influence, this is difficult to determine. Nevertheless it has been noted that when three or four faults cross each other forming a triangle or a trapezoid the part of the bed limited by the faults as a rule is richer than other parts of the same bed. A bed recently opened yielded 100 kg. tungsten concentrate per square meter from a space within three faults, and several lumps of pure mineral weighing 30 to 100 kg. were found; another bed found within triangular faults

yielded about 90 kg. tungsten per square meter. Such cases frequently occur.

Exploitation has been going on for about 10 years in the Panasqueira group on the north slope of the Panasqueira Sierra, where an incomplete concentration plant has been put up. There are 15 proved quartz beds on the north slope and corresponding beds have been found on the south slope. Only four of these have been exploited to any extent. The proprietors of the mines have in 10 years exploited, concentrated and sold 600 metric tons of tungsten concentrate (68 to 71 per cent. WO_3) which cost, including freight to Fundao, about \$130, Portuguese currency, per ton. The prices obtained during the last three years have varied between \$5.25 and \$10 per unit (\$365 to \$700 per ton) c.i.f. Continental ports.

About 30,000 sq.m. of quartz beds have been stoped out during these 10 years, yielding 25,000 tons of ore for treatment at the plant. The mill has been slightly improved but about 25 per cent. of the wolframite is still lost in the dumps; the mines are at present producing at the rate of 200 to 250 tons of concentrate (68 to 71 per cent. WO_3) per year.

In the Cabeco Piao group exploration has proved the existence of 10 beds, and in four of these about 60,000 sq.m. have been laid bare with very little work; these average 25 kg. of wolframite (68 to 71 per cent. WO_3) per square meter. Simultaneously some development was done and during two years about 50 tons of wolframite were sold. This ore was obtained with a very incomplete washing plant.

Spain.—La Sorpresa tungsten mine[1] has been exploited since 1903 by Koch & Villar. It is situated in the province of Cordoba in Las Cabezas mountains at an altitude of 710 m. A good wagon road leads from Marmolejo, 22 km. distant. The deposit lies at the contact between Cambrian slates and granite in which there are numerous veins of white quartz interspersed with stringers of tungsten mineral. They pass uninterruptedly from the slate to the granite. The thickness of these stringers varies from 20 to 140 cm. The mineral zone is a white and compact quartz, which is sufficiently rich to be worked at all points. Both wolframite and scheelite occur in the deposit.

The mines are worked by open cuts. After sorting and cobbing, the mineral is calcined in a small kiln, and then is crushed and washed. The concentrate varies between 68 and 74 per cent. WO_3 in the case of the wolframite, and 69 to 80 per cent. in the case of scheelite. The concentrate from ore in which the two minerals are so intimately mixed that they cannot be separated by sorting, constitutes another class whose tenor in WO_3 is approximately 75 per cent. The mines employ 200 men and produce 10 tons of washed mineral per month for export to Hamburg, Germany.

[1]*Revista Minera*, Mar. 8, 1906.

United Kingdom.—The production of tungsten ore in the United Kingdom was 251 long tons in 1906 against 136 tons in 1905 and 161 tons in 1904. The ore is found in the tin mines of Cornwall, principally as wolframite, although small amounts of scheelite and tungsten-ocher occasionally occur. The chief producing mines are: East Pool, South Crofty, Clitters United, and Drakewalls. Recently considerable attention has been devoted to the separation of cassiterite and wolframite in mixed ores, for which the Wetherill magnetic separator has been successfully employed.

URANIUM.

Among steel hardening metals, uranium is distinctly subordinate in importance to vanadium, chromium, nickel and tungsten. It has, however, other uses, notably in the manufacture of incandescent gas mantles, where its presence is considered to improve the properties of thorium, cerium and zirconium salts applied for the same purpose. The chief minerals which yield uranium oxide are pitchblende and carnotite. In the United States during 1906 pitchblende was produced only from a few scattered prospects in Gilpin county, Colorado, where the mineral occurs with auriferous sulphide ores in the Kirk and Wood mines. One carload shipment was made from the Kirk mine in 1906. The mineral is shipped to New York whence it is exported to Hamburg, Germany.

Uranium ore, as pitchblende, is worth $2 per lb. Carnotite ore (low-grade sandstone), with 3 to 5 per cent. of uranium oxide, U_3O_8, calls for $5 per unit, or at 40 per cent., $200 per ton. But the high-grade hand-sorted ore calls for $1.50 per lb. of contained uranium oxide; and at, say, 50 per cent. oxide, figures $1500 per ton.

VANADIUM.

The use of special steels containing vanadium in combination with other steel hardening metals, such as chromium and nickel, has increased very largely, especially in the automobile business. These vanadium steels contain from 10 to 20 per cent. vanadium and their production at the present time is practically all concentrated into English hands, where 80 per cent. of the total production is taken by motor car and motor omnibus manufacturers.

During 1906 a discovery of uranium and vanadium minerals was made by A. H. Caywood and associates on Skull creek in Rio Blanco county, Colorado. A small shipment amounting to 10 tons of mineral was made to Denver for experimental purposes. A concentrating plant will be erected at the deposit if the test upon the mineral warrants such a step. The main vein of this deposit is in sandstone and it shows two streaks of mineral, one 20 in. and the other 30 ft. wide. The Vanadium Alloys Company erected a plant largely for experimental purposes at Newmire on the San Miguel river, 12 miles below Telluride, Colo., and this was put into operation in April, 1906. The plant was built especially for the treatment of the deposits of vanadium ore lying in the bluffs on both sides of the river and along Bear and Leopard creeks for a distance of about 15 miles.

Prices.—Vanadium ores are sold upon the basis of their content of vanadium oxide, $Vd_2\ O_5$ of which they must contain from 3 to 5 per cent. at least. Ore of this character is rated at $5 per ton f.o.b. cars.

Dr. Leon Guillet[1] describes the Herrenschmidt method of extracting vanadium from ores as follows: The ore is vanadinite from the Santa Marta mines of Spain, and carries 14 per cent. vanadic oxide and 50 per cent. lead.

This ore is smelted in a reverberatory furnace, with carbonate of sodium and coal. Metallic lead carrying all the silver is formed, together with a slag containing vanadates, aluminates, silicate of sodium and oxide of iron. To render this slag soluble in water, it is melted in a reverberatory furnace, and, after melting, is blown with air until the vanadium is completely oxidized. The mass is then poured into boiling water to granulate it. After three leachings the residues consist chiefly of alumina, silica, and oxide of iron containing not more than 2 per cent. of vanadic acid. As, per ton of ore treated, there remains but 280 kg. of this residue, consequently 95 per cent. of the vanadic acid contained in the ore is extracted in solution.

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[1] Address to the Iron and Steel Institute, Sheffield, Sept. 26, 1905.

The sodium vanadate solution does not contain any alumina, and even if aluminate of sodium be added to the solution of vanadate, the alumina is precipitated.

In order to eliminate the silica contained in the impure sodium vanadate, a certain quantity of the vanadate is evaporated to a syrup, and to it is added sulphuric acid of 66 deg. B.; vanadic acid and sulphate of sodium is formed, but the whole of the vanadium is not precipitated, about 10 per cent. of the original content of the solution remaining. The mixture is therefore placed in contact with the solution of impure vanadium, and, after energetic stirring, it is taken to the filter press. The whole of the silica contained in the silicate of sodium is precipitated in this manner, while the vanadium remains in solution.

The alkaline solution of vanadate, thus freed from silica, contains no other impurities. It is concentrated, and the vanadium is precipitated by an excess of sulphuric acid. It is evaporated to dryness to drive off the excess of sulphuric acid, and then washed. Vanadic acid of 92 to 95 per cent. purity is thus obtained, with 5 to 8 per cent. of foreign matter.

ZINC.

By W. R. Ingalls.

The production of virgin spelter in the United States in 1906 was 225,494 short tons, that figure being the total of reports received from all the producers. The details of the production in 1906, compared with previous years, are as follows:

PRODUCTION OF SPELTER IN THE UNITED STATES.

States.	1899.	1900.	1901.	1902.	1903.	1904.	1905.	1906.
Colorado	....	....	....	....	877	4,906	6,599	6,260
Illinois (a)	49,290	37,558	44,896	49,672	49,526	47,607	45,357	48,238
Kansas	55,872	57,276	74,270	87,321	87,406	103,721	114,948	129,741
Missouri	15,710	20,138	13,083	10,548	9,894	12,056	11,800	11,088
South and East (b)	8,803	8,259	8,603	10,698	10,799	13,513	23,044	30,167
Total tons of 2000 lb.	129,675	123,321	140,822	158,239	158,502	181,803	201,748	225,494
Total tons of 2240 lb.	115,781	110,028	125,734	141,283	141,520	162,324	180,132	201,343
Total metric tons	117,644	111,794	127,751	143,552	143,792	164,921	183,014	204,548

(a) Up to 1903, inclusive, includes also the production of Indiana. (b) New Jersey, Pennsylvania and Virginia, and since 1903) West Virginia.

The statistics for 1906 include only virgin spelter, i.e., spelter produced from ore. There is an additional quantity, estimated at 3000 tons, from the marketing of scrap zinc and the resmelting of galvanizers' dross and other waste products. Some of these are converted into spelter of very good quality. Also there is a considerable exportation of dross to the European smelters.

PRODUCTION OF ZINC IN EUROPE AND AMERICA. (a)
(In metric tons.)

Year.	Austria	Belgium.	France.	Germany.	Holland.	Italy.	Russia	Spain.	United Kingdom.	United States.	Totals.
1896	6,888	113,361	45,585	153,082	4,770	*Nil.*	6,257	6 133	25,278	70,432	421,786
1897	6,236	116,067	38,067	150,739	6,600	250	5,868	6,244	23,805	91,070	444,946
1898	7,302	119,067	37,155	154,867	6,700	250	5,664	6,031	28,387	103,514	468,937
1899	7,192	122,843	39,274	153,155	6,235	251	6,331	6,184	32,322	117,644	491,331
1900	6,742	119,315	36,305	155,799	6,845	547	5,963	5,611	30,207	111,794	465,438
1901	7,558	127,170	37,600	166,283	7,855	511	6,090	5,354	29,877	127,751	516,049
1902	8,309	124,780	36,282	174,927	9,910	485	8,280	5,569	40,244	143,552	552,338
1903	8,949	131,740	27,462	182,548	11,515	126	9,901	5,134	44,110	143,792	560,017
1904	9,159	137,323	43,196	193,058	12,895	189	10,607	5,887	46,218	164,921	623,453
1905	9,210	143,165	44,075	198,208	13,550	(b)	7,520	5 500	50,125	183,014	654,367
1906	10,780	152,461	(c)48,286	205,691	14,650	(b)	9,610	(c)5,500	52,587	204,548	704,113

(a) From the official statistics of the various Governments, except 1905 and 1906, for which years the figures reported by Henry R. Merton & Co. have been used where the official statistics were unavailable. In addition to the production reported in this table, Australia produced 286 long tons in 1903; 299 in 1904; 544 in 1905; and 1008 in 1906. (b) Included in Austria. (c) An approximate separation of the total which is reported for "France and Spain."

EXPORTS OF ZINC ORE AND ZINC OXIDE FROM THE UNITED STATES. (a)

Year.	Ore.			Oxide.		
	Short tons.	Value	Value per ton.	Short tons.	Value.	Value per ton.
1896	(b) 2,324	$47,408	$20.40	(c)	....	....
1897	9,251	211,350	22.85	1,859	$104,140	$56.02
1898	11,782	299,970	25.50	3,925	252,194	64.25
1899	28,221	725,944	25.90	5,343	366,598	68.61
1900	42,062	1,134,663	26.98	5,656	496,380	87.76
1901	44,146	1,167,684	26.45	4,561	393,259	86.22
1902	55,733	1,449,104	26.00	5,358	433,722	80.93
1903	39,411	987,000	25.04	7,215	578,215	80.14
1904	35,911	905,782	25.22	8,157	628,494	77.05
1905	30,946	848,451	27.41	11,280	810,203	71.83
1906	27,720	733,300	26.45	15,578	1,149,297	73.78

(a) In addition to the exports of ore, 15,887 short tons of zinc dross (galvanizers' waste) were exported in 1906, against 5318 tons in 1905. (b) Includes oxide. (c) Included in ore.

IMPORTS OF ZINC AND ZINC OXIDE INTO THE UNITED STATES.
(In pounds.)

Year.	Sheets. Blocks, Pigs and Old.		Manufactures	Total Value.	Oxide.	
	Amount.	Value.			Dry.	In Oil.
1896	856,044	$25,904	$15,728	$41,632	4,572,781	311,023
1897	2,557,341	95,883	19,431	115,314	5,564,753	502,357
1898	2,742,357	109,624	13,448	123,072	3,342,235	27,050
1899	2,985,463	151,956	14,800	166,756	3,012,709	41,699
1900	2 013,196	97,772	36,836	134,608	2,618,808	38,706
1901	775,881	30,920	42,643	73,563	3,199,778	128,198
1902	1,238,091	46,713	37,191	83,904	3,271,385	163,081
1903	728,614	30,900	18,938	49,838	3,487,042	166,034
1904	933,474	44,326	11,918	56,244	2,585,661	224,244
1905	1,042,081	51,052	12,390	63,442	3,436,367	342,944
1906	4,407,481	253,310	17,385	270,695	4,191,476	292,538

EXPORTS OF DOMESTIC SPELTER FROM THE UNITED STATES. (a)

Year.	Plates, Sheets,	Pigs and Bars.	Wares.	Total Value.
	Short Tons.	Value.	Value.	
1896	10,150	$1,013,620	$51,001	$1,112,029
1897	14,245	1,356,538	71,021	1,743,049
1898	10,499	1,033,959	138,165	1,724,188
1899	6,755	742,521	143,232	1,978,295
1900	22,411	2,217,963	99,288	2,317,251
1901	3,390	228,906	82,046	310,952
1902	3,237	300,557	114,197	414,754
1903	1,521	163,379	71,354	234,733
1904	10,073	1,094,490	117,957	1,212,447
1905	5,516	682,254	159,995	842,249
1906	4,670	583,526	204,269	787,795

(a) There is also a comparatively insignificant re-export of foreign-made spelter and zinc wares.

Zinc Oxide.—The production of zinc oxide (including zinc-lead pigment) in the United States in 1906 was 77,800 tons, against 72,603 tons in 1905. It is no longer possible to report the production of zinc oxide alone, because certain manufacturers now produce both zinc oxide and zinc-lead pigments, and a further analysis of the statistics would disclose individual business.

There is a small production of zinc oxide by the combustion of spelter, but the great bulk of the American product is made directly from ore. If the zinc content of this oxide be added to the spelter production, the United States is by far the largest producer of zinc in the world. Early in 1906 the Ozark Smelting and Mining Company completed and put into operation a new oxide plant at Coffeyville, Kan., making a total of six works in the United States engaged in the manufacture of oxide directly from ore and one in the manufacture from spelter.

PRODUCTION OF ZINC OXIDE IN THE UNITED STATES. (a)

Year.	Quantity. Short Tons.	Quantity. Metric Tons.	Value. Totals.	Value. Per Short Ton.	Year.	Quantity. Short Tons.	Quantity. Metric Tons.	Value. Totals.	Value. Per Short Ton.
1897.......	26,262	23,285	$1,686,020	$64.26	1902.....	52,730	46,929	$4,023,299	$76.30
1898.......	32,747	29,708	2,226,796	68.00	1903.....	59,562	54,034	5,005,394	83.69
1899.......	39,663	35,982	3,331,692	84.00	1904.....	59,613	54,081	4,523,414	75.88
1900.......	47,151	42,775	3,772,080	80.00	1905.....	72,603	65,859	5,808,240	80.00
1901.......	46,500	42,266	3,720,000	80.00	1906.....	77,800	70,573	6,257,361	80.43

(a) The figures for 1905 and 1906 include zinc-lead pigment, which was not included in the statistics for previous years.

New Smelting Capacity.—Several new plants were in course of construction in 1906, but the only one to go into operation was the works of the Mineral Point Zinc Company, at DePue, Ill. Among the others, the works of the Lanyon-Starr Smelting Company, at Bartlesville, Ind. Ter., and the new works of the Caney Zinc Company, at Deering, Kan., went into operation early in 1907. The works of Hegeler Bros., at Danville, Ill., which have been under construction for a long time, are also to go into operation in 1907. Several other new works are planned for 1907, but only one of them is likely to be in operation before the end of the year. In 1906 another one of the old smelters at Pittsburg, Kan., was put in operation, making two now running at that place.

Among the older companies, the Caney Zinc Company added a new furnace of 300 retorts to its plant at Caney, Kan. The Cockerill Zinc Company added three blocks of furnaces, with a total of 672 retorts, to its plant at Pittsburg, Kan. The Matthiessen & Hegeler Zinc Company, of LaSalle, Ill., added 300 retorts, while the Illinois Zinc Company, of Peru, Ill., added 40. The United Zinc and Chemical Company, of Iola, Kan., added two furnaces, each having 240 large retorts (muffles). The Sandoval Zinc Company, of Sandoval, Ill., added two furnaces. The Mineral Point Zinc Company, of DePue, Ill., started with two furnaces, having an aggregate of 1600 retorts. In the East the New Jersey Zinc Company added to its Palmerton works six new furnaces with a total of 1200 retorts. The total of the additions in 1906 was 4560 retorts and 480 muffles. Conse-

quently there was a large increase in the smelting capacity, although only one new plant went into operation.

By-Product Lead.—A feature of considerable interest in connection with the American zinc industry is the increasing production of lead obtained as a by-product from the smelting of zinc ore. In 1906 smelters in Colorado produced 542 tons of pig lead from residues received from zinc smelters in Kansas, and I estimate that they must have produced also 600 tons of lead from residues smelted in Colorado. Moreover, one smelter produced 75 tons of pig lead as a by-product in the refining of spelter.

Consumption of Spelter.—The stock of spelter in the hands of the smelters at the beginning of 1906 amounted to 4000 tons. The production of virgin spelter during the year was 225,494 tons. The imports amounted to 2203 tons. The total supply was consequently 231,697 tons. The exports of spelter during the year were 4670 tons. The stocks in the hands of smelters at the end of the year amounted to 4550 tons. The domestic consumption of virgin spelter was consequently 222,477 tons. The figures for 1906, in comparison with those of 1905, are given in the accompanying table:

CONSUMPTION OF SPELTER IN THE UNITED STATES.
(In tons of 2000 lb.)

	1905	1906
Stock, Jan. 1	6,500	4,000
Production	201,748	225,494
Imports	521	2,203
Total supply	208,769	231,697
Exports	5,515	4,670
Stock, Dec. 31	4,000	4,550
Consumption	199,254	222,477

In addition to the production of virgin spelter there is a considerable production of spelter from the resmelting of zinc dross obtained from galvanizing. This, together with scrap that is remarketed, is estimated to amount to about 3000 tons per annum. Consequently, the total consumption of spelter in the United States in 1906 may be assumed to have been about 225,500 tons.

In order to distribute the consumption according to the principal purposes, reports were obtained from the consumers. It is never possible to check closely in this way the computed consumption, because there are many small consumers whose use of metal is nevertheless large in the aggregate, from whom it is impossible to obtain reports. The reports actually received for 1906, including a few trade estimates, which could be safely considered as close approximations, footed up to a total of 208,479 tons. In the case of the galvanizing industry, wherein the consumers are chiefly large concerns, practically complete reports were received,

which reduces the uncertainty in this investigation to the matter of the consumption of spelter for brass-making, sheet-zinc rolling and miscellaneous purposes. The consumption of spelter for the desilverization of lead is estimated on the basis of the desilverized lead produced, a consumption of 0.8 per cent. of spelter being reckoned. The statistics of consumption according to use are summarized in the accompanying table.

USES OF SPELTER IN THE UNITED STATES.
(In tons of 2000 lb.)

Purpose.	1905	1906	1905	1906
Galvanizing	100,000	124,000	50%	55%
Brass-making	52,000	57,000	26	25¼
Sheet zinc	34,000	36,000	17	16
Lead desilverization	2,400	2,500	1¼	1
Other purposes (*a*)	10,854	6,000	5¾	2¾
Total	199,254	225,500	100	100

(*a*) The apparent falling off in the consumption of zinc for "other purposes" in 1906 is explained by a more complete itemization of the consumption in 1906; in other words, there was probably more spelter used for brass-making in 1905 than the above table shows.

It will be observed that the largest percentage of increase in 1906 was in the galvanizing business. The consumption of spelter for brass-making shows an increase, but the percentage with respect to the total consumption was a little less than in 1905. Incidentally this figure gives a rough idea of the consumption of copper for brass-making. If 57,000 tons of spelter were used for that purpose, there must have been employed about 114,000 tons of copper for the same purpose. This figure corresponds to about 33¾ per cent. of the domestic consumption of new copper in 1905. The estimated consumption of copper for brass-making in 1905 was 104,000 tons, which was 34 per cent. of the domestic consumption of new copper that year.

Production of Zinc Ore in the United States.

The production of zinc ore in the United States in 1906 was 905,175 tons, against 795,698 tons in 1905, and 693,025 tons in 1904. The statistics for 1906 are not strictly comparable with those for 1905, because while the latter omitted the zinc ore used for the production of zinc oxide and zinc-lead pigment, the statistics for 1906 include zinc ore consumed for all purposes. These statistics are based chiefly on reports from the smelters of the ore which they actually received, and partially on reports of miners of their shipments to smelters. Consequently, they represent the concentrated marketable product, and show the actual supplies of raw material that were available to the smelters. In the case of one smelter which operates a concentrating mill in connection with its works a reduction in the figures reported has been made in order to put all the returns upon a uniform basis. Discrepancies among the statistics of zinc ore production

reported by various authorities are to be explained by differences in the bases, the production of crude ore (which must be concentrated or separated into zinc ore and lead ore) being frequently confused with the final product of marketable zinc ore, in which only is the smelter interested.

It will be observed from the table that a large part of the production of ore in 1906 was due to New Jersey, of which the product is used to only a small extent for making spelter in the United States, the bulk of it being employed for the manufacture of oxide and for export to Europe as a spelter ore. Omitting the New Jersey production, it appears that the Western spelter and oxide manufacturers received 590,000 tons of ore in 1906. The production of the Joplin district accounts for about 137,000 tons of spelter, against 124,000 tons in 1905. Consequently that district more than held its own in so far as percentage of the production of Western spelter is concerned.

PRODUCTION OF ZINC ORE IN THE UNITED STATES.

State.	1904		1905		1906	
	Tons.	(g) Value.	Tons.	(g) Value.	Tons.	(g) Value.
Arkansas	(e) 1,900	$66,000	2,200	$96,000	4,200	$168,000
Colorado	(a) 94,000	940,000	105,500	1,529,750	114,000	1,824,000
Idaho	*Nil.*	....	1,700	37,400	2,150	45,150
Kentucky	(d) 958	10,538	(d) 414	6,624	975	34,125
Missouri-Kansas	(b)273,238	9,692,160	(b)258,500	11,455,280	(b)283,500	12,219,675
Montana	*Nil.*	....	2,000	25,000	4,900	98,000
Nevada	*Nil.*	....	*Nil.*	....	7,080	70,800
New Mexico	(e) 21,000	168,000	17,800	222,500	30,000	360,000
New Jersey	(d)280,029	560,058	(d)361,829	723,658	404,690	809,380
Utah	*Nil.*	....	9,265	120,445	10,700	214,000
Wisconsin	(c) 19,300	598,300	32,690	1,307,600	42,130	1,390,290
Others	(a) 2,600	36,400	(f) 3,800	72,200	(h) 850	17,000
Totals	693,025	$12,071,456	795,698	$15,596,457	905,175	$17,250,420

(a) Estimated. (b) Production of Joplin district, plus output of southeastern Missouri, the latter as reported by the State mine inspector. (c) According to H. F. Bain, "Contributions to Economic Geology," 1904. (d) Report of State Geologist; crude ore. (e) Partly estimated. (f) Arizona, Nevada, Illinois, Iowa, Tennessee and Virginia. (g) Values are estimated, no direct reports having been received; they are reckoned, in all cases f. o. b. mines, on the basis of the average price for spelter in each year and the average grade of ore. (h) Indian Territory, Tennessee, Arizona and California.

IMPORTS OF ZINC ORE INTO THE UNITED STATES.

Source.	1904	1905	1906
British Columbia	2,100	8,561	600
Mexico	?	(a)32,164	(a)88,900
Totals	?	40,725	89,500

(a) The actual tonnage of ore imported was somewhat greater than this figure, but it included some mixed ore, which for statistical purposes has been reduced to the zinc ore equivalent. This table is based on reports from the smelters of the ore received by them from these countries.

A question of great importance to the American zinc industry in 1906 was the interpretation of the Dingley tariff as relating to zinc ore. By an order of the Treasury Department, Feb. 10, 1906, all kinds of zinc ore were made subject to a duty of 20 per cent. *ad valorem.* The smelting

interests carried the case before the Board of General Appraisers, which late in the year decided that the term "calamine" included not only all forms of the carbonates and silicates of zinc, but also that sulphide ore was admissible into the United States, free of duty, except for 1.5c. per ℔. upon its lead content. The Government appealed from this decision to the Circuit Court, from which a decision is expected shortly. Whatever be the decision of the Circuit Court, the case is likely to be taken to the Circuit Court of Appeals, from which a decision can hardly be expected before the end of the year.

In the meanwhile, importations of zinc ore have been made on a large scale, as is shown in the accompanying statistics. As to the status of zinc silicate ore, there was, of course, no question, that being calamine according to all authorities, and calamine being specifically on the free list. Imports of carbonate ore have been made by the smelters giving bond to pay the duties in the event the case should be decided against them. Feeling the utmost confidence in the soundness of their case, they have not hesitated to import large quantities of that class of ore. The question as to zinc sulphide ore was more doubtful, and consequently the smelters imported but little of that class, which accounts for the great falling off in the importations from British Columbia.

Arkansas.—Production of zinc ore in this State showed a gratifying increase in 1906, the output in that year having been 4200 tons, against 2200 tons in 1905. According to R. W. Willett, ore buyer for the Sandoval Zinc Company, the production of Arkansas for 1904 was 2090 tons; and 1385 tons in 1903. Geo. I. Adams, of the Geological Survey, reports the production as 1000 tons in 1902, and 500 tons in 1901.

California.—This State appeared in 1906 for the first time as a producer of zinc ore, the Western Zinc Company, of San Francisco, having operated the Silverado mine in Orange county. The company has works at San Francisco, at which it manufactures the sulphate and chloride of zinc, and contemplates the production of spelter. In the meanwhile it is planned to export the surplus of ore to Europe. The company has a concentrating mill at the mine. The ore is blende and pyrites, the latter mineral being gold-bearing.

At the Afterthought mine, in Shasta county, some experimental work was done on the production of zinc oxide from the mixed sulphide ore, which is mined and smelted there primarily for copper. The mines of Shasta county, especially the Bully Hill mine, contain large bodies of zinky ore of similar character. The association of the sulphides is extremely intimate, rendering mechanical separation highly difficult, if not impossible.

A little development work was done on a zinc prospect near Wawona at the western side of Yosemite park, but the deposit is not now available on account of lack of transportation facilities, which, however, will proba-

bly be supplied in the near future. The ore at this place is a sulphide. A small amount of development work was done on some zinc prospects, near Victorville, San Bernardino county, but no shipments were made. The ore is said to be of good quality, but of small development.

Colorado.—As a producer of zinc ore, this State ranks third. Its importance as compared with New Jersey is much greater than appears from the statistics of ore production, because the ore of Colorado assays nearly twice as high in zinc as the New Jersey ore. As in previous years, the largest part of the output of Colorado was obtained from Leadville. According to the Leadville *Herald-Democrat*, the production of that district in 1906 was 228,565 tons, of which the Iron Silver Mining Company produced 111,500 tons, the Western Mining Company, 51,170 tons, and the Ibex Mining Company, 24,870 tons. These figures refer to the production of crude blende-galena-pyrites ore before concentration or separation.

Besides Leadville, zinc ore was produced at Georgetown, Red Cliff, Creede, Rico and Kokomo, and at several places in Chaffee and Pitkin counties, but their total was comparatively small. According to the State commissioner of mines the zinc content of the ore produced in Colorado in 1906 was 42,744 tons, of which the Leadville ore contained 35,100 tons.

The United Rico Mines Company is now engaged on plans to develop the extensive deposits of mixed sulphide ore at Rico, where the Stallman-Germer process of separation—a species of flotation process—is to be introduced.

During 1906, the Iron Silver Mining Company, of Leadville, produced 145,120 tons of ore, which contained 2043 oz. of gold, 425,050 oz. of silver, 69,477 ℔. of copper, 4,542,938 ℔. of lead and 61,738,134 ℔. of zinc. The total cost of production was $574,681, divided as follows: Labor, $367,739; mine supplies, $62,411; mine timber, $32,849; insurance, $5432; legal expenses, $1927; New York and Detroit expenses, $6908; taxes, $12,851; expenditures on the Tucson mine, $66,754; expenditure on the Dome mine, $17,810. The Moyer mine was the chief producer, its output being 139,050 tons, of which 97,192 tons was zinc ore. The average value of all the ore produced by this mine was $7.53 per ton, and the cost of mining, including all incidental expenses, but not the expenditures on the Dome and Tucson mines, was $3.53 per ton.

Idaho.—The production of zinc ore in this State was made chiefly by the Success mine in the Cœur d'Alene. The only other production was in the Wood River district, where the output was insignificant.

Kentucky.—The output of this State showed a good increase in 1906, although the total is still small. However, a further increase is to be expected, inasmuch as a successful method of separating the mixed blende-galena-fluorspar ore has been developed by W. M. Sanders, who has organized the Sanders Separating Company, which has a small plant at Marion.

The system of separation is a form of flotation process, which differs from that in use at Broken Hill in that no acid is employed in the bath, the agent in the latter being aluminum sulphate. Moreover, instead of effecting the separation in a deep pointed vat, a shallow pan, with stirrers, is employed.

Missouri and Kansas.—With the exception of a small quantity of calamine mined in southeastern Missouri, the entire production of these States is obtained from the Joplin district, which showed a great increase in 1906. Activity was naturally stimulated by the high price for ore prevailing throughout the year, but the increase in production is largely attributable to the improvements in the method of milling. Whereas, only a year or two ago, the average extraction of mineral from the crude ore was probably not more than 67 per cent., at present it is probably in the neighborhood of 75 per cent. This alone is sufficient to account for a large part of the increase in production in 1906.

Another important factor in increasing the production in the Joplin district is the recent erection of mills of much larger capacity than formerly. Whereas the standard mill used to be of 10 tons nominal capacity per hour, mills of 20 to 30 tons capacity are now becoming common. The new mills are equipped with rolls of larger size (up to 42 in. diameter) and have in addition to the jigs, slime tables of the Wilfley, Standard and Neosho patterns. By virtue of these improvements, the tailings now contain less than 1 per cent. of mineral, which accounts for the higher extraction noted above. The improvements in these particulars have been made possible by the mining of the "sheet" ground, which comprises extensive orebodies, justifying the installation of larger and more expensive plants than formerly. In other words, mining in the Joplin district is being put upon a more permanent basis than ever before in its history. It is estimated that approximately 40 per cent. of the present output is obtained from sheet ground, some of the latter being mined which yields as little as 3 per cent. of mineral.

The mining of such low-grade ore, of course, has been rendered possible only by the high prices obtained for the product. Although the introduction of improved mining and milling machinery, and the prosecution of operations on a larger scale than formerly, have led to many economies, which have been helped also by the piping of natural gas into the district and the general use of the latter, either under the boilers or in gas engines, the advantages gained in those respects have been largely offset by the increased cost of labor. Drill runners now receive $3 per 8 hours, while muckers get $2.50 per 8 hours. Jig-men obtain $3.50 to $4 per 10 hours, while surface laborers in general receive $2.50 per 10 hours. These are heavy increases over the rate of a few years ago, and even under the present conditions, labor in the district is scarce. Steel and other mining supplies also commanded advanced prices in 1906. The only supplies

which did not increase in cost were coal, which was obtainable at $1.75 per 2000 lb. of mine run; and dynamite, which cost $4.25 per box of 40 per cent. grade. The cost of delivering ore to the railway cars was somewhat reduced, because of the further extension of railway switches to the mines. On the other hand, royalties are somewhat higher than a little while ago, the present terms to first lessees being 10 to 15 per cent. on blende, probably averaging more nearly the latter figure than the former. The miner is assisted, however, by a somewhat larger yield of lead ore from the sheet ground, the ratio of lead ore to zinc ore being in the neighborhood of 1 to 6.

SHIPMENTS OF ZINC AND LEAD ORE FROM THE JOPLIN DISTRICT.
(In tons of 2000 lb.)

Year.	Zinc Ore.	Lead Ore.	Year.	Zinc Ore.	Lead Ore.
1894.......	147,310	32,190	1901........	258,306	35,177
1895.......	144,487	31,294	1902........	262,545	31,625
1896.......	155,333	27,721	1903........	234,873	28,656
1897.......	177,976	30,105	1904........	267,240	34,362
1898.......	234,455	26,687	1905........	252,435	31,679
1899.......	255,088	23,888	1906........	278,930	39,189
1900.......	248,446	29,132			

AVERAGE MONTHLY PRICE OF ZINC BLENDE ORE AT JOPLIN MO.
(Dollars per 2000 lb.)

Year.	Jan	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct	Nov.	Dec.	Year.
1900.....	30.23	29.36	28.45	28.42	26.92	25.00	24.23	25.67	24.25	24.25	24.45	25.40	26.50
1901.....	23.72	23.96	23.70	24.58	24.38	24.22	24.68	23.88	21.63	21.63	26.15	28.24	24.21
1902.....	26.75	27.00	28.00	28.85	29.23	34.10	34.37	32.50	33.58	33.58	32.10	29.25	30.73
1903.....	31.50	32.50	35.75	37.75	36.60	36.50	36.00	36.00	34.40	34.40	30.75	30.00	34.44
1904.....	32.12	34.00	36.00	36.40	34.63	32.62	35.00	37.00	40.40	40.00	44.25	46.13	37.40
1905.....	51.94	53.65	47.40	43.93	43.74	40.75	43.00	50.24	46.80	49.37	50.37	47.67	47.40
1906....	49.33	49.25	45.60	44.00	41.50	44.20	43.88	44.38	43.20	42.50	44.43	45.55	44.82

(By Jesse A. Zook.)—In writing at the close of 1905 it was noted that 1906 "should show a marked reduction in cost per ton of concentrate. unless the demands of labor should absorb the saving." Additional expense for labor prevented any marked reduction in cost, but it was the exigencies arising from a scarcity of labor rather than the demands of labor. The prospect of trouble from organized labor, manifest at the close of 1905, was swept away by the general prosperity of the district and a higher wage scale resulted from the competition for labor.

The rainfall of 1906 was largely confined to the period between the latter part of April and the first of July, and was so distributed as to cause no serious obstacle to mining. Subsequent to July the year was comparatively dry and exceptionally good weather prevailed for mining, development and prospecting. The principal drawback was the lightness of demand for ore throughout the spring and summer.

The mines of the Chitwood group, in Joplin, which held up the Joplin

product for several years, were not such heavy producers in 1906, the points of production changing to the west and southwest, and east and southeast of Joplin, the largest development of the year being on the southwest. Oronogo, Webb City and Carterville have become merged by the development of new mines in the south part of the corporate limits of Oronogo, the output of which mines has been classed in Webb City-Carterville, taking Center creek as the natural dividing line, instead of the line of municipal incorporation of Oronogo, which extends nearly a mile on the Carterville side of the river. The Porto Rico group maintained the equilibrium of the worked-out mines of Duenweg. Alba and Neck City show the heaviest increase of shipments on the Missouri side of the line, while the Galena-Empire mines held up the Kansas side with a material increase.

The record of the last three years has fairly demonstrated that the price for zinc concentrate cannot be held below $45 per ton of 60 per cent. zinc to insure a permanently steady output. The coming year (1907) should bring economies in labor-saving machinery in every manner practicable, to offset the scarcity of labor felt during 1906. It begins with several mills in operation that will change many fixed ideas regarding size, economy in operation and savings.

During 1906 many mills were equipped with tables for handling slimes. The best form is yet to be determined. The same is evident in the installation of gas engines. There are too many cheap, makeshift ones introduced. While the idea is slowly changing to the installation of heavier and better machinery, the peculiar prospecting and development conditions of the district require only a cheap plant, and the same cheapness is too often followed in subsequent mill installation.

Zinc concentrate sold at $53 per ton the first week of 1906, advancing to $54 the following and succeeding four weeks, dropping successively to $53, $52, $51, and $50 for two weeks, then to $49 the third week in March. March closed at $52 and April opened at the same price, followed by a decline to $49, to $47, and to $45. All through May $44 was the highest. June showed a fluctuation from $45 at the first to $50 three weeks later, declining to $49, $48 and $47, up to $48 for four weeks, then $48.50 and $49, returning to $48 in the first fortnight of September, dropping to $46 and closing the month at $47. For three weeks of October the price was $46, closing at $48. The first half of November it was $48, the third week $49.50, closing at $48.50. The succeeding weeks of December the price was $49, $49.70, $49.50, $50 and closed the year at $49.50. The highest price paid for first-grade zinc concentrate was $54 in January and February, buyers inaugurating a decline immediately upon the decision of the Treasury assessing an *ad valorem* duty on importations, from which the price continuously declined to the low point of $44 in May.

The method of buying zinc ore has now been changed to an assay basis

strictly, with deduction for actual moisture carried in the concentrate. The highest assay basis price during 1906 was $50 in January and February; the lowest $40 the last week of April, all of May and the first week of June.

Lead concentrate began 1906 at $80.50 per ton, advancing to $81, declining to $80 and closing January at $76. The first half of February it was $78.50 and the last half down to $72. March recorded $73, $76.50, $79, $80, and closed at $78. April fluctuated from $76 to $78.50; May from $79 to $83; June from $83 to $78; July from $78 to $75; August from $78.50 to $82; September from $84 to $82.50; October from $82 to $84.50; November from $83 to $85.50, the price advancing to $87 the last week of 1906.

The highest price for lead concentrate was $87, at the close of December; the lowest price for first grade was $72 in February; the average, all grades, eleven months, was $77.78, as against $62.12 in 1905.

Montana.—The production of zinc ore in this State showed an increase in 1906, but not so much as was expected, because the separating works of the Montana Zinc Company, at Butte, were destroyed by fire in June, after which no zinkiferous ores were treated in the State, except at Corbin, where an experimental plant was put in commission in October to test the Baker-Burwell process. Operations were suspended at the end of the year.

Nevada.—This State appeared in 1906 for the first time as a producer of zinc ore, its output being chiefly calamine from the Potosi mine in the southwestern part of the State. It is testimony to the remarkable development of the zinc industry west of the Rocky Mountains that it is possible to work a mine of comparatively low-grade ore at so remote a place. Zinc ore was also produced by the Nevada Commonwealth Mining and Milling Company, of Washoe, Washoe county, which has a mixed sulphide ore carrying lead, zinc, copper, arsenic and gold. A mill of 150 tons capacity per day was erected, but no shipments of zinc ore were made in 1906; they were begun in 1907, the ore going to Iola, Kan.

New Jersey.—The entire output of zinc ore credited to this State is from the Franklin mine of the New Jersey Zinc Company. The statistics represent the production of crude ore. It is, however, utilized entirely for one purpose or another, there being no waste except an insignificant proportion of calcite tailing, but even that is a marketable product. The ore is separated magnetically into a franklinite product which is employed for the manufacture of zinc oxide and spiegeleisen, and a willemite product, which is used for the production of spelter, a portion of this ore being exported to Europe.

New Mexico.—The increase in the production of this Territory was due chiefly to the more extensive exploitation of the Kelly and the Graphic mines, at Magdalena. The successful manufacture of zinc-lead pigment at Coffeyville, Kan., led to a greater demand for ore from the Graphic

mine, and some was taken also from the Kelly mine. The ore shipped from the Graphic mine contained from 20 to 25 per cent. zinc, and about 15 per cent. lead. The ore shipped from the Kelly mine consisted of calamine assaying 36 to 37 per cent. zinc, first-class sulphide ore assaying 34 to 35 per cent. zinc, and second-class sulphide ore assaying 30 to 31 per cent. zinc. A considerable tonnage of the second-class sulphide ore was exported to Germany. The Tri-Bullion Smelting and Development Company, which owns the Kelly mine, is now erecting a smeltery and oxide plant at Albuquerque.

(By Charles R. Keyes.)—At present the largest producing camp is Magdalena, in Socorro county. Since the opening of these mines other camps have been found to contain large amounts of zinc ore. The principal shipping points are Magdalena, Hanover, Tres Hermanas, Pinos Altos, Las Cruces, Cooks Peak, Granite Gap and Hachita; and a number of other locations will soon have adequate transportation facilities.

The chief feature to be noted in regard to the New Mexican zinc ores is their large percentage in the carbonate form. In the case of the Magdalena mines, which for 30 years were worked for lead only, the walls of the lead stopes have been found to consist entirely of replacements of zinc carbonate. Besides the carbonate ores there are large deposits of blende; also complex ores containing copper, lead, gold and silver. Zinc mining promises to be one of the most important of the mineral industries of this Territory.

Utah.—The production of zinc ore in this State was made by the Daly-Judge mine, of Park City, the Scranton mine in the Tintic district, and the Horn Silver mine at Frisco, Beaver county.

Wisconsin.—The increase in the production of zinc ore in this State in 1906 was decidedly more important than the statistics show, because the more general introduction of magnetic separating plants resulted in an increase in the average grade of the ore. Consequently while the production of ore increased from 32,690 tons in 1905 to 42,130 tons in 1906, the increase in the total zinc content was in considerably greater ratio. The best ore produced in the Wisconsin field is now on even terms with the average ore of the Joplin district in respect to zinc content and freedom from objectionable impurities.

(By E. W. Moore.)—It is now about five years since the first modern mill was put in active operation in the southwestern Wisconsin zinc field and those who at first were pessimistic have been forced to admit that they were wrong. A factor of great importance is the perfection to which magnetic separation has been brought. This has made it possible to work up a vast number of old dumps that were considered useless.

From the advent of the first mill in the district to the present day, the necessity for more economical and dependable methods of ore concentra-

tion has been continuously in evidence. The majority of the engines that are now being installed are of the Corliss type, which has been found to be particularly adapted to the strenuous work required in the concentrating and power plants.

It has been found that the small mill (of about 50 tons capacity per 10 hours) is best suited to the underground conditions of this district. If for no other reason, it enables the operator to cull down his mine run and only run through the dirt that contains pay ore, whereas with the larger mill it is necessary to run through everything to keep things going and the ground boss is often tempted to side-shoot in the deads in order to "give 'em enough dirt on top."

Unless one is thoroughly acquainted with the existing conditions in the Wisconsin-Illinois zinc district, it will be difficult to see much of an increase in tonnage production, as against last year. It will be necessary to go behind the reported figures showing the net tonnage shipped to the smelters. If this is done it will be found that 75 per cent. of all the ore now produced in the district is finally treated in magnetic separating plants which materially reduces the gross weight; consequently the total gross tonnage exceeded that of last year by about 85 per cent., a marvelous growth when it is remembered that it has been necessary to locate the ore by the slow and laborious method of churn-drilling. Last winter and spring 300 or 400 drills were busily prospecting for ore; today it is estimated that there are less than 75. This is due to several reasons, chief among them being that those who were drilling are now spending their money in development work.

During 1906 it is estimated that $1,500,000 was invested in new enterprises. Nearly 70 per cent. of the new ventures, it is claimed, will be successful. A noticeable feature is that fully 90 per cent. of the new strikes have been made in the immediate vicinity of, or on the outskirts of, the lead diggings of the early days, and that nearly all of the strikes in new territories show the same conditions of the early days, excepting that the old water levels have been lowered.

Improved pumping methods have demonstrated the fact that the water problem is not nearly as bad as at first it was thought it would be. In the early days the inadequate pumps made it appear that the water in the mines was of unusual amount and that it took a fortune to keep the mine unwatered. It is now known that a great number of the mines do not yield anywhere near enough water to supply the concentrating plants.

The territory surrounding Platteville, Hazel Green, Benton, Cuba City, Mineral Point, and a few other camps, has seen more paying properties come to life than any of the others. Probably the main reasons for the prominence of these camps are their proximity to railways and good hotel facilities. So far the principal activity has been confined to the camps

where ore was known to be, although between Platteville and Big Patch camps there is now a string of mines widely separated, but nevertheless showing that the ore runs (as some are inclined to believe) in undulating waves from these two camps at least.

The Mineral Point Zinc Company (a subsidiary of the New Jersey Zinc Company) has pursued its policy of buying up a number of zinc properties. A commendable proceeding of some connected with this company was in several instances to advance the funds to carry on operations to the producing period, securing as a remuneration a half interest in the mine. Other large companies, heavily interested in zinc elsewhere, have been quietly acquiring control of good mines and mining lands.

The railroads continued their broad policy and aided developments materially by their readiness to meet the operator half way whenever it has been found necessary to run spurs or sidings. The C. M. & St. P. is at present contemplating a new route into Platteville, which will give several of the promising mines splendid side-track facilities.

The results in the zinc industry of southwest Wisconsin have more than exceeded the most sanguine expectations of a year ago. Today there are in operation about 60 concentrating plants and a conservative estimate shows that at least 50 more mines will be ready for mills before 1907 closes. At the beginning of 1906 there was a general rush on all sides to put up mills, regardless of development work, and in the excitement several mills were erected a little prematurely, and one or two were built on properties that will probably never make good, but fortunately the majority of the prospectors learned a lesson and are benefiting thereby, as also is the district generally.

Virginia. (By J. A. Van Mater.)—Some prospecting was carried on in 1906, but no important results were realized. The Albemarle Zinc and Lead Company continued to explore its mine at Fabers, Nelson county, by sinking the shaft 60 to 70 ft. deeper and driving a cross-cut at the bottom, since which time the property has been idle. The company has a mill equipped with Krom crushers and rolls and Krom air jigs, but it has never been run, except experimentally.

The Cedar Spring Zinc Mine and Development Company, now called the Columbus Mining Corporation, at Cedar Springs, near Rural Retreat, in Wythe county, also has a mill which has never been run, because sufficient development work has not been done to supply it. The company is now giving its attention to prospecting and developing its property.

The only zinc and lead production during 1906 was made by the Bertha Mineral Company from the old Wythe mines at Austinville, in Wythe county, and this was entirely from surface workings, which are now practically exhausted. The total amount of zinc concentrate (carbonate and silicate) was about 500 tons and about 50 tons of lead concentrate (car-

bonate and galena). This output was considerably curtailed by the scarcity of men during the year, necessitating the closing down of all surface mining operations the early part of the summer. Development was prosecuted vigorously throughout the year on the sulphide ores, which were found about 200 ft. below the surface. The shaft was sunk 100 ft. below the tunnel level, or 335 ft. below the surface and the mineral developed on both the 235-ft. and 335-ft. levels by drifts and a winze between these two levels; in places considerable pyrite was encountered. Boilers and compressors have been erected for furnishing the mine with air, and air drills were in use throughout the year. The development, on the whole, has been encouraging, though no large area has yet been opened up. The deposit is not continuous, but the ore seems to occur in faulted zones wherever the rock conditions have been favorable for mineral deposition, it being a replacement in the altered limestone, sometimes following bedding planes, while at other times following the complex system of faults in a general way. It is a difficult property to develop, as there is apparently no connection between the deposits already located or with others, if they exist. A mill of 10 tons capacity per hour was erected during the year for concentrating these sulphide ores and will be ready for operation about Feb. 1, 1907. If the separation is a success this property will be a small producer during 1907; in the mean time the development work will be pushed as much as possible.

The plant for recovering the zinc from the tailings dump at the old Bertha mine, which consisted principally of zinkiferous clay, or "buck fat," burned in June, 1905, and afterward rebuilt of steel, was started up in January, 1906, was idle in February, through a breakdown in machinery, but ran the remainder of the year except at short intervals for repairs. The output from this plant was 882 short tons of material containing an average of 72.46 per cent. zinc and 5.81 per cent. lead with an average recovery of 73 per cent. of the total zinc in the tailings, which average 15 per cent. zinc and 30 per cent. water. Various efforts had been made to concentrate this waste product, hydraulically as well as magnetically, but without success, after which a combination of the two processes was tried. All failed and as a last resort metallurgical concentration was attempted, and it was only after the expenditure of many thousand dollars, and a year of constant study and experiments the process proved a success. The product from this plant is sent to the spelter furnaces at Pulaski where it is now successfully used, though it is far from being a kindly working material.

The Zinc Industry in Foreign Countries.

The production of spelter in Europe showed a large increase in 1906. In Upper Silesia, the most important single district, the output was 134,-

180 long tons, against 127,895 in 1905. The production of Belgium was 150,060 tons, against 143,300 in 1905. The European smelters received bountiful supplies of ore from Broken Hill, Australia, whence the offering of this raw material is destined to increase greatly. Largely on this account, the smelting capacity of Europe is being greatly increased, new and large works being now under construction in Great Britain, France and Germany. Of the new works, the Metallhütte A. G. of Duisburg, Germany, has already (March, 1907) gone into operation, while the International Metal Company of Hamburg expects to fire up three furnaces in the autumn. Australia itself will also become a larger producer of spelter, the idea being to make as much there as is required for home consumption. The Sulphide Corporation has already a works in operation at Cockle Creek, and its capacity is to be increased materially in 1907. The Central Zinc Company, a subsidiary of the Sulphide Corporation, is building a large works in Great Britain. The Broken Hill Proprietary Company completed a furnace at Wanatah, Australia, where a plant has been begun, which it is intended to extend in 1907, but the initial attempt at smelting toward the end of 1906 did not prove successful. In the meanwhile, the Zinc Corporation is considering the erection of a large plant. The outlook is, therefore, for a large increase in the supply of spelter outside of the United States in the near future.

In its report for 1906, the Rheinisch Nassauische Bergwerks und Hütten A. G. expresses some alarm respecting the erection of new smelting works that is now going on in Germany. It states that there are six large works in course of construction in Upper Silesia and Western Germany, the reason for this movement being the over-supply of ore recently offered, and the high rates for smelting exacted by the existing works, which have led various mining companies to become interested in works of their own. Some of the new works are being erected in positions which offer better facilities for the transportation of ores to them by canal and river from Rotterdam and Antwerp than any other smelters in Germany at present possess, and there is no doubt that the new works will capture a good deal of the business of smelting foreign ore. Whether the hopes of great profits will be fulfilled, however, seems doubtful, in view of the probability that, with increased competition, the smelting prices will fall to a normal level. But even if all difficulties be surmounted, nothing is to be feared from the new smelters during 1907, and as regards the more distant future, the results may be that the new works will only satisfy the annual increase of the demand, while the old works will maintain their present production.

Algeria.—According to a recent British consular report, the districts of Souk-Ahras and Tébessa abound in zinkiferous deposits still unworked. So far only one of these mines—Beccaria—has been seriously exploited. A large mass of mineral between layers of dolomite and marl was found,

and since 1899 calamine of good quality has been extracted. The calamine deposits at Djebel-Bourouman, Bou-Jaber, and Khanguet-Tenoukla are now being prospected.

Australia.—There was a great increase in the zinc production of this Commonwealth in 1906, as is shown in the accompanying table, which is based on the official report of the Department of Mines of New South Wales. The entire production was derived from Broken Hill where, as is well known, the resources of mixed sulphide ore are immense, the question of the day being simply to provide adequate milling facilities for the separation of the ore. The largest producer of the year was the Broken Hill Proprietary Company, which employs the Delprat flotation process. The suit of the owners of the Potter process against the Broken Hill Proprietary Company, on the ground that the Delprat process was an infringement of the Potter process, was decided in favor of the Proprietary Company. The plans of the Proprietary and other companies for the erection of smelting works have been referred to in a previous paragraph of this article. The production of spelter credited in the accompanying table has been made so far entirely by the Sulphide Corporation. The Zinc Corporation did not make the expected progress with its plans for the treatment of the great tonnage of ore in the various dumps which it purchased in 1905, but did a large amount of experimental work, in connection with which it made a small output, and doubtless will be in a position to make a large output in 1907. There is no question whatever that a great increase in the supply of zinc ore from Broken Hill is to be expected in the near future. The magnitude which it has already attained has had a highly important effect upon the zinc smelting industry in Europe, and upon the market price for spelter.

PRODUCTION OF ZINC IN NEW SOUTH WALES.
(In tons of 2240 lb.)

	1903	1904	1905	1906
Spelter	286	290	544	1,008
Zinc in ore exported	14,625	22,318	30,637	33,427

Canada.—The zinc industry of the Dominion, which gave considerable promise in 1905, came nearly to a standstill in 1906, the imports of zinc ore from British Columbia into the United States, which practically represent the total production of the Dominion, having been only 600 tons. The smelting works, erected by the Canadian Metal Company, at Frank, Alberta, were put in operation late in the spring, but because of defects in their construction they were closed down after a short experimental run, and since then have been idle. They made a small production of spelter, but the amount was insignificant. Experiments with the Snyder process of

electrothermic smelting were continued during the year, and a plant of capacity for treating 10 tons of ore per day is now being erected at Nelson, B. C. It will obtain power transmitted electrically from Bonnington Falls. Some prospecting for zinc ore was done on Calumet island, Quebec, where promising indications are said to have been found.

The stagnation in the development of the zinc industry of Canada was due entirely to outside commercial conditions. The European smelters were offered such bountiful supplies of ore from many sources that the prices they were willing to pay decreased materially (the margin in smelting increasing correspondingly), while American smelters withdrew entirely from the Canadian market for reasons which have been explained in a previous portion of this article. Evidently, the future of zinc ore production in Canada is largely dependent upon the final interpretation by the courts of the American tariff law.

Germany.—The average quarterly prices for spelter f.o.b. works in Upper Silesia, and the imports and exports of spelter and zinc ore are given in the accompanying tables.

AVERAGE PRICE OF SPELTER PER 1000 KG. IN SILESIA.

	1906	1905	1904
First quarter	509 Mk.	468 Mk.	416 Mk.
Second quarter	516	454	421
Third quarter	523	481	427
Fourth quarter	538	547	464
Average	521.50 Mk.	487.50 Mk.	432.50 Mk.

GERMAN IMPORTS AND EXPORTS IN CENTNERS OF 100 KG.

	Imports.		Exports.	
	1905	1906	1905	1906
Spelter	268,406	370,359	623,233	633,947
Zinc sheets	544	808	189,817	172,979
Broken zinc	27,425	22,777	53,515	57,007
Zinc ore	1,265,773	1,790,360	389,727	426,055
Zinc dust (poussiere)	....	6,033	....	40,443
Oxide of zinc	....	52,310	....	141,057
Lithophone	9,073	15,104	77,467	79,947

Japan.—A company has been formed in Japan for the purpose of establishing spelter and zinc rolling works with the object not only of supplying domestic demand, but also of cultivating an export trade. Heretofore the zinc ore mined in Japan has been exported to Europe. The plant will, it is said, have a daily capacity for smelting 50 tons of ore and rolling six tons of sheet zinc.

Mexico.—The production of zinc ore in this Republic increased largely in 1906, the American tariff question (which has been referred to in a

previous portion of this article) having had no effect in checking the marketing of Mexican ore among the American smelters. The chief part of the Mexican production in 1906 came from the districts around Monterey, which place was the chief shipping point, and consisted mainly of carbonate and carbonate-silicate ore, assaying from 30 to 40 per cent. zinc, averaging about 37 per cent. Some mixed sulphide ore was shipped from the Calera mine in Chihuahua, and magnetic separating plants for the treatment of such ore were installed by the American Smelters Securities Company at its mines at Velardeña, Durango, and at its Tecolotes mine at Santa Barbara, Chihuahua. A plant comprising Sutton-Steele dry tables and Sutton-Steele dielectric separators was installed at the Tiro General mine, Charcas, San Luis Potosi. Up to date, however, these plants are still more or less in the experimental stage.

Norway.—The Government has granted a concession to a German and Belgian company, to work zinc mines near Grua railway station, 33 miles from Christiania. The annual output is estimated at 10,000 to 12,000 tons of ore for export, containing 42 to 45 per cent of zinc.

Rhodesia.—Zinc ore is now produced at Broken Hill, in northwestern Rhodesia. The shipments of calcined calamine in the last three months of 1906 amounted to 5512 tons. The total for the year was 8965 tons.

The Broken Hill mining field consists of a number of kopjes which have only to be quarried for the zinc and lead ore which they contain. The zinc ore is calcined on the spot and is shipped via Beira to Swansea, Wales. The calcined ore assays 50 or 60 per cent. zinc. The cost of mining and sending the ore to Wales, including all charges, is about $19.47 a ton. The lead ore will be smelted on the spot; this ore also is rich.

Russia.—The zinc industry of this Empire, which is still confined to Poland, was hindered considerably by strikes in 1906, but by no means to the same extent as in 1905, wherefore the set-back in the production of spelter in the latter year was entirely recovered. The mines still experienced a good deal of trouble, and their production was considerably below the average, but the smelters had large stocks of ore carried over from 1905, which they were unable to smelt in that year owing to the troubles with their men. The production of calamine ore in Poland in 1906 was 3,806,685 poods against 5,730,027 poods in 1905; the production of sulphide ore was 500,740 poods in 1906, against 335,667 poods in 1905.

Tunis.—The shipments of zinc and lead ore from Tunis in 1906 amounted to about 50,000 metric tons.

The Spelter Markets in 1906.

New York.—The spelter market during 1906 displayed a comparative degree of steadiness. Fluctuations were far less wide than in former years, although they occurred at frequent intervals. This would seem

due to the tendency of smelters to sell only their current output, so that during periods of a heavy demand prices were driven up quickly, while upon the subsidence of business a decline was brought about through efforts to market the spelter as produced. The movements in either direction were, however, held in check, at times by the proximity of the European market and throughout by the fact that the domestic consumption about equaled production.

Except for the high-grade special brands produced in the East, for which there is a regular outlet in Europe, there were no exports. The increased consumption in this country was taken care of through the operation to the full capacity of all existing works during most of the year. The active utilization of complex zinc ores in the West and a more comprehensive development of the calamine mines in the northern part of Mexico have supplied the increased requirements of raw material. As a result, the margin between the price of ore and the price of spelter was wider than a few years ago, when its closeness worked to the great detriment of the smelting industry.

The prices at the beginning of January stood at 6⅜c. St. Louis and 6.50c. New York. A surplus had been carried over from the previous year, and in an effort to market it prices were depressed, at first gradually, and then quickly, until they reached 5.90c. St. Louis, 6.05c. New York, at about which level the market remained during the whole of February. The lower level brought about a buying movement in March of very large proportions, which lifted the market to 6.20c. St. Louis, 6.35c. New York by the middle of the month. The discussion of the possibility of a coal strike caused buyers to hold off about that time, and as a consequence the market again receded gradually until it drifted to 6c. St. Louis and 6.15c New York toward the end of March. In April the decline made further progress, there being a desire on the part of smelters to put out some spelter against their supplies of ore, which came in freely from the West at the opening of the season. The same causes affected the market during May, until 5.80c. St. Louis and 5.95c. New York were touched about the last of that month.

A new factor made its appearance in June through a rather brisk advance in the London price to about the parity of the domestic market. This resulted in a very heavy inquiry for export. There being no surplus stocks, business for export did not result; but the aforesaid conditions stimulated buying for both speculative and consumers' account in this country, so that in the month of June prices recorded an advance to about 6.10c. St Louis, 6.25c. New York.

With a decline in the London market, prices here again drifted back almost to the starting point, and business during July was done between 5.80 to 5.90c. St. Louis and 6 to 6.05c. New York. During the entire

month of August the market remained steady at around this level, with a good business doing from day to day. The great industrial activity, which had previously had a pronounced effect on all other metals, finally made itself felt in the spelter market in September, and under a very heavy demand from all interests prices moved up to 6.15c. St. Louis, 6.30c. New York, at which figures they stood at the close of the month.

A slight reaction took place in October, but buyers only waited for some recessions to renew their purchases on a large scale, and by the beginning of November the market had again made up the decline. A steady and sustained demand resulted not only in the maintenance of prices, but brought about a further slight advance during November, which closed firm at 6.25@6.30c. St. Louis, 6.40@6.45c. New York. At the beginning of December it became apparent that all available supplies had been absorbed by the unprecedented consumption on the part of both galvanizers and brass manufacturers. In consequence of this consumers deemed it advisable to cover their requirements for the early part of the coming year, even though they were compelled to do so at advancing figures. The year closed with the market steady at 6.60c. St. Louis, 6.75c. New York.

AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900.....	4.65	4.64	4.60	4.73	4.53	4.29	4.28	4.17	4.11	4.15	4.29	4.25	4.39
1901.....	4.13	4.01	3.91	3.98	4.04	3.99	3.95	3.99	4.08	4.23	4.29	4.31	4.07
1902.....	4.27	4.15	4.28	4.37	4.47	4.96	5.27	5.44	5.49	5.38	5.18	4.78	4.84
1903.....	4.87	5.04	5.35	5.55	5.63	5.70	5.66	5.73	5.69	5.51	5.39	4.73	5.40
1904.....	4.863	4.916	5.057	5.219	5.031	4.760	4.873	4.866	5.046	5.181	5.513	5.872	5.100
1905.....	6.190	6.139	6.067	5.817	5.434	5.190	5.396	5.706	5.887	6.087	6.145	6.522	5.882
1906.....	6.487	6.075	6.209	6.078	5.997	6.096	6.006	6.027	6.216	6.222	6.375	6.593	6.198

AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN ST. LOUIS.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1903.....	4.688	4.681	5.174	5.375	5.469	5.537	5.507	5.550	5.514	5.350	4.886	4.556	5.191
1904.....	4.673	4.717	4.841	5.038	4.853	4.596	4.723	4.716	4.896	5.033	5.363	5.720	4.931
1905.....	6.032	5.989	5.917	5.667	5.284	5.040	5.247	5.556	5.737	5.934	5.984	6.374	5.730
1906.....	6.337	5.924	6.056	5.931	5.846	5.948	5.856	5.878	6.056	6.070	6.225	6.443	6.048

London.—The opening of the year found producers sold well ahead, and only moderate quantities in second hands. The galvanizing trade was very active, necessitating large purchases of spelter; and sheet zinc was in good demand. Under these conditions the market was firm at about £29 10s. for specials, until Jan. 15. On that day there was a sensational fall in prices, due to forced sales of speculative holdings, in-

volving a drop of £1 per ton. Within three days ordinary brands were offering at £27 12s. 6d., and specials at £28 2s. 6d. At the end of the month, ordinaries were quoted £27 1s. 3d. and specials £27 7s. 6d. Febary opened quietly, but on the 13th a sensational fall was caused by the operations of disappointed speculators, followed by further bear sales. Consumers' support was lacking, and the market was the sport of small holders who—toward the end of the month—depressed the price to £25. The month closed at a slight improvement, ordinary brands being held for £25 3s. 9d. and special brands for £25 10s. March opened with moderate buying on the part of consumers whose favorable attitude was, however, counterbalanced by persistent speculative selling; and within a week the price of ordinary brands had fallen to £24 5s. Producers, however, took a different view of the situation, and would make no concessions; and the influence of a strong buying movement on the Continent caused a sharp recovery. The month closed at £25 for ordinaries, and £25 12s. 6d. for specials.

April brought confirmation of reports of depleted stocks and brisk consumption. Sheet zinc in particular showed increased activity. Galvanizers were soon aroused to enterprise, and there was some anxiety shown to cover requirements for May and June, forward delivery commanding slightly better prices than the early. Ordinary brands, which were obtainable at the outset at £25, closed at £26 15s. and special brands at £27. May opened with renewed activity in the galvanizing trade. Sheet zinc also developed increased activity at home and abroad, with consequent advance in prices. At the end of the month, ordinary brands commanded £27 12s. 6d. and specials £27 15s. June found the market rather languid. The Silesian producers had already sold out for the third quarter of the year. Galvanizers were busy, and the trade in sheet zinc was unprecedentedly large. Consumers bought boldly, and the month closed with a firm market at £27 5s. for ordinary brands, and £27 7s. 6d. for specials.

July opened at the reduced price of £27, and there the price remained for a week or more, with no active demand on the part of consumers and no pressure to sell on the part of producers. A speculative movement reduced prices to £26 10s., and the month closed at £26 7s. 6d.@ £26 10s. for ordinary brands, and £26 15s. for specials. August opened with the market sensitive to the influence of small lots on offer, but influential buying soon gave it an improved tone. Toward the middle of the month producers marketed large quantities on the Continent. Toward the close some important orders from India pointed to increased consumption in yellow metal, and galvanizers shook off their reserve and bought boldly. The result was a gradual rise to £27 7s. 6d. for ordinary brands, and £27 10s. for specials. September was quiet. Con-

tinental producers were firm in their prices, having mostly sold out their production till the end of the year. There was great activity in sheet zinc at frequent advances in price. The month closed quietly, but with a strong under-current, values being £27 15s. for ordinary brands, and £27 17s. 6d. to £28 for specials.

October opened with little business passing. The market was strong, and though second-hand lots were again pressed for sale and values consequently declined, the month closed quietly at £27 15s. to £28, and specials £28 to £28 5s. In November the market was listless and Continental dealers became restless in consequence and made cheap offers to consumers, thereby forcing values down about 10s. Toward the middle of the month a large business developed with home consumers, all cheap second-hand lots were absorbed and sheet zinc displayed great activity. The loss was soon retrieved and the month closed at £28 for ordinary brands and £28 5s. to £28 10s. for specials.

December opened with a large business with consumers for delivery early in 1907, and at full prices. Efforts to depress prices met with little success, it being evident that little material was free for market purposes, and the trade in galvanized iron was persistently expanding. By the middle of the month dealers had sold well, and producers were holding for prices higher than those currently reported. Fluctuations were very narrow, but generally upward. Toward the close additional strength was imparted to the market by the advent of orders for yellow metal, and closing prices were £28 2s. 6d. to £28 5s. for ordinary brands, and £28 7s. 6d. to £28 10s. for specials.

AVERAGE MONTHLY PRICE OF SPELTER IN LONDON.
(Pounds sterling per ton of 2240 lb. of good ordinary brands.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	£	£	£	£	£	£	£	£	£	£	£	£	£
1905.....	25.063	24.594	23.825	23.813	23.594	23.875	23.938	24.675	26.375	28.225	28.500	28.719	25.433
1906.....	28.225	25.844	24.563	25.781	27.000	27.728	26.800	26.938	27.563	28.075	27.787	27.938	27.020

Progress in the Metallurgy of Zinc.

The chief feature of progress in 1906 was the further attention devoted to the mechanical separation of mixed sulphide ores and the improvement in processes already introduced for that purpose, which have put it beyond all question that an immense increase in the supply of zinc is now available from this class of ore, formerly of little or no value. It is now quite well established that there are few varieties of this sort of ore which cannot be raised to a grade of 40 per cent. zinc by magnetic separation, while in many cases much better can be done. Moreover, there

are some ores which can be even more efficiently worked by the Sutton-Steele pneumatic table, by electrostatic separation or by flotation. Electrostatic separation does not appear to have made much further headway, but in the flotation processes great strides have been made. At Broken Hill the flotation processes have clearly beaten magnetic separation at all points, including extraction of minerals, degree of concentration, and economy both in operating expense and in first cost of plant.

Mechanical Separation.

It is another example of the backwardness of American metallurgists in adopting new inventions of foreign origin that the flotation processes, which have become of so great importance in Australia, are still but little known in the United States. However, it is satisfactory to report that during 1906 an installation of this kind was made at Marion, Ky., for the separation of the blende-galena-fluorspar ore of that district (which previously had been a difficult problem) and the results have been all that was hoped for. The samples of the concentrates that I have seen have certainly been of excellent quality. It is interesting, moreover, to remark that the zinc of this ore occurs distinctly as resin blende, which character of mineral, it had been previously stated, was incapable of flotation. The fact is that at the present time no one knows what is the real theory of flotation, and the adaptability of the process to any particular ore can be determined only by experiment.

The process employed at Marion, Ky., is essentially different from the process developed in Australia, inasmuch as no acid is used in the bath, the latter being, on the contrary, basic, being in fact a very dilute solution of aluminum sulphate. Moreover, instead of being performed in deep vats, as in the case of the acid flotation process (wherein the depth of the vat has been found to be a rather important consideration), the basic flotation process is performed in shallow pans, in which mechanical stirrers are operated. It is obvious that this is a very different process from that which has come into use in Australia, and must necessarily be entirely free from the patent litigation which has arisen over the latter. Besides the work at Marion, Ky., some experiments with the Sanders process have been made by the Tennessee Zinc Company, at Straight Creek, Tenn.

Another form of flotation process, in which no acid or other chemical agent is employed, has been developed by Otto Germer and Otto Stallman, being known as the Stallman-Germer process. The action of this process depends solely upon the physical principle of surface tension. It is claimed that highly successful results have been obtained in the preliminary experimental work. Anyway, a mill for the exploitation of the process is now being erected at Rico, Colo., where the ore of the Atlan-

tic Cable mine is to be treated, an arrangement having been made with the United Rico Mines Company, the owner of that mine.

In Australia, the Broken Hill Proprietary Company has continued to use the Delprat process. A new feature of the latter has been the increase in the depth of the separating vats employed. It has been found that the deeper the vat, the better is the separation. Outside of the Broken Hill Proprietary Company, the largest part of the concentrate produced at Broken Hill has been obtained by the Potter process, but for some reason, which has not yet been published, the Zinc Corporation, which it was supposed would adopt that process, abandoned it after a rather extensive trial and is introducing the Cattermole process. The Potter process was the original flotation process. The Delprat process is a slight modification of the Potter, being designed to evade the patents on the former. In addition to these processes, there have developed many others, including the DeBavay, Gillies, Ballot (or Cattermole), and the Elmore Vacuum, all of which are being experimented with on a more or less extensive scale at Broken Hill. The time has not yet come to make any comments upon the relative efficiency of these processes.

Zinc Smelting.

In zinc smelting there were not any particularly noteworthy innovations in 1906, although there were as usual many improvements in details of the art, while moreover the year was prolific in the presentation of new ideas, which are to be tried, but have not yet demonstrated their advantage, if any. Among these are the Imbert process, and various new processes of electrothermic smelting.

Roasting Furnaces.—The new plants that have been erected in Illinois, all of which are planned with a view to the manufacture of sulphuric acid, have been equipped with the Hegeler roasting furnace. The advantages of the McDougall type of furnace for this purpose continue to be overlooked. In Kansas, where sulphuric acid manufacture is not generally considered, all the new plants have adopted the Zellweger furnace, which does good work but like other furnaces of its type is wasteful of fuel. The use of the Ropp, Cappeau and Brown furnaces is limited by patent rights, wherefore outside companies are obliged to adopt some other form of furnace. At the present time, of course, no one would think of using the old hand-raked kilns. At the Argentine works of the United Zinc and Chemical Company, furnaces of the McDougall type have been used for the roasting of Western ores, but Hegeler furnaces are being substituted because of the desire to have larger units. The ore is roasted at Argentine for the manuufacture of sulphuric acid, and the cinder is shipped to Iola for reduction to spelter.

The Greenawalt roasting furnace was tried on a working scale by the

Prime Western Smelter Company at Iola, Kan. This furnace is a mechanically-raked reverberatory, with a hearth 120 ft. long and 10 ft. wide, the essential feature of which is that the hearth is constructed in a porous form, so that air may be percolated up through the hearth into the incandescent ore resting upon the latter. Theoretically, of course, this idea is excellent, and in experimental work it was found that all expectations were fulfilled, in so far as increasing the hearth efficiency, reducing the fuel consumption per ton of ore, etc., were concerned, but in practice the furnace has not been found to work so well, owing to the clogging of the pores of the hearth by the sintering of the ore upon it, a result which would naturally be expected.

Distillation Furnaces.

Various new types of distillation furnaces have been tried experimentally, but no remarkable improvement appears to have been effected. Smelters generally stick close to the lines of their experiences and are timid about venturing far from them. The character of the new installation at DePue, Palmerton, and elsewhere, was referred to in my article in the last volume of THE MINERAL INDUSTRY.

The only innovations of 1906 in the United States were the introduction of natural gas fired furnaces with large retorts by the United Zinc and Chemical Company, at Iola, Kan.; and experiments with a chimney-draft natural-gas fired furnace, patented by William Lanyon, and introduced at his works at Caney, Kan. The Lanyon furnace (U. S. patent No. 839,160, Dec. 25, 1906) was constructed as a massive of 300 retorts, in which the gas and air are introduced at the bottom, the products of combustion rising to the arch, then being deflected downward, and finally escaping to the chimney through flues in the middle wall.

Retorts and Charging Devices.—Experiments have been made, with successful results, with the Queneau composite retort and the Queneau charging machine. A similar form of charging machine has been tried at Ougrée, Belgium, which was described in the *Engineering and Mining Journal,* of Oct. 6, 1906. The idea of charging the retorts mechanically has long been hopefully looked forward to, but after all it is questionable as to how much advantage will be gained and whether the machines that have now been invented, which certainly are workable, will come into general use.

In Europe there has been increased use of siloxicon and similar highly refractory material in the batch for making the retorts, while in America the Queneau composite retort, which is an ordinary retort made by hydraulic pressure and lined with highly refractory material in the single operation of manufacture, has given successful results.

New Processes.

Among the new processes, experiments have been made with the Imbert process in France, with the Ganelin process at Overpelt, Belgium, and with the Schmieder process in Silesia. The Imbert process is to have its first practical trial on a commercial scale during 1907, a furnace being planned for at Bartlesville, Ind. Ter. Further experiments were to have been made with the Lungwitz process at Warren, N. H., but they were postponed again. Electric furnaces have received considerable attention, but so far they are in use only in Scandinavia (de Laval furnace) and they do not yet present any promise of displacing the standard form of furnace. In Scandinavia they are employed rather for the smelting of waste products than for the direct smelting of ore. In the latter work it has not yet been possible to obtain any product but blue powder, which must be resmelted. These works in Scandinavia enjoy very cheap power. Experiments with the Snyder furnace were made in British Columbia during 1906, and a commercial plant, of capacity for smelting 10 tons of ore per day, is now being erected atNelson, B. C.

A REVIEW OF THE LITERATURE ON ORE DEPOSITS IN 1906.

By J. F. Kemp.

The last review, which appeared in Vol. XIII of The Mineral Industry, brought the literature well toward the close of 1905. The present chapter begins with contributions dating either from the closing months of that year or from 1906 itself. The English translation by W. H. Weed of Richard Beck's "Lehre von den Erzlagerstätten," under the title "The Nature of Ore Deposits," marked the opening of this period. It places at the command of English-speaking or English-reading students of the subject a very valuable and modern work which had previously been limited to those who had German at command. To the ground covered by the original, Mr. Weed adds many observations on North American localities, which give an added value to the book on this side of the Atlantic.

During the past year the third and concluding volume has appeared of "Die Erzlagerstätten," the posthumous manuscript of Prof. Alfred Stelzner of Freiberg, edited and in part rewritten by Prof. Alfred Bergeat of Clausthal. The two earlier volumes have been already mentioned in these pages. The concluding one completes the discussion of the various kinds of veins, following out the mineralogical characteristics and types peculiar to the Freiberg school of observers. Cave-fillings and replacements are also treated, and finally placers and residual deposits, the last two constituting the deuterogenic group. A very interesting discussion is given of the whole question of the filling of veins, of the derivation of the waters which have been the vehicles of introduction and of the source of materials, the general conclusion being the importance of deep-seated sources and agents. All in all, the three volumes constitute a monument to the thorough scholarship of both author and editor. Even the most recent work in all parts of the world has been reviewed and duly cited, with the result that the books are cosmopolitan in scope.

A briefer text-book of a general character is the "Economic Geology of the United States," by Prof. Heinrich Ries of Cornell University.[1] As the work bears the same title as an earlier book by Dr. Ries' colleague, Prof. R. S. Tarr, and comes from the same publishers, we infer that it will replace the latter. Professor Ries gives greater prominence to the non-metallic products than to the ore deposits; but in the last half of the book he presents a concise review of the nature and distribution of the mining

[1]New York, The Macmillan Company, 1905.

districts of this country. As a text-book for colleges, as well as for the general reader, the work is admirably adapted to the needs of a wide circle.

The results of the Missouri Bureau of Geology and Mines, under Dr. E. R. Buckley as director, relating to the study of the lead and zinc mines of the State, have been awaited with much interest. A report on the "Geology of the Granby Area" by E. R. Buckley and H. H. Buehler gives some new interpretations of this interesting district. Readers familiar with the literature will recall that in explanation of the breccias of chert in which the blende occurs the following views have been successively advanced. A. Schmidt in 1871-72 emphasized the shrinkage from dolomitization, and the collapse of caves. E. Haworth in 1884 brought forward for certain areas the oscillations from folding. F. L. Clerc in 1887 reaffirmed the collapse of caves. With these general views Arthur Winslow was also in accord. W. P. Jenney in 1893 laid stress on faults, a view largely adopted by H. F. Bain in 1901. Doubt was cast on the reality of the faults by C. E. Siebenthal in 1905, with emphasis on unconformities and a renewed appeal to collapse from underground drainage. Buckley and Buehler now develop a view different from any of the earlier ones. The elevation of the region after the deposition of the Burlington limestone and its extensive erosion and dissection in Lower Carboniferous time are demonstrated. During this process great surface breccias of residual chert are believed to have been produced on the hillsides and along the edges of the stream-valleys. Subsidence during the period of the coal measures sufficed to bury them under the accumulating sediments of the Pennsylvanian strata, where they now lie as apparent fault breccias, or collapsed breccias, but really due to surface weathering. They also of necessity lie along the horizon of what is now a marked unconformity, giving them the semblance of faults. The metals and their ores are believed by the two authors to have been derived from the overlying Pennsylvanian shales and sandstones through the agency of descending surface waters.

The Bureau has also issued a report by F. B. Van Horn upon Moniteau county, in the central part of the State, which describes the deposits in caves, or "circles" of that section, besides giving a review of the local geology. The reports are well printed and illustrated, but if when future volumes relate to counties, they might have small sketch maps of the State, so as to show the location of the area described, readers at a distance would more quickly orient themselves. The insertion of the date of publication on the title-page would also add to the value of the volumes.

The lead and zinc deposits of southwest Virginia, hitherto the subject of only scattered papers, have been selected as the theme for Bulletin I of the recently established Geological Survey of Virginia. Prof. Thomas L. Watson gives the local details of the mines, shows that the ores occur

in a region of rather intense folding and faulting, and advances the view that, as in the explanations most frequently urged for the deposits of the Mississippi valley, the metals have been gathered by descending meteoric waters and precipitated where now found in the limestones as blende. Subsequent oxidation has yielded the crusts of calamine. It would seem, however, in the mines as if the descending oxidizing waters operated to develop the somewhat permanent crusts of calamine immediately beneath the residual clays and on the limestone rather than to carry the zinc down to lower depths. If the surface waters of the past have acted as do those of today, it is somewhat questionable whether they had the power to carry the zinc beyond this dead line; and one may perhaps still ask whether there is not yet something to be said in favor of waters uprising from profound depths. Besides the deposits in southwest Virginia, there is in Albermarle county a vein in schists with galena and blende associated with fluorspar and quartz. Diabase and diorite dikes are near, and the latter are believed by Professor Watson to have been connected with the ore deposition.

The source of the carbonic acid gas in mineral springs has received interesting treatment from Rudolf Delkeskamp in a paper whose title in English would be "Vadose and Magmatic Carbonic Acid." (*Zeit. f. prakt. Geologie*, February, 1906). The subject is of much interest in connection with ore deposits, as it bears directly on the source of deep spring waters. After a review of the earlier theories which have been presented to explain the carbonic acid in the European springs, such as the alteration of limestone by heat or by silicification or by acids, the oxidation of siderite, the oxidation of carbonaceous deposits, etc., Doctor Delkeskamp falls back on magmatic waters from expiring igneous action as having the best claims in many of the instances. The question takes on some added interest in America owing to the recent agitation at Saratoga against tapping off the subterranean sources of the carbonated spring waters by artificial borings, the stimulating motive for which was in large part the desire to obtain and compress for the market the exhalations of carbonic gas.

In the *Zeit. f. prakt. Geol.* for July, 1906, appear two important papers on manganese. Prof. J. H. L. Vogt discusses bog-manganese ores and the relations between the iron and the manganese both in these types and in the lake ores. After a brief description of several Norwegian localities, the general subject is followed through with many analyses. In the same number Dr. E. Hussak describes the manganese ores of southern Brazil. There is also a review of a curious paper read by Prof. H. Barvir before the Royal Bohemian Academy of Sciences, in Prague, Feb. 23, 1906. The author states that electric currents stream into the atmosphere from bodies of metallic ores at or even beneath the surface more richly than elsewhere. He believes that these can be identified and shown by pho-

tography, and he suggests this method of prospecting. In the September number, Dr. Hussak gives valuable notes on the occurrence of palladium and platinum in Brazil. In the October number the same author discusses the diamonds of western Minas Geraes and neighboring states, and gives many details which are of especial interest in connection with the high price of the gems today.

In Vol. XII of The Mineral Industry, pp. 386-388, the present writer briefly reviewed the results of the study of the enormous bodies of magnetite at Kirunavara and Luossavara within the polar circle in northern Sweden. That at Kirunavara is believed to be the greatest single body of iron oxide known. It lies between a foot-wall of orthoclase-porphyry, rich in soda and a hanging wall of quartz-porphyry likewise rich in soda, and has a steep dip. Five suggested methods of origin were mentioned in the review. Recently O. Stutzer of Freiberg has visited the mines and has given a further contribution upon the formation of the ore. From the general presence and occasional abundance of apatite and from the relations along the contact, Mr. Stutzer concludes that the ore must be an intrusive, basic dike between porphyry walls. ("The Iron Ores at Kiruna," *Zeit. f. prakt. Geol.*, March, 1906, p. 65.) Almost all observers have felt conservative about igneous dikes of a nearly pure base, such as iron oxide, although believing that titaniferous varieties, in which the TiO_2 was analogous to SiO_2, might form them. At the same time the renewed suggestion of this method by an observer who is obviously unprejudiced must serve to make others willing to view these and similar orebodies from the same standpoint.

In the same number of the *Zeitschrift* Prof. Richard Beck contributes a paper read by him at the meeting of the British Association for the Advancement of Science held in South Africa in 1905. It is entitled, "The Relations Between Ore-veins and Pegmatites." To the well-known veins of tin which are of pegmatitic character Professor Beck adds others of copper, nickel and gold. In this connection it is interesting to mention Professional Paper No. 55 of the United States Geological Survey[1] which discusses this same theme at much greater length and with an exceedingly thorough review of previous literature. In the Silver Peak area there is an extensive series of limestones, slates and quartzites belonging to the Cambrian and Ordovician. Tertiary lake beds are the next sediments. Presumably in the Jurassic or Cretaceous they were penetrated by extensive intrusive masses of a granitic rock consisting of quartz and alkali feldspar called alaskite. Various grades are found from feldspar-rich to quartz-rich varieties. Besides the large masses there are lenses in the sediments not only of normal alaskite but of quartz with little or no feldspar and carrying gold. The gold is mostly free, less often with pyrite.

[1] "Ore Deposits of the Silver Peak Quadrangle. Nev," by J. E. Spurr, 1906.

The quartz is regarded as an extremely silicious phase of the alaskite magma from which the silica in company with much water and possibly other mineralizers come off as a viscous end-product. By a process of intrusion the quartz is believed to have forced its way into the slaty rocks, parting them and giving rise to the lenses. The quartz is believed to have entered practically in a state of fusion and not as dissolved material in magmatic waters. The connection of the process with pegmatites is discussed at length with many citations of cases. Diorites which followed the alaskite and which cut the quartz lenses are later and have only exercised a small enriching effect.

The eagerly awaited report of Messrs. Lindgren and Ransome upon Cripple Creek, Colo., has appeared during the past year,[1] and is a work of more than ordinary interest. It brings up to date the earlier descriptions of Cross and Penrose and by the increase in mining the authors are able to demonstrate a number of new points. The shape and outline of the old volcano are now pretty definitely worked out, so that we know that a huge crater was blown up through the granite and schist, roughly 5 miles long from northwest to southeast and 3 miles wide. The walls in the older granite are nearly vertical, and may even overhang. All the eruptives, whether occurring as fragments in the breccia or in sheets and dikes, are intrusive rather than surface flows, and have crystallized at some depth.

The petrographic details have been worked out by L. C. Graton and we learn that the general eruptive is a phonolite or a variation of this standard type. As a whole, they belong to the group of rocks called "latites," which partake alike of the characters of trachytes, andesites and basalts. There are more basic forms called trachy-dolerites and some very basic dikes.

The entrance of the ores followed the last intrusion of a basic dike. The heated alkaline waters followed systems of small fissures which roughly radiate from the center of the old crater. They are insignificant in individual size, although often so grouped as to make the rock sheeted. They rarely show appreciable displacement of the walls. They are referred to as a subsidence or settling of the breccia after the entrance of the basic dikes. The ores were then probably introduced by hot, alkaline, magmatic waters which rose from some deeply buried mass below. There is considerable metasomatic replacement. Twenty-three original and nine secondary minerals occur with the veins, and 68 good species and eight doubtful ones are known in the district. Nearly 6¼ millions of ounces of gold have been produced and about one-tenth as many of silver. The report further contains many details of the several mines, with brief histories and statistics. Three maps and numerous plates make matters

[1] "Geology and Gold Deposits of Cripple Creek Colo." Professional Paper No. 54, U. S. Geological Survey, 1906.

clear. The report was prepared by master hands at mining geology and is a rare combination of scientific results and their practical applications, showing how well the two may go harmoniously together.

In addition to Silver Peak, another mining district in the Great Basin is the subject of J. E. Spurr's monograph upon Tonopah.[1] The volume contains matter of much more than local interest and importance. The history, local geology and details of structure at Tonopah are set forth with much fulness and with many illustrations. The history of the discovery does not lack the personal element and the romantic feature attaching to so many Western camps. The local geology presents another of those volcanic centers with which we are more and more associating the newer mining districts. There are recognizable at Tonopah as the oldest volcanics an earlier andesite and a later one. The former is slightly more acidic than the latter, and if considered as representing the original magma, we may conclude that, on differentiating, it separated into a somewhat less silicious portion, the later andesite, and a much more silicious portion, the dacite, which in this case followed the later andesite. The dacite, which is the third outbreak in age, was succeeded by rhyolite, and finally the eruptive activity was closed by basaltic extrusions. This succession corresponds very well with Von Richthofen's natural system, as set forth nearly forty years ago. Showers of tuffs were likewise sent forth, and lake beds figure also in the total.

The principal mineralization took place after the outbreak of the earlier andesite, and the resulting veins are limited to it, being cut off by the later flows. They are silver ores in quartz gangue, but with associated adularia, or potash feldspar. A little gold and some chalcopyrite, pyrite, galena and blende are also present; but even in mass, regardless of value, the silver exceeds the base metals. In connection with the rhyolite-dacite flows, there are at least two other series of veins, but with small values.

Mr. Spurr has made a careful study of the vein contents and of the alteration processes which have been especially developed in the earlier and later andesites. He reaches the following very interesting conclusions: The waters which deposited the veins in the earlier andesite entered before the later outbreaks and were in all probability magmatic. They were yielded in the expiring stages of this period of igneous activity. Yet while coming from a rock of medium composition, which contains far more soda and other bases than potash, they were characterized by dissolved potassium and silica. On the other hand, the waters which produced the alteration of the later andesite, and which are believed to have been associated with the rhyolitic outbreaks, rocks high in the alkalies and low in the other bases, were nevertheless characterized by dissolved lime, magnesia and iron. A curious antithesis is thus afforded which Mr. Spurr

[1] J. E. Spurr, "Geology of the Tonopah Mining District, Nev.," Professional Paper No. 42, U. S. Geological Survey.

does not attempt to explain fully. Obviously, however, it is directly opposed to any leaching process of the walls, such as lateral secretion.

The closing chapters of the book are practically an argument for the importance of magmatic waters. This view is fortified by the relations of the present hot and cold springs in Nevada—and by the broad connections between the great metalliferous belt which surrounds the Pacific Ocean and which is coincident with the principal regions of productive precious metal mines of the world. The association of the latter with this igneous zone, and the infrequency and comparative inferiority of the mines elsewhere mark the significant relations of the two.

The association of orebodies with contact zones has been one of the subjects of especial interest discussed during the past year. The importance of this relation was brought out ten or twenty years ago by the Norwegian geologists, but in connection with the comparatively unimportant deposits of this type which occur in the Scandinavian peninsula. Later Mr. Walter H. Weed in the papers cited below[1] laid deserved emphasis upon it. The present writer, in connection with a Mexican locality, presented the interpretation of the addition of silica and some bases to the limestone by emissions from the eruptive,[2] and Prof. W. O. Crosby,[3] in describing Washington Camp, Arizona, has argued for the recrystallization of the sediments into garnet, wollastonite, etc., without contributions from the eruptive. The present writer has again taken up this theme in a paper before the International Geological Congress, in Mexico, Sept., 1906[4], and has sought to show that the prevailing garnet is an iron-lime garnet, such as could not have been furnished by the limestone, rather than the hitherto assumed alumina-lime garnet which observers have believed might be supplied by an earthy limestone. The endeavor is also made to show, by such analyses of the limestone and the contact minerals as are as yet available, that contributions from the eruptive are necessary. The extraordinary abundance of deposits of this character in Mexico and in the southwestern States and Territories gives this subject an interest and importance which is daily increasing.

In the closing paragraphs of this review in Vol. XIII, the writer referred to the anticipated appearance of the magazine *Economic Geology* under the editorship of Prof. J. D. Irving. This new venture has successfully passed its first year and has presented to its readers a volume filled with suggestive and stimulating papers, the larger part of which relate to ore-deposits. In turning over its pages one cannot but be impressed with the

[1]W. H. Weed, "Contact Metamorphic and Other Ore Deposits Near Igneous Contacts," *Eng. and Min. Journ.*, LXXIV, 513, 1902; "Ore Deposits Near Igneous Contacts." *Trans.* A.I.M.E., 1903, XXX 511, 715.

[2]J. F. Kemp, "Copper Deposits of San Jose Tamaulipas, Mex." *Trans.* A.I.M.E., 1906, XXXVI, 178. "On the Formation of Garnet Zones at the Contacts of Eruptive Rocks and Limestones," *Min. and Sci. Press*, Mar. 31, 1906, 220.

[3]W. O. Crosby, "Limestone-Granite Contact-Deposits, Washington Camp, Arizona," *Ibid*, p. 626.

[4]J. F. Kemp, "Ore Deposits at the Contacts of Intrusive Rocks and Limestones, and Their Significance as Regards the General Formation of Veins." Printed in advance of the Proceedings of the Internat. Geol. Congress, in *Economic Geology*. January, 1907.

vigorous young school of workers in this branch, now observing and pondering the facts of mining geology throughout North America. Among the papers of more than ordinary interest are the following: "Ore-deposition and Deep-Mining," W. Lindgren, p. 34; "Genesis of the Lake Superior Ores," C. K. Leith, p. 47; "Precipitation of Copper by Natural Silicates," E. C. Sullivan, p. 67; "Ore-deposits and Industrial Supremacy," J. L. Stewart, p. 257; "Sedi-genetic and Igneo-genetic Ores," H. F. Bain, p. 331; "Metasomatic Processes in the Gold-deposits of Western Australia," W. Lindgren, p. 530; "Experiments on the Solution, Transportation and Deposition of Copper, Silver and Gold," H. N. Stokes, p. 644; "The Microscopic Examination of Opaque Minerals," Wm. Campbell, p. 751. The last-named paper is of more than ordinary suggestiveness to the investigator. Dr. Campbell has applied to the study of the minerals in ores, the methods already developed and established in the investigations of alloys in metallography. Coming to the subject as an experienced worker in petrography as well, he has blocked out a line which will be of extreme significance as regards the order of formation, and the processes of alteration which characterize the metallic minerals which from their opacity have hitherto been difficult subjects.

Mr. L. C. Graton[1] has prepared a valuable paper on the gold and tin deposits of the southern Appalachians and to it Mr. Lindgren has added a chapter on Dahlonega, Ga. The first portion contains many details of mines in North and South Carolina; and the latter part, while treating of Dahlonega in particular, yet has some generalities on the Southern gold veins in the large way. Mr. Lindgren remarks the well nigh universal association of the veins with the contacts of intrusive granite on schist. Like so many of the Western deposits, they thus seem referable to the expiring stages of the igneous activity which developed the granite.

Much activity in the study of the geology of Alaska continues, and from the United States Geological Survey a number of reports have emanated which give much valuable information on the placers and other resources. The list below contains the titles published in 1906[2]. A. H. Brooks in Professional Paper No. 45 has prepared a general review of the geography and geology of the Territory.

Although published in 1905, the work of Prof. A. P. Coleman[3] upon the nickel-copper district of Sudbury was not referred to in the last review as the writer had not then seen the report; nor was proper mention made of the earlier investigations of Dr. A. E. Barlow[4]. As a result we now

[1] L. C. Graton, "Reconnaissance of some Gold and Tin Deposits of the Southern Appalachians," with "Notes on the Dahlonega Mines," by Waldemar Lindgren, Bulletin No. 293, U. S. Geological Survey.

[2] F. H. Moffit, "Mineral Resources of Kenai Peninsula," Bulletin No. 277. L. M. Prindle and F. L. Hess, "The Rampart Gold Placer Region," Bulletin No. 280. A. H. Brooks and others, "Report on Progress of Investigations of Mineral Resources of Alaska in 1905," Bulletin No. 284. A. C. Spencer, "Juneau Gold Belt," and C. W. Wright, "A Reconnaissance of Admiralty Island," Bulletin No. 287. L. M. Prindle, "Yukon-Tanana Region," Bulletin No. 295.

[3] A. P. Coleman, "The Sudbury Nickel Field." *Report* of the Bureau of Mines of Ontario, XIV, iii, 1905.

[4] A. E. Barlow, "Report on the Nickel and Copper Deposits, etc.." Geological Survey of Canada, XIV, part H., 1904. A summary of the questions of origin is given by Dr. Barlow in *Economic Geology*, 1906, I.

know that all the nickel and copper deposits are in the outer edges either of one great intrusive mass of norite or of relatively small offsetting dikes of the same. The largest deposits are associated with the largest mass. After its intrusion and while yet molten the magma is believed to have broken up into an upper, lighter and acidic portion and a lower heavier and basic portion. In the bottom of the latter the sulphides settled presumably because of their greater specific gravity. When the whole mass was subsequently folded in a synclinal trough the lower parts, which were enriched with ore, became the borders which often dip steeply toward the bottom of the trough. Some secondary rearrangement is admitted, but the greatly predominating cause was the one outlined above.

SAMPLING AND ASSAYING.

By F. F. Colcord.

This article is a summary of the important literature of 1906 relating to the general subject of sampling ores and minerals and the analytical methods for the determination of their constituents.

Sampling.

Hand Stamp for Preparing Mine Samples.—G. J. Bancroft (*Min. and Sci. Press*, XCII, 365) has written of an ingenious hand stamp which he has found useful in preparing mine samples. In mine sampling it is often desirable to take large samples and in the absence of crushing machinery, the crushing is done with hand hammers or sledges. To obviate their use the hand stamp was devised. It is made by shoeing a suitable beam or pole with heavy iron. The operation of the stamp is similar to pestle and mortar work with a piece of iron plate as the mortar.

Sampling Sands in Cyaniding.—Duncan Simpson (*Trans.* I. M. M., Oct. 18, 1906) discusses the sampling of treated and untreated sands in the cyaniding of gold. Various sampling methods were tried with the result that he finds the automatic sampler unreliable. The best method was to discharge half of the sand from the vat leaving a vertical face at the diameter. This face was divided into horizontal zones 1 ft. wide and the face of each zone was scraped into a bucket. The bucket sample from each zone was then assayed separately and collectively with the others. While this method cannot be used constantly, an occasional resort to it is well worth while. Among the methods of rod sampling was one which gave excellent results. A 40-ft. vat was divided approximately into eight equal parts by weight and the rod sample taken from the center of mass of each part. This was accomplished by inserting the rod perpendicularly at points 7.07, 12.24, 15.81 and 18.7 ft. distant from the center on each side along the diameter. The rod method has its limitations but it is possible by the foregoing method to obtain fairly accurate results. In sampling tailings, a sample taken at regular intervals with a hand scoop from the discharge belt is reliable.

Tailing Sampler.—R. G. Brown (*Min. and Sci. Press*, XCII, 542–543) gives a description of a tailing sampler in use in British Columbia. The sampler consists essentially of a box with a $\frac{1}{16}$-in. slot that is swung back and forth through the tailing discharge. The sampler is actuated by a rod connected to a crank shaft on which is mounted eccentrically a water

can. Water drips into the can until it overbalances and falls causing the shaft to turn and the sampler by means of the connecting rod is swung backward and forward across the discharge, which is through a vertical chute. The momentum is sufficient to return the can to its original position where it is held until filled again. The operation can be easily regulated by increasing or decreasing the flow of water to the can. The apparatus is extremely simple to construct and requires little attention.

Sampling Ore Containing Metallics.—C. C. Sample (*Eng. and Min. Journ.*, LXXXII, 362–363) states that the sampling of a lot of ore containing metallics is considered by some as a difficult and uncertain operation, but this is not necessarily so, if a slight deviation is made from the ordinary method. The departure consists of screening the ore after each crushing, before quartering down, and weighing the metallics and fines. The fines are then quartered down, crushed and screened as before, and the operation repeated until the final sample is obtained. The metallics from the various screenings are cleaned, sampled and assayed separately. From the assays of the metallics and the pulp the final assay of the lot is calculated according to the weights obtained, and with the use of the author's rules the calculations are simple.

F. F. Colcord (*Eng. and Min. Journ.*, LXXXII, 1164) has described the methods in use for the sampling and assaying of the Cobalt district ores. These ores are extremely varied in character and carry a considerable proportion of their silver in the form of metallics. The ore is crushed to 0.25 in. size, and to avoid the collection of the larger pieces on the circumference of the cone, as in the usual coning and quartering method of sampling, the ore is piled in a ridge about 30 ft. long. The ridge is halved by alternately shoveling to either side forming two parallel ridges, one of which is rejected and the other again halved until a 5000- to 6000-℔. sample is obtained. This sample is coned and quartered to about 1000 ℔. and that portion is rolled to pass an 8-mesh sieve. The metallics on this sieve are ground in a drug mill and mixed with the fines, and the sample quartered down to 20 ℔. After sifting on a 20-mesh sieve, treating the metallics as before, the sample is quartered to about 5 ℔., which is ground in a pebble mill and sifted on an 80-mesh sieve, resulting in metallics and fines. The metallics are assayed by scorification and the fines either by fusion or combination assay according to the grade. The fusion charge consists of 0.5 a.t. of ore and 200 grams of the following flux: Bicarbonate of soda, 10 parts; litharge, 20 parts; potassium carbonate, three parts; silica, three parts; borax glass, four parts. Enough flour is added to make a 30 to 40 gram button and salt is used for a cover. The combination assay is as usual except that the ore is dissolved in nitric acid alone and then diluted with water, and the residue filtered and scorified separately from the precipitated silver.

The New Denver Mint.—R. L. Whitehead and F. E. Healy (*Mines and Minerals*, XXVII, 1–4 and 54–57) have described the new Denver Mint. The building and equipment are fully up to Government standard and among the important innovations is the treatment of the old crucibles and sweeps. The treatment consists of wet mill grinding and amalgamation.

Conversion Tables.—G. T. Holloway (*Min. Journ.*, LXXIX, 44–45) has calculated tables for the conversion of percentage into ounces, or pennyweights or grains per ton of 2240 ℔. and also per ton of 2000 ℔.

Assaying.

Assaying Auriferous Tinstone.—C. O. Bannister (*Trans.* I. M. M., April 19, 1906) has compared the results obtained by various methods in the assaying of auriferous tinstone. The ores were from Lower Burma and contained from 54.2 to 72.0 per cent. of tin and from 0.25 to 26 oz. gold per ton. The methods employed were: (1) Scorification assay; (2) crucible assay; (3) wet method, dissolving the gold in aqua regia and then precipitating the gold; (4) concentration of the gold in part of the tin; (5) collection of the gold in the whole of the tin. In the last two methods the tin button obtained was dissolved in hydrochloric acid and the gold and silver residue cupelled. The crucible assay gave the best results with the following charge: Ore, 25 grams; red lead, 60 grams; charcoal, 1.5 grams; sodium carbonate, 40 grams; borax, 10 grams. With the use of a large amount of sodium carbonate and red lead the tin was slagged and the resulting lead button was clean enough to cupel without preliminary scorification. In ores carrying over 1 oz. gold it was necessary to rerun the slags. The slags were cleaned with red lead, 30 grams, charcoal 1.5 grams and sodium carbonate 10 grams. The scorification method invariably gave low results due to the unsatisfactory nature of the slags, while the crucible assay gave excellent results for both gold and silver. The wet method gave good results for gold (the silver was not determined) but the method is tedious and without any advantages over the crucible assay. The concentration in part of the tin gave low results partly owing to gold being left in the slag and also to loss on treatment of the tin buttons, while the silver results were unreliable. The collection of the gold in the whole of the tin was good with low grade material, but is unreliable for silver.

Assay of Gold Bullion.—A. Douglass (*Aust. Min. Stand.*, XXXV, 451–452) describes the assaying of gold bullion and pertinently remarks on the need of the sample being representative of the material. Segregation is very apt to occur, especially in low grade bars, but by thorough stirring of the molten metal, and by casting it in a thin bar, the segregation can be nearly eliminated. The usual method of gold bullion assaying is

followed except in the use of more copper in the cupelling charge and in the use of three parting acids.

COMPARISON OF GOLD ASSAYS OBTAINED BY THE DIFFERENT METHODS.

Ore.	Tin.	Scorification.			Crucible.			Wet.			Concentration in Tin.			Collection in Tin.	
	%	Oz.	dwt.	gr.	Oz.	dwt.	gr.	Oz.	dwt.	gr.	Oz.	dwt.	gr.	Oz.	gr.
No. 1.........	64.2	0	5	5	0	5	21	0	5	5	0	5	5	0	21
" 2.........	72.0	1	5	11	1	6	18	1	6	3	1	5	11	1	18
" 3.........	69.4	4	10	4	4	16	16	4	18	0	4	5	0	4	19
" 4.........	54.2	9	2	22	10	11	15	10	1	5	8	10	16	9	0
" 5.........	62.1	24	16	12	25	19	9	26	2	16	24	11	7	25	8

Antimonial Gold Ores.—Wm. Kitto (*Bull.* I. M. M., No. 26) takes up the various methods of assaying antimonial gold ores. A comparatively pure stibnite is taken and silver and gold in the form of a solution of these metals added. The ore in this manner was made to contain 10 oz. each of gold and silver per ton of 2240 ℔. Method 1*a* was a partial oxidizing fusion with niter. The ore had a reducing power of 6.5 and charge was 326.6 grains of ore, 2000 grains of litharge, 400 grains niter, 200 grains soda, 100 grains borax and 200 grains silica. The charge was fused slowly at a low temperature in order not to reduce antimony. If the button showed much antimony it was scorified (method 1*b*).

Method 2 was to fuse 326.6 grains of ore with 550 grains niter, 500 grains soda and 250 grains of borax until quiet and then add 550 grains litharge, 25 grains charcoal and 50 grains soda and finish fusion.

Method 3 was to scorify 100 grains of ore with 1500 grains lead and 5 grains borax in a 2.75-in. scorifier, twice.

Method 4*a* consisted of roasting the ore with the addition of silica and fusing the residue with 1000 grains litharge, 15 grains charcoal, 1000 grains soda and 400 grains borax. Method 4*b* consisted of roasting the ore with the addition of charcoal and fusing the residue with 500 grains litharge, 20 grains charcoal, 500 grains soda, 100 grains borax and 100 grains silica.

In each case the slags were rerun with usually the following charge: 300 grains litharge, 15 grains charcoal and 50 grains soda. The silvers were cupelled with checks.

ASSAYS OF ANTIMONIAL GOLD ORES.

Method.	Gold.			Silver.			Method.	Gold.			Silver.		
	Oz.	dwt.	gr.	Oz.	dwt.	gr.		Oz.	dwt.	gr.	Oz.	dwt.	gr.
1*a*.....	10	0	0	9	19	0	4*a*.....	9	19	0	9	8	0
1*b*.....	10	0	0	9	18	0	4*b*.....	9	8	0	9	4	0
2......	9	18	0	9	15	0		9	7	0	9	9	12
3......	9	19	6	9	16	0		9	6	0	9	4	12

The partial niter oxidizing fusion gave the best results but as E. A. Smith (*Bull.* I. M. M., No. 27) remarks there are many details lacking, to judge the whole. It would have been interesting to have used typical ores and tried the effect of various amounts of ore, of the nature of the gangue and of different fluxes.

C. O. Bannister (*Bull,* I. M. M., No. 27) agrees with the author of the above as regards the crucible method being a good method. In method 2 he prefers to charge the ore, with sufficient niter to render oxidation complete, very slowly into a hot crucible and then scrape out the oxidized mass before it melts. While still hot it is ground in an iron mortar and mixed with the necessary red lead, charcoal and fluxes for the fusion. With this method there is less need of cleaning the slags on medium or low grade ores. He has also found roasting in some cases to give admirable results both with the use of silica and carbon. He has cupelled 5 per cent. antimonial lead buttons in patent cupels without injuriously affecting the gold and silver.

G. T. Holloway (*Bull.* I. M. M., No. 27) in connection with Mr. Kitto's paper describes his fusion method. The charge consists of ore 20 grams, soda ash 60 grams, borax 20 grams, red lead 80 grams and niter 14 grams. The soda slags the antimony and a 50-gram lead button is obtained with a particular ore. It is necessary to scorify the lead button but the slags on material under 2 oz. in gold are practically free of value. In the presence of silver, however, the slags are retreated with 20 grams of red lead and three grams of flour. He uses clay instead of silica in the charge to protect the crucible.

Determination of Silver in Tin-Copper Alloys.—P. J. Thibault (*Aust. Min. Stand.,* XXXV, 475) has experimented with methods for the determination of silver in argentiferous tin-copper alloys. Fusion methods all resulted in obtaining coppery buttons and a heavy loss of silver in the slags. Calcining the alloy followed by a fusion gave similar trouble in the fusion. He finally arrived at the conclusion that the scorification method was the only correct and feasible method for the silver determination. The author neglected to mention, however, the danger of collecting impurities in the silver bead by this method, especially bismuth from the test lead.

Assaying in South Africa.—A. Whitby (*Journ.* Chem. Met. and Min. Soc. of South Africa, March, 1906, 264–273 and Aug., 1906, 33–35) has an excellent paper dealing with the general conditions of assaying, especially those in South Africa. In regard to general conditions he says: "Good work is a resultant of good working conditions and among these I class the equipment and design of the laboratory." The location of the laboratory should be given careful thought. He calls the attention of the assayer to the relationship between the accuracy of the pulp balance

and its weights on the one hand and the assay balance and its weights on the other. The use of the bucking plate he condemns as wasteful in energy and material. The evaporation of slimes samples to dryness is attended with a greater or less loss of gold deposited on the sides of the evaporating dish, or absorbed by it. To overcome this loss a little cyanide is added to the slimes residues, then small quantities of copper sulphate, dilute sulphuric acid and sodium sulphite and the sample after vigorous shaking is turned out into a shallow enamelled pan and dried. Some of of the charges which he has found to give good results are presented in the accompanying table:

WHITBY'S ASSAY CHARGES.

Material	Ore A.T.	Litharge A.T.	Sodium Carbonate A.T.	Borax A.T.	Charcoal.	Button Obtained Grams.
Low grade pyrite	1.0	1.0	1.0	0.25	Sufficient	35
50 per cent. calcite	1.0	1.25	.33	....	2	20
Screen and pulp samples	2.5	3.0	2.5	0.50	Sufficient	30 to 60
Spitzlutte concentrates	1.0	2.0	1.0–1.25	0.25	None	30 to 60
Sand charges and residues / Slime charges and residues	2.5	2.0	2.5	0.50	Sufficient	30 to 60 / 30 to 60
Sand with pyrites	2.5	3.0	2.5	0.75–1.	None	....
Black sands	.5	1.5	1.0	0.25	None	25–28
Black sands	1.0	3.0	2.0	0.50	None	50–58
		Red Lead				
Black sands	1.0	2.5	2.0	.50	None	35–40
Frue vanner concentrates	.5	2.0	1.5	.50	None	50

In reply to the question of the value of borax in the charge these observations were made, that when borax was omitted: (1) the pot requires constant watching or the charge will creep over the side; (2) the fusion is considerably retarded; (3) greater heat is required than is usually attainable; (4) the slag is in consequence pasty and flows badly; (5) the other fluxes must be increased unduly to the detriment of the life of the pot; (6) comparison has shown no higher results obtainable:

EFFECT OF VARYING QUANTITIES OF FLUX.

Rough Estimate of Pyrites.	Pb O	Na_2CO_3	$Na_2B_4O_7$	Weight of Button Grams.	Appearance of Button.	Appearance of Slag.
8%	3.0	1.0	0.5	22	Clean	Yellow
	2.0	1.0	0.5	25	Sulphury	Patchy
	1.5	1.0	0.5	21	Very sulphury	Opaque & black
	1.5	1.5	0.5	27	Little sulphur	Clear
	1.5	2.0	0.5	30	Clean	Clear
10%	2.0	1.0	1.0	17	Very sulphury	Black opaque
	2.0	1.5	1.0	20	Sulphury	Clear
	2.0	1.0	0.5	21	Sulphury	Yellow & patchy
	2.0	1.5	0.5	29	Clean	Clear
	2.0	1 5	0.25	29	Clean	Clear
	1.5	1.5	0.5	26	Sulphury	Black
	1.5	2.0	0.5	28	Little sulphur	Fair
15%	2.5	1.5	0.5	38	Little sulphur	Fair
	2.5	2.0	0.5	44	Clean	Clear
	2.5	2.5	0.5	43	Clean	Clear
	2.5	3.0	0.5	45	Clean	Clear

The effect of varying amounts of sodium carbonate, litharge and borax on the nature of the slag and button are shown in the accompanying table:

E. J. Laschinger (*Journ.* Chem. Met. and Min. Soc. of South Africa, June, 1906, 367–369) has carried out a tentative plan for a laboratory suggested by A. Whitby in the course of his paper. A sketch is shown of the arrangement of the building and the estimate calls for a brick building costing £2000.

Assay of Gold Bullion.—A. C. Claudet (*Bull.* I. M. M., No. 27) describes his method of assaying gold bullion. He takes up the preparation of the assay piece, the rough weighing of the same and the fine weighing. The amount taken is 0.5 gram = 1000 milliêmes and is weighed by a substitution method, by placing the 1000 milliême weight on the balance and exactly counterpoising it. The milliême weight is taken off and replaced by the sample which is filed to exactly balance the counterpoise, wrapped in 4000 milliêmes of lead foil with 2500 milliêmes of silver and 35 milliêmes of copper and cupelled, 20 assay pieces at a time, including one check. The cupellation takes about eight minutes and the cupels are removed from the furnace while the metal is still molten. The buttons are cleaned, rolled and annealed as usual, and parted in the customary platinum apparatus, the first acid being 1.16 specific gravity and the second and third acids 1.25. The time in the second and third acids is 20 minutes each, after which the tray is drained, washed twice with distilled water, drained, dried and annealed. The cornets are weighed by the substitution method, adding weights to the cornets to balance, the surcharge amounting to 0.7 to one milliême. Precautions are taken as to the accuracy of the balances and the weights, the freedom of the lead, silver and copper from gold and the absence of hydrochloric and sulphuric acids in the nitric acid, as well as making the nitric acid solutions of the correct specific gravity.

The proof gold for checks is made by dissolving cornets in aqua regia, expelling the free acid and diluting with water. The solution is allowed to stand for three weeks and is then siphoned off through a filter and the gold precipitated with oxalic acid. The precipitated gold is washed with boiling hot water, hydrochloric acid, water, ammonia and water in the order named. It is then dried and melted, the resulting button cleaned with water and hydrochloric acid, remelted in a plumbago pot, and cast in a small clean mold. The bar is also washed, cleaned and rolled into strips, and cleaned again before using.

J. E. Clennell (*Eng. and Min. Journ.*, LXXXII, 1057-59) has written of the methods of sampling and assaying bullion as practiced at the Redjang Lebong mine, Sumatra. Selenium is found in the ores of that mine, and to avoid any interference of that metal in the assaying, the following methods are adopted: The selenium, presumably present as

silver selenide, follows the precious metals through the process and is found in the zinc box precipitate. The precipitated metals are treated with sulphuric acid, washed and smelted, thus eliminating much of the selenium and leaving 1 to 2 per cent. in the bullion. The bullion is fairly homogeneous, drill, dip and chip samples from various parts of the bar agreeing very closely. The bars weigh about 600 oz. troy and measure $3\frac{3}{8}$ in. thick, 9x4 in. on top, and $8\frac{1}{8}$x$3\frac{1}{4}$ in. at bottom. Dip samples are taken with a plumbago spoon and the sample granulated in water, the molten metal being thoroughly stirred before the sample is taken. Drill samples are taken with a 5/64-in. drill running at 1680 r.p.m. Two holes are drilled in the top 1 in. from the long side and 1 and 4 in. respectively from the end, while two holes are similarly drilled in the diagonally opposite corner of the bottom. Any foreign material is removed, and the larger pieces are cut or ground up. The dip samples are flattened with a hammer, cleaned and cut up.

Duplicate portions of 500 mg. are weighed from each sample with an accuracy of .05 mg. and each wrapped in 5 grams of lead-foil. A gold and silver check-piece is weighed for each set of duplicates of the approximate gold and silver composition of the bar, being similarly wrapped in lead-foil. The assays are cupelled in batches of twelve, in four rows of three each, each row consisting of duplicate assay pieces and their check. The position of the check varies in each row. Morganite No. 5 cupels are used, and when thoroughly heated in a charcoal-fired furnace, 10 grams of lead are dropped into each cupel. When the lead is cupelling freely the assay and check-pieces are dropped in the cupels and cupellation is finished, the door being closed near the end of the operation. When cupellation is complete, the cupels are drawn to the front of the muffle and allowed to stand until the beads have brightened and solidified. The beads are removed from the cupels, cleaned and weighed, and prepared by rolling and annealing for parting. The parting is carried on in porcelain crucibles with one part nitric acid (1.42 sp.gr.) and seven parts of water. In this first treatment the acid is brought to a boil, is then kept just below the boiling-point for 15 to 20 minutes, and is then again boiled. The acid is now drained off and replaced with stronger acid, three parts nitric acid (1.42 sp.gr.), and one part water. The same procedure as in the first instance is followed, except that the time during which the acid is kept just below the boiling-point is extended to 25 minutes. The pieces are washed, annealed and weighed, and correction is made as shown by the check-piece both for the silver and the gold. The high proportion of lead to the amount of sample taken (30:1) is necessary to remove the selenium during cupellation. The author has found that the amounts of selenium, copper and zinc present in his bars do not affect the results, and they are usually omitted in making up the check-piece.

C. Hoitsema (*Zeit. Anal. Chem.*, 1906, XLV, 1-14) has described a method for sampling and assaying large quantities of silver coins. The coins are sampled down to 1000, from each of which is punched a small portion, varying in size and shape according to the particular coin. The punchings are then sampled down to the requisite amount. Stas's modification of the Gay-Lussac method is used in the silver determination, except that 20 grams are taken instead of 1 gram. Correspondingly 2000 c.c. and 500 c.c. pipettes are used, and, 5 grams of silver are taken for the proof. With proper care in observing the necessary precautions, an accuracy within .000001 is obtainable.

L. Schneider (*Oest. Zeit. f. B. u. H.*, 1906, LIV, 81-84, 96-100), has written a review of the development of gold assaying, especially of silver and lead alloys.

Determination of Lead.—D. J. Kelley (*West. Chem. and Met.*, II, 127–130) compares the fire and wet methods for the commercial determination of lead. The principal weakness of the fire assay is its absolute unreliability in all cases where the ores treated are complex or low grade. Sulphide ores of a refractory nature, high in zinc and iron and low in lead rarely indicate by fire assay a fair percentage of the amount of lead found by wet methods, while chemical analysis of the buttons shows large amounts of impurities. With the wet methods, of which there are several good ones, accurate results can be obtained. The author believes that the fire assay should be abandoned as has been the dry copper assay.

Assaying Sulphide Ores.—W. G. Vail (*West. Chem. and Met.*, II, 14–15) advocates the niter assay for sulphide ores, as one is able to obtain uniform buttons, can use large quantities of litharge which helps to make a liquid slag and will not be troubled with the burned off ends of nails in the lead button. He has not found that niter causes any unusual boiling of the fusion. One particular advantage of the use of niter is that in arsenical ores there is not any speiss formed as in the nail method.

Assaying Copper at Lake Superior.—C. W. Macdougall (*Eng. and Min. Journ.*, LXXXI, 654–655, 708–709, 806–807) describes in detail the methods of determining the copper present in the waste sands, concentrates, slimes and slag of the Quincy mine. The methods of sampling and preparation of the samples are thoroughly treated. Both the wet and fire methods are used for the determination of the copper, but on different classes of material. The fire assay is used on the high-grade concentrates. The flux is 120 grams of cream of tartar, 80 grams of fused soda and 75 grams of borax glass. Fifty grams of ore are mixed in a No. 5 Hessian sand crucible with 30 grams of flux and covered with 10 grams of crucible slag from previous assays. Four covered crucibles are placed in a hot furnace and surrounded with soft coal and after five min. the fire is poked to create a draft. At the end of eight min., more coal is added,

in 13 min. the fire is again poked and at the end of 17 min. the crucible is removed if reaction is complete. The button is allowed to cool in the crucible. The keeping of records and reporting of assays is carefully shown.

Assaying Cyanide Solutions for Gold.—C. H. Jay (*West. Chem. and Met.*, II, 143) treats of several methods of assaying cyanide solutions for gold. Method 1 consists of evaporating 0.5 to 1 a.t. of solution in lead foil boats, fusing in a crucible, cupelling and weighing. Its limitation is the small amount of solution that can be taken. Method 2 is to evaporate 5 a.t. of solution in a porcelain evaporating dish, with a sprinkling of litharge over the solution at the start to give sufficient lead for the lead button. The evaporation is done over steam and the residue brushed into a crucible and assayed as usual. The method is long and tedious, but gives good results. Method 3 is as follows: to 5 to 10 a.t. of solution in a 600 c.c. beaker, with enough of potassium cyanide to bring the solution up to 4 or 5 ℔. of cyanide per ton, add 10 c.c. of a 10 per cent. solution of lead acetate, and 1 to 2 grams of zinc shavings. The whole is brought to a boil, and 15 c.c. of hydrochloric acid added and the solution allowed to stand on a hot plate for 15 min. With the use of rubber cots on the fingers the precipitated lead and gold is gathered into a ball, squeezed, placed in lead foil and cupelled. If necessary silver is added to obtain the ratio for parting. The method is rapid, accurate and easy to manipulate.

Solubility of Platinum.—J. F. Thompson and E. H. Miller (*Journ.* Am. Chem. Soc., XXVIII, 1115–1132) studied the solubility of platinum when platinum-silver alloys are treated with nitric acid. Previous workers have obtained very conflicting results as Winkler (*Zeit. f. Anal. Chem.*, 1874, p. 368) finds the strength of acid of little importance, while J. Spiller (*Proc.* Chem. Soc., 1897, XIII, 118) uses acid of 1.42 specific gravity as the best solvent, and dissolves from 0.75 to 1.25 per cent. of the platinum. H. Rose ("Handbuch Analyt. Chem.," 6th Ed., II, 226) states that, at the most, only 10 per cent. went into solution with the silver and W. W. Perry (*Eng. and Min. Journ.*, 1879) finds that the platinum can be completely dissolved. The writers have studied the melting-points and cooling curves, microstructure, specific gravity and electrical conductivity of a series of alloys in an endeavor to account for these results. The platinum used was purified to free it from iridium and the alloys were carefully prepared in order to make them homogeneous. The standard sulphuric parting method of analysis was used to determine the platinum and silver. It was found, however, that in decanting the silver solution fine particles of platinum were carried off, so that filtering was necessary, and secondly, that on the higher platinum alloys, the silver retained in the residue amounted to considerable.

ASSAYS OF PLATINUM-SILVER ALLOYS.

Pt. supposed	10.00	20.00	30.00	40.00	50.00
Pt. actual	10.39	20.59	31.46	37.89	57.05
Ag. retained by Pt.	tr.	0.59	0.98	2.24	2.70
Ag. in filtrate	89.54	78.62	67.51	59.81	40.26
Summation	99.93	99.80	99.95	99.94	100.01

Delépine (*Compt. rend.*, 1906, CXLII, 631) claims that considerable amounts of platinum are dissolved in boiling concentrated sulphuric acid on treatment for 40 to 50 hours, but the authors found that for the short time they used, only unweighable amounts of platinum went into solution. Nitric acid of 1.1 and 1.4 specific gravity was used in the parting tests and the results clearly show the impossibility of separating platinum from gold, iridium, etc., when alloyed with silver by means of one parting with nitric acid. The results with 1.1 sp.gr. acid are given in the accompanying table:

PARTING PLATINUM-SILVER WITH ACID OF 1.1 SP. GR.

% Pt. in alloy	10.39	20.59	31.46	37.89	57.05
% residue	3.86	8.58	36.59	49.13	65.16
% Ag. in residue	0.27	1.81	12.09	13.64	12.19
% Pt. in residue	3.59	6.77	24.50	35.49	52.97

As will be noted there is always some silver left in the residue. From the thermal data and microstructure it was concluded that the alloys consist of a variety of platinum-silver compounds, their relative proportions varying with the rate of cooling, and upon whose individual solubilities depends the solubility of the final alloy as a whole. The existence of these platinum-silver compounds of varying solubilities is probably the explanation of the irregular results obtained in parting with nitric acid.

A. Jarman (*Trans.* I. M. M., May 17, 1906) has presented a form of platinum parting apparatus devised by Professor Liversidge that commends itself both for its practicability and for its low cost. In this apparatus the thimbles have a lip or edge turned on the top by which the thimble is supported in the tray. The accompanying plan and photograph (Figs. 1 and 2) show the construction. A larger tray would necessitate additional stiffening. In comparison with the apparatus described by Percy[1] and Rose[2] this form admits a freer circulation of the acid, and the washing and draining are both more readily performed. The pin apparatus in use at Sydney Mint (Fig. 3) is still better in these respects, but this form does not commend itself for general use. Professor Liversidge's apparatus effects a saving of about 45 per cent. in platinum in the

[1]Percy, "Metallurgy of Gold and Silver," Part 1, p. 263.
[2]Rose, "Gold," p. 415.

tray over the old forms. A nine-hole tray without the thimbles weighs about 12.5 grams. The foil is .006 in. thick and the legs are riveted or soldered and extend 2 mm. below the thimbles. A glass beaker is used for boiling.

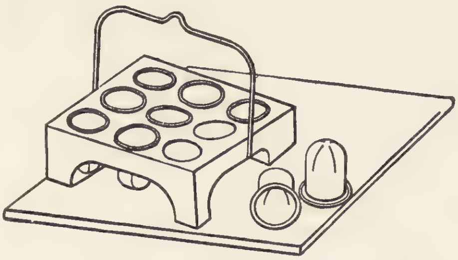


FIG. 1.

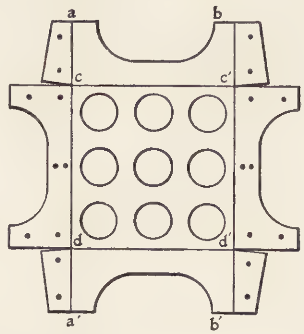


FIG. 2.

PROF. LIVERSIDGE'S PLATINUM PARTING APPARATUS.

E. A. Smith in connection with the foregoing paper presents another form of apparatus that has the same good points as Professor Liversidge's. It is constructed of platinum wire of about No. 17 B. w.g. the pieces being soldered together. The frame of such a form, strongly constructed for constant use, weighed 54.5 grams; the twenty cups, 4 grams each. A similar six-cup frame weighed 25.5 grams and the six cups 22.5 grams.

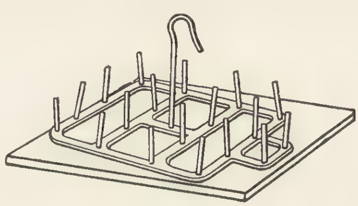


FIG. 3.—APPARATUS IN USE AT SYDNEY MINT.

Notes on Assaying.—C. H. Fulton (*West. Chem. and Met.*, II, 155–166, 169–177, to be continued) published some notes on assaying that were prepared for use at the South Dakota School of Mines, but are also of interest to others. The reduction and oxidation reactions are taken up and the author then exhaustively treats of the formation and calculations of crucible assay slags. The notes so far published give great promise for the whole when completed.

Apparatus.

Equal Arm Balance.—J. H. Gottschalk (*West. Chem. and Met.*, II, 37–44) has translated the results of Carpenter and Bisbee's work (*Physical Review*, Jan., 1906) on the equal arm balance into non-mathematical statements. The authors deduce an equation containing the length of

the beam, the weights of the beam and pan, the weight on one pan, the excess weight on the other pan and the deflection of the pointer expressed in terms of the ratio of the length of one scale division to the pointer length. By means of this equation and with two observations with different loads a complete curve is drawn showing the relation of the load to the sensibility. Also from these observations they calculate the position of the center of gravity of the beam and of the whole pan

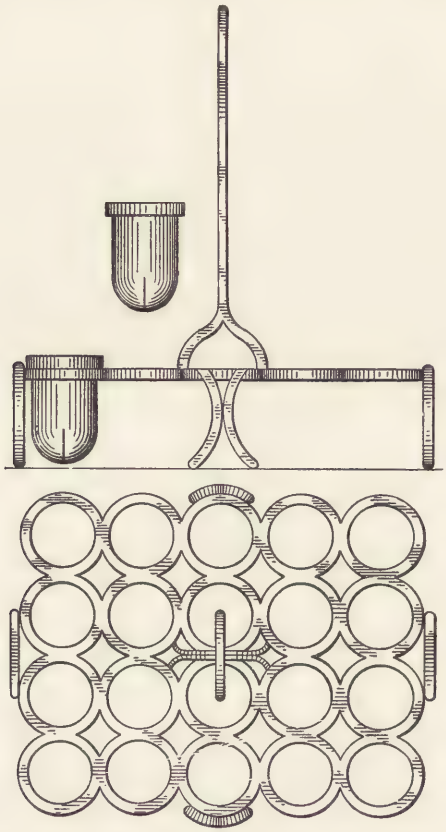


FIG. 4.—PLATINUM PARTING APPARATUS, SUPPLIED TO THE SHEFFIELD ASSAY OFFICE.

system inclusive of the beam, and from the curve certain peculiarities of the balance when weighing can be predicted. These curves are shown in Fig. 5.

Curve I is that of a short high truss beam that is absolutely rigid and whose knife edges are all in the same plane. Curve II illustrates a balance with the same kind of a beam as I, but with the plane of the terminal knife edges below the central knife edge plane. Curve III is a short beam high truss balance with the terminal knife edge planes above the

plane of the central knife edge. Curve IV is an untrussed beam balance with knife edges all in the same plane but under increasing load the beam is distorted and the terminal knife edges assume a lower plane than the central knife edge and the curve resembles II. Curve V is the same type of balance as under IV except the terminal knife edge planes are higher than the central. Curve VI is the same as V except with the sensibility raised unduly high.

From these curves we may deduce the following: (1) With all three knife edges in one plane, the sensibility is independent of the load. (2) With the terminal edges higher, the sensibility first increases, goes through a maximum, and then decreases steadily. (3) With the terminal edges lower, the sensibility decreases steadily. (4) The beam is always deformed some by loads, superimposing a greater or less decrease of sensibility in the cases 1, 2 and 3. With short, high truss beams (4) is least.

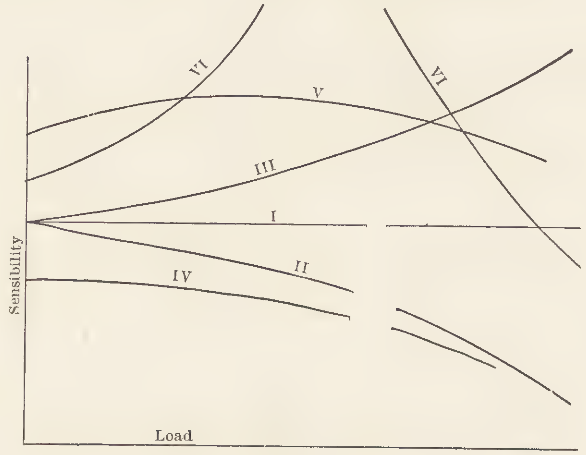


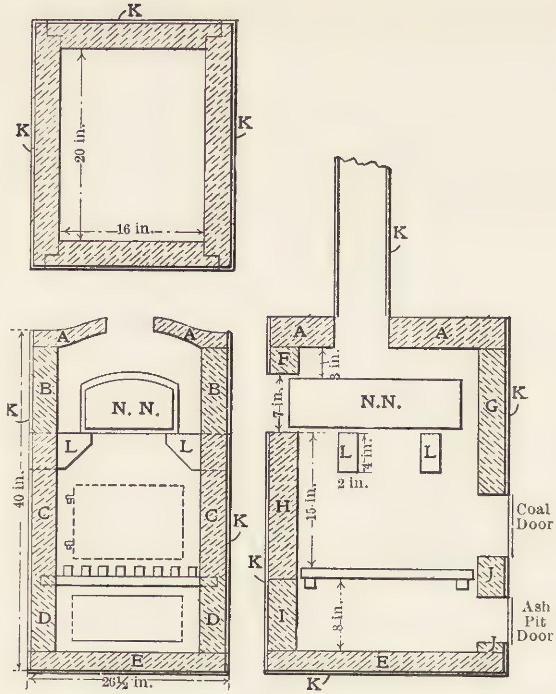
FIG. 5.

Use and Care of the Balance.—J. H. Gottschalk (*West. Chem. and Met.*, II, 55–69, 83–87, 91–102) has compiled some notes dealing with the general principles underlying the use and care of the balance. A detailed explanation is given of the following methods: Determination of the Zero Point of a Balance. Determination of the Sensibility. Determination of the Relative Length of the Arms. Determination of the Sensibility Curve. The interpretation of all modes of performances of which balances, good or bad, are capable is then taken up including directions for the various adjustments, discussions on the accuracy attainable under given conditions and on the accuracy required for the different kinds of work. The author gives much information about the balance that some obtain only after years of experience and this should be of value to all intelligent users of balances.

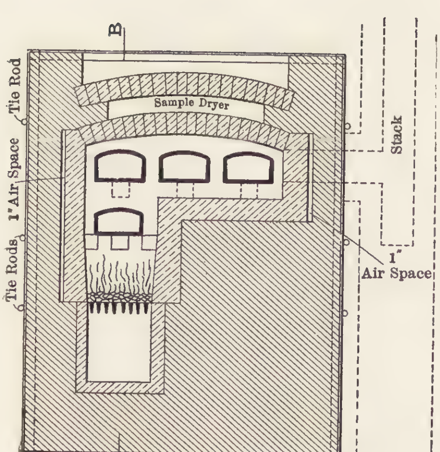
Assay Furnaces.—E. H. Nutter (*Min. and Sci. Press,* XCII, 329) has given his experience with a wood-burning assay furnace. He has found the furnace to be very efficient and to effect a considerable saving over the former practice of using a pot furnace and muffle furnace with charcoal as a fuel.

H. J. Reese (*West. Chem. and Met.*, II, 70–72) has devised a portable coal-fired muffle furnace of simple construction and of small cost. It is as portable as a coke furnace and more satisfactory to use as well as cheaper. The first cost is low, it is easy to erect and has a large capacity. The body of the furnace consists of a shell of sheet iron or steel, lined with 0.25 in. asbestos board and then with fire clay tile. The construction is shown in Figs. 6–8.

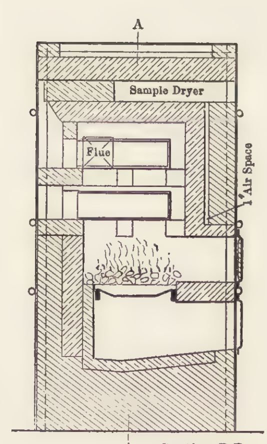
J. I. Brown (*West. Chem. and Met.*, II, 178–181) has brought out a four or five-muffle assay furnace which is recommended where a large capacity is desired. In the upper and lower muffles over the fire box, fusions may be carried on, while in the next right-hand muffle scorifications, and in the next cupellation. The furnace is economical and being lined with fire clay tile is easy to repair. Above the muffles is a space for drying samples. The furnace is fired with coal under forced draft. The construction of this furnace is shown in Figs. 9 and 10.



FIGS. 6–8.



SECTION A A
FIG. 9.



Section B-B

FIG. 10.

THE ADVANCES IN ORE DRESSING IN THE LAST DECADE.

By Robert H. Richards.

Eleven years ago I made a trip through the principal milling centers of the United States to gather material for my work on "Ore Dressing" which appeared in 1903. The advances made since 1895 have been considerable, and as a consequence I have engaged upon the preparation of material for an appendix. It seems fitting that this year's article should contain some statement of progress covering this period.

For *coarse breaking* the Blake jaw breaker, the Gates gyratory breaker and the Dodge jaw breaker are still standard. If anything, the Gates has found more favor, especially where a large crushing unit is desired.

The improvements in *rolls* have been more along the line of perfection and strengthening the parts, rather than in any radical changes. For fine crushing rigid rolls without springs have been installed in many instances, and appear to be giving satisfaction. The type of high speed, narrow faced rolls appears to have lost ground and the tendency is, at the present time, to use moderately wide-faced rolls.

The use of *steam stamps* has been discontinued in all places except at Lake Superior, where they are especially adapted to the crushing of native copper rock. In that district the old single cylinder stamp has given way in many instances to the steeple-compound Nordberg stamp which has yielded increased capacity and greater efficiency of power. A cross-compound steam stamp was tried in one mill but did not give satisfaction. The use of small steam stamps for fine crushing in gold milling is very limited.

The *gravity stamp* has been improved materially in detail rather than in any radical change of principle. There is a tendency toward the use of heavy stamps especially in South Africa, where the problem is to obtain as high a tonnage as possible. This tendency toward heavy stamps has not been felt to any great extent in this country as yet. The details which have been improved are the foundations, which have been made of masonry and concrete, iron anvil blocks which have been put under the mortars, frames and bins which have been made of steel and concrete, and other minor parts, such as guides, etc. Where electricity is available several motors have been installed in some mills to run separate batteries in preference to using one single large motor. The single stamp mills, such as the Nissen and Merralls, have found considerable use, especially where a

rapid discharge and high capacity is desired, for example, where concentration is the object rather than amalgamation.

Of the *fine crushers*, those that are most used are Chile mills, Huntington mills, amalgamating pans, ball mills and tube mills. The Chile and Huntington hold their own for regrinding middlings. The use of the old amalgamating pan has been revived, especially in Australia, where modified Wheeler pans find extensive application for regrinding coarse sands in cyanide plants. The use of ball mills has not increased to any great extent but tube mills have had very rapid growth and their use is widely distributed. This is due to the change in cyanide practice which calls for extensive sliming of the ore for cyaniding, and the tube mills appear to be the most satisfactory machine to do this work. Of course they are not adapted to pure concentration work.

For *sizing*, trommels are almost universal. Several makes of shaking screens are upon the market and have been installed in many cases, but for some reason their use does not appear to gain. The limit of fine wet screening has been considerably extended beyond the former limit of 1 or 2 mm. by the introduction of the Callow traveling belt screen and the the King screen. It is now claimed that wet screening can be successfully carried out down to $\frac{1}{8}$ mm. and even finer. This is an important step in advance because if such screens work satisfactorily they may replace classifiers which are less positive in their work than screens and require considerable water.

Various forms of *classifiers* have appeared. The Richards classifiers of both the pipe vortex and the annular vortex variety have been successfully used in various places. The Dorr scraping classifier is one of the many special forms. For fine settling the Callow tank, the Merrill tank and the Sherman tank are doing excellent work. Mention might also be made of the Woodbury classifying jig, first installed at the Calumet & Hecla mill, which has the function of eliminating fines at the very start of the washing process. The settling and clarifying of slimy water has been well developed in Arizona, where the scarcity of water requires economy in its use. I have continued my investigations upon the laws of free settling and hindered settling, and have in preparation papers giving the results of my investigations. These will shortly appear in print. Mr. Bring of Sweden has also contributed the results of his experiments upon the work of classifiers and jigs.

The Harz *jig* is the standard of the country to day. The Collom jig, which found extensive use at Lake Superior and in Montana 10 years ago, has now almost entirely disappeared. Its place has been taken at Lake Superior by a new form of accelerated jig which may be termed the Lake Superior *jig*. Another type of accelerated jig, the New Century, has come somewhat into use, but the greatest departure in jigging is in the

use of the Hancock jig which originated in Australia and has spread so that it is represented in the majority of the districts of this country. Its use, however, is still largely experimental and it has not as yet worked out its final place in the mills.

For *fine concentrators* the field is held by riffle tables of the Wilfley type, round tables and belt vanners. The more important makes which follow the Wilfley are the Card, Pinder, Overstrom, New Standard and New Woodbury. These all follow the Wilfley principle but differ in details of mechanism or of tops and riffles. This type of riffle table appears to have worked out its place in the mills. Its coarser limit is up to 1.5 mm. in some places, while at the fine end it is able to treat everything but slimes. That is to say, the fine material which is of a granular character may be successfully separated provided the water quantities are reduced and coarse material is eliminated. The Frue vanner is still a standard machine for fine work and the end-shake vanner is probably the best machine for the final enrichment of fine stuff such as canvas table concentrates. The Johnston suspended vanner is finding favor as is evidenced by a large installation in Utah. Wilfley slimers or canvas plants are the most satisfactory apparatus for concentrating slimes. I have made a study of the principles of the Wilfley table and the results will be the subject of a paper shortly to be published. The weaknesses of this table in treating, let us say, a natural product, are three fold. 1. Coarse grains of concentrates are carried down into the middlings. 2. Fine grains are carried down into the middlings and tailings. 3. Slimes are washed directly across the table, carrying values with them. These weaknesses are known to the mill men and are overcome in various ways. The Sherman plan is to employ close classification, sending spigots to different tables and returning the middlings of the tables back to the classifier. The Callow scheme is to eliminate fines to be treated separately. Another scheme which is much used and gives excellent results is to run the Wilfley table in conjunction with the round table, the round table making clean tailings and concentrates which go to the Wilfley; the dirty tailings of the Wilfley are sent back to the round table.

Among the *miscellaneous processes*, magnetic separation has shown a steady growth. For eliminating pyrites from zinc concentrates, the roasting and magnetic process is extensively used. Pneumatic concentration is limited to a few localities. A recent invention in this line is the Sutton-Steele table, which is reported to do satisfactory work. The electrostatic process has been tested out in various mills, and under special conditions appears to have made a field for itself. The various flotation processes are reported to be doing good work in Australia, but their use in this country has not extended beyond the testing stage as yet. It seems that some of our difficult problems may be solved by this method of concentration.

The Elmore oil process does not appear to have made any great advance.

In *general milling practice* the greatest advance has been made in the Southwest, i. e. in the copper camps of Arizona and Sonora, where large concentration mills have been erected in recent years. The peculiar conditions in that district, namely, high price of materials and scarcity of water, have led to a high standard of concentrating work.

The country as a whole appears to be tending toward larger mills; whereas 10 years ago the Calumet & Hecla company was the only one which had a capacity of 4000 tons a day, there are at the present time several others either in active operation or under construction which will come up to that figure. The extensive use of capital in developing low grade copper deposits at Ely, Nev., and at Bingham, Utah, is one of the features in this increase.

The changes in Montana practice have been along the line of increasing the size of the plant and increasing the economies of operation by the installation of improved machinery wherever possible.

Similarly, at Lake Superior, increased economy has been obtained by increased capacity and reduction of the losses at numerous points. The most important point of saving is in the arrangements for coarser crushing and for the recovery of large copper nuggets, before they are abraded into slimes. Graded crushing by breakers has had a thorough trial in one mill, and the results have not been so satisfactory as those obtained by the use of steam stamps.

In Idaho one mill is reported to be trying the experiment of jigging only one size of material, with the idea that the saving in the use of trommels, elevators, etc., will offset the increased loss in the jig tailings. In other words, the cost of building and operating a simple mill jigging a natural product (unsized) is so low that the company can afford to undergo a little greater loss of values than the increased running cost of a more complex mill.

For *testing apparatus* the sizes of holes in screens are receiving considerable attention, and it is to be hoped that the result will be to bring some sort of order out of the present chaotic and antiquated methods of designating screens.

PROGRESS IN ORE DRESSING AND COAL WASHING IN 1906.

BY ROBERT H. RICHARDS AND CHARLES E. LOCKE.

CRUSHING MACHINERY.

Hathaway Breaker.[1]—The Hathaway Gyratory Breaker made by the C. L. Hathaway Company of Denver differs from the ordinary Gates style of rock breaker in having the crushing head and liners near the lower bearing for the vertical shaft, while the eccentric bevel gears and counter shaft are at the top of the machine. In principle this breaker is related to the Gates in the same way that the Blake jaw breaker is related to the Dodge. It gives the maximum movement on the largest lump while the Gates has the largest movement on the smallest lump. Treating Gunnison granite of 5-in. size, a No.1 breaker gave the capacities shown in the accompanying table.

HATHAWAY GYRATORY BREAKER.

Size of Product.	Capacity per Hour.	Horse Power Required.
¼ inch	1¾ tons	5¼
½ inch	4 tons	4
1 inch	6½ tons	5¼
1½ inch	10 tons	5
2 inch	14 tons	3½

Laboratory Steam Stamp.[2]—The University of Idaho has fitted up an Ingersoll "baby" drill as a steam stamp. The mortar block is four 8x8 in. timbers 46 in. long. The mortar weighs 180 lb. and is ½ in. thick at the top, 1½ in. at the die. The shoe weighs 9 lb. and the die 12 lb. They are 3 in. in diameter. The stem is fitted into the chuck of the drill. The screen opening, 5x9 in., is too small. The stamp does its best work with a stroke of 4 in., 200 per minute, with a boiler pressure of 80 lb. One test showed a capacity of 2570 lb. per 24 hours through a 40-mesh screen. The results have been very satisfactory both in crushing and amalgamating.[3]

Rolls vs. Stamps.[4]—The Boston Consolidated Mill at Bingham, Utah,

[1]*Mining Reporter*, 1906, LIV, p. 370.

[2]Louis Fogle and Reginald Leonard, *Min. and Sci. Press*, 1906, XCIII, p. 319.

[3]The writers are informed that a similar stamp has been developed by Prof. J. B. Porter of McGill University and installed in his mining laboratory.

[4]*Eng. and Min. Journ.*, 1906, LXXXI, p. 763.

is installing Nissen stamps for fine crushing of a soft light colored porphyry containing finely disseminated copper sulphides. Its neighbor, the Utah Copper Company, is to use rolls and Chile mills on the same kind of ore. The Nevada Consolidated Copper Company is erecting an immense mill at Ely, Nev., which employs breakers and Huntington mills in crushing a low grade copper ore. Comparative results of these three installations will be awaited with interest.

Comparative tests of the Nissen stamp with ordinary stamps, rolls and various circular mills were made by the Boston Consolidated Mining Company on quartzite of medium hardness crushed through a 28-mesh screen.[1] The capacity per stamp was 10.75 tons per 24 hours and the slime produced was 10 per cent. less than that of any other competing mill. As a result the company is building a concentrating mill which will contain 360 of these stamps.

Rolls without Springs.[2]—Six pairs of Hadfield crushing rolls, 36x15 in. have been built for the Caucasus Copper Company for fine crushing. These rolls have no springs, as the latest experience in fine crushing indicates that springs not only cause an uneven product but are productive of uneven wear on the shells and sliding bearings.

Chile Mills.[3]—A Chile mill, 6 ft. diameter, is used for crushing and amalgamating at the Golden West mill near Rochford, S. D. It crushes 100 tons in 24 hours but is not satisfactory as an amalgamating machine. It is proposed to install three Chile mills of the Lane type, 10 ft. diameter, the six rollers making 10 r.p.m., to regrind the product of the present mill. Fine grinding is necessary and to do this the mill must have slow speed, high discharge, screens not too close to the wheels, the wheels not too heavy, and the feed about ¼ inch.

Huntington Mills.[4]—This valuable article contains the results of experience with Huntington mills in Southern Rhodesia. The installation from which most of the data were gathered consisted of a grizzly, Dodge breaker, bucket elevator, five 1250-lb. stamps crushing through a 1.5, 2.8, 3 or 4-mesh screen, amalgamated copper plate 9 ft. long and 5 ft. wide, 5-ft. Huntington mill with 32- or 35-mesh screens, and two amalgamated silvered plates (12x5 ft.). The Huntington was of the latest direct overhead driven type, weighed about 12,000 lb. and was set on a combined timber and concrete foundation.

The general criticism is that these mills are troublesome to run and expensive to keep up, and in some respects they are poorly constructed. The rollers, yokes and shafts might be considerably improved as to the method of oiling and also by the adoption of removable bushings which would ob-

[1] *Min. and Sci. Press*, 1906, XCIII, p. 240.
[2] *Min. Journ.*, 1906, LXXX, p. 593.
[3] E. J. Kennedy, *Min. and Sci. Press*, 1906, XCIII, p. 545.
[4] C. E. Parsons, *Trans.* I. M. M., 1906, XV, p. 587.

viate the purchase of an entire new roller head, shaft and yoke when these parts have become worn. The expense for wearing parts is shown in the accompanying table:

WEARING PARTS OF HUNTINGTON MILL.

Part.	Approximate Life.	Weight in Pounds.	Cost of Each at Mine.
Yoke	3 months	97	$34.50
Headshaft	4 months	232	44.00
Cap	5 months	....	6.75
Ball races	4 months	....	17.75
Shell	6 weeks	142	19.30
Die ring	8 weeks	610	17.00
Balls (per dozen)	6 months	....	1.85

The total expenses for the Huntington mill (not including power) were as follows: spare parts, $154.30; white wages, $34.80; native wages, $13.50; screening, $60.00; lubrication, $3.75. All of the above figures will be somewhat reduced in the future.

Careful feeding is necessary since the machine has no means of automatically regulating the feed supply. Underfeeding causes noise and vibration; overfeeding causes packing and stoppage. The latter was remedied by increasing the hight of the stamp discharge temporarily or by hanging up one or two of the stamps for five or 10 minutes.

The roll shells and die rings are carefully centered and fastened in place with wedges of dry hard wood. In the case of the die ring, iron wedges supplement the wood. It is important to keep these parts wearing evenly. The die ring may be ground down to an even surface by substituting a "truer shoe" for one of the rollers. This slides around the inside of the die ring. The shells may be taken out occasionally and turned down in a lathe. Since the ore is fed at the side a hole wears into the die ring on either side just below the feed. Center feeding would avoid this. Shells wear from 1½ in. down to ½ in. in thickness and then they crack and come off. Die rings wear from 2 in. down to ¾ inch.

The four scrapers are best held in place by iron keys. The scrapers should be set about ¼ in. above the bottom; but where inside amalgamation is used, it is best not to lower the scraper until the mill has been cleaned out; otherwise it churns up accumulating amalgam. Mercury was fed into the Huntington, about 2 oz. every hour. Amalgam accumulated inside and was cleaned out at intervals, varying from two to 14 days, depending on the richness of the ore. An inside mercury well worked well.

The screens in the mill were five in number, each 9x30 in. The pulp is dashed against these screens with considerable force when the mill is run at its customary speed of 80 r.p.m. and the screens wear rapidly and

soon break. The wear appears to be more rapid when the mill is run with two rollers than with four. Diagonal slot screens No. 40 wore well but the discharge was considerably reduced so that woven wire screens with heavy wire were preferable. Overfeeding wears the screens badly. The horizontal wires wear faster than the vertical. One experiment was tried of placing the screen diagonally so that the wires were inclined 45 deg. to the horizontal, This screen lasted 30 hours while with the wires in a horizontal position a screen lasted only 20 hours and 50 min. When placed diagonally the long wires wear faster than the cross wires.

Full details are given of capacities, time lost for repairs, amalgam recovered, assays, etc. The following general conclusions are made: The machine has a low first cost but a high cost for wearing parts. It produces an even-graded pulp owing to large discharge area. Ample plate area should be supplied. Best results are obtained if the feed is ¼ in. or less especially when dealing with hard quartz. For a 5-ft. mill the best speed is 80 to 85 r.p.m. The shells and die rings should be kept smooth and circular. A form of central feed would be more satisfactory than the side feed. The mill doubled the duty of a five stamp mill while the water consumption was almost the same as would be required for five stamps alone. Details of Huntington mills in eleven instances are given in the accompanying table.

DETAILS OF HUNTINGTON MILLS.

Items.	1st Case.	2d Case.	3d Case.	4th Case.
No. of Mills	One	Three	One	One
Type	5-ft. underdriven	5-ft. underdriven	5-ft. underdriven	5-ft. underdriven
No. of Stamps	...	...	...	...
Screens, variety	Wire	Wire	Wire	Wire
Screens, cost monthly	Hard ore uses more	$45, 3 discharges	?	$25, 3 discharges
Screens, size or mesh	No. 26. Apertures 0.022. 4 screens	28.3	30	26
No. of revolutions	94	74	68	73
Nature of ore	Very hard	Half quartz, half schist, compact, fairly tough	Hard quartz	Medium quartz
Size fed to Huntington	...	...	1¼ in.	¾ in.
Amount crushed per day	20 tons	25 tons	15 to 24 tons	24 to 30 tons
Horse power required	6	?	5	6
No. of working days per mo.	27	24	20	27
Time mill has run	16 months	13 months	8 months	8 months
Pulp or ordinary ore	Ordinary	Ordinary	Ordinary	Ordinary
Water consumed	1300 gal. per hour	?	1500 gal. per hour	900 gal. per hour*
Cost of repairs	$125 per month (approx)	$135 per month (approx)	More than 10 stamps	?
Cost of spares a ton	25c. per ton	12c. excluding screen	?	21c. to 24c.
Ring dies, life	8 weeks	12 weeks, 1600 tons	8 weeks	12 weeks
Ring dies, wear	Uneven	Uneven	Even?	Uneven especially below feed
Shells, life	6 weeks	6 weeks	6 weeks	8 weeks
Shells, wear	Uneven	Uneven	Even?	Even
Foundations	Logs, stone, cement	Timber	Stone and cement	Timber
Amalgamation	Inside filled mercury well	...	...	Inside
Amalgamation, extraction	75%	60%	70%	80 to 90% of total from inside
Mercury required	...	...	...	...

Items.	5th Case.	6th Case.	7th Case.	8th Case.
No. of Mills	One	One	Two	One
Type	5-ft. underdriven	5-ft. underdriven	5-ft. underdriven	5-ft. underdriven
No. of stamps	....	....	Roller mill	5
Screens, variety	Wire	Wire	Wire	Wire
Screens, cost monthly	$37.50	$65.90, 5 discharges	....	$50, 5 discharges
Screens, size or mesh	28.3	40	35 to 20	3 battery, 30 Huntington
No. of revolutions	80	75–80	75	82
Nature of ore	Hard quartz........	Quartz very hard and rubble	Hard white quartz	Hard quartz, some schist
Size fed to Huntington	$\frac{3}{4}$ in.	$\frac{3}{4}$ to 1 in.	$\frac{3}{8}$ in.	$\frac{1}{3}$ in.
Amount crushed per day	14 tons hard 18 tons medium	25 quartz 40 rubble	....	4.6 tons
Horse power required	16 actual........	10	....	....
No. of working days per mo.	26	25–26	....	23
Time mill has run	8 months	13 months	....	7 months
Pulp or ordinary ore	Ordinary	Ordinary	Crushed first in rolls	Pulp
Water consumed	1300	?	9 tons water to 1 ton ore	600 gal.*
Cost of repairs	$200 to $225 hard quartz, $150 to $175 medium	$200 to $250	....	$135 per month
Cost of spares a ton	62c. hard, 37c. to 50c. medium	....	....	16c. per ton
Ring dies, life	6 weeks	6 weeks	Uneven	13 weeks
Ring dies, wear	Uneven below feed	Uneven below feed	....	Uneven in undulations, below feed
Shells, life	6 weeks	4 weeks	....	4 weeks
Shells, wear	Even	Sometimes uneven	Slightly uneven	Uneven
Foundations	Concrete	Wood and concrete	Timber	Concrete
Amalgamation	....	Inside mercury well	Inside and outside	Inside
Amalgamation, extraction	80%	4 to 90% of total inside	45%	30%
Mercury required	....	....	....	....

Items.	9th Case.	10th Case.	11th Case.
No. of Mills	Four	Two	One
Type	6-ft. overhead	5-ft. underdriven	5-ft. overhead
No. of stamps	40	10	5
Screens, variety	Wire	Wire	Wire
Screens, cost monthly	....	$43.85	$60–5 discharges
Screens, size or mesh	4 battery, 25 Huntington	$\frac{3}{32}$ in. battery, $\frac{1}{32}$ in. Huntington	Battery 2.8 and 1.5 Huntington 35
No. of revolutions	64	72	85
Nature of ore	Soft, fines	Very hard indeed	Hard quartz and steatite
Size fed to Huntington	$\frac{1}{5}$ in.	$\frac{1}{11}$ in.	$\frac{1}{5}$ in.
Amount crushed per day	11.5 tons	5.74 tons	8.3 to 10.2 tons
Horse power required	8	6	Probably 12 to 15
No. of working days per month	....	$18\frac{1}{2}$	24
Time mill has run	Just commenced	11 months	5 months
Pulp or ordinary ore	Pulp	Pulp, slimes removed	Pulp
Water consumed	....	2400 gal. per hour	6 tons water to 1 ton ore
Cost of repairs	....	$369.50	$266.35
Cost of spares a ton	....	59c.	....
Ring dies, life	....	10 weeks	8 weeks, 2000 tons
Ring dies, wear	....	Uneven	Uneven below feed
Shells, life	....	2.8 weeks	5 weeks and less
Shells, wear	....	Uneven, come off frequently	Uneven
Foundations	Concrete	Concrete and timber	Concrete and timber
Amalgamation	Not inside	Inside	Both
Amalgamation, extraction	....	70%	80%
Mercury required	....	....	3.86 dwt. per ton

*Return water used.

Ball Mills.[1]—The Kominor ball mill and the Dana tube mill are described. These have found extensive use in grinding cement clinker and iron ores. The ball mill acts as a preliminary crusher for the tube

[1] *Thonind. Ztg.*, 1906, XXX, p. 685.

mill. It differs from the ordinary ball mill in that the feed coming in at one end passes through the mill and out the other end upon a conical screen surrounding the mill. The oversize of this screen is delivered back into the mill with the feed.

Ball Mills.[1]—In recrushing tin ore a No. 2 Krupp ball mill run wet failed because the feed was too fine. Too much material accumulated on the screens and packed them. The mill had 25-mesh wire screens and a sizing test of the product showed the following: through 20 on 30 mesh, 13.75 per cent.; through 30 on 40 mesh, 22.5 per cent.; through 40 on 50 mesh, 15 per cent.; through 50 on 60 mesh, 11.25 per cent.; through 60 on 80 mesh, 8.0 per cent.; through 80 mesh, 30.5 per cent.; total, 101.0 per cent.

Ball mills in Kalgoorlie, Australia, are ordinarily fed with the products of a Gates breaker. In one instance excellent results were obtained in feeding material which contained 3 per cent. coarser than 6 in. and 75 per cent. coarser than 2 in. A mill crushed 33 tons of calcareous and silicious material through 20 mesh in 24 hours. Screen analyses of the product gave: On 40 mesh, 24.1 per cent; through 40 on 60 mesh, 9.4 per cent.; through 60 on 80 mesh, 6.2 per cent.; through 80 on 100 mesh, 6.6 per cent.; through 100 on 150 mesh, 3.2 per cent.; through 150 on 200 mesh, 4.2 per cent.; through 200 mesh, 46.3 per cent. The total weight of the balls in the mill varied from 2000 to 2200 ℔. and one ball weighing 18 ℔. was added every 24 hours. The lining lasts seven or eight months and costs $900. Each mill makes 25 r.p.m. and requires 24 i.h.p. The key to success seems to be in the use of forged steel balls. Chilled iron, cast steel and manganese steel balls do not wear evenly and reduce the efficiency of the mill.[2]

Ball Mills and Tube Mills.[3]—The Gröndal and Pape-Henneberg wet crushing ball mill, without a sieve, is made in several sizes. The larger mills are 1775 mm. diameter inside and 1050 mm. long. The material is fed continuously through the center axis at one end and the discharge is through large openings around the axis at the other end. Hard steel linings are used. The lining staves are 85 mm. thick and they may be readily replaced by removing a section of the outer cylindrical housing of the mill. The capacity and fineness of the product can be regulated by varying the quantity of material and water fed. Recently classifiers were installed after these mills to take out the coarse material and return it to the mill. The advantage of this form of ball mill is that there is no screen to clog. G. Gröndal has tested these ball mills with stamps on iron ores at Pitkaranta, Sweden, and found that the ball mills had greater capacity and less loss.

In cement works the wet ball mill of the dimensions previously given

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[1]A. H. Bromley, *Trans.* A. I. M. E., 1905, XXXVI, p. 227.
[2]*Mining Reporter*, 1906, LIV, p. 599.
[3]C. Blömeke, *Metallurgie*, 1906, III, p. 408.

crushes 80 tons of hard limestone and marl in 24 hours with a loss of 27.8 grams per ton for the balls and 30.47 grams for the linings.

In 14 mills in Scandinavia there are 48 Lohnert wet ball mills in operation on iron ore mixed with serpentine and they crush 30 to 50 tons in 24 hours according to the hardness of the material. A wet ball mill 1775 mm. diameter and 1050 mm. long, inside dimensions, uses about 25 h.p. and has a capacity of 2.5 to 4.5 metric tons per hour.

In a plant in South Germany a capacity of 200 tons in 24 hours is obtained by a coarse crusher run dry followed by three wet ball mills with 1500 to 1700 kg. of balls, and a No. 16 Lohnert tube mill 1600 mm. diameter and 6000 mm. long. The product of these ball mills yielded the following sizes: On 1.27 mm., 25 per cent.; through 1.27 on 0.5 mm., 30 per cent.; through 0.5 on 0.36 mm., 10.2 per cent.; through 0.36, on 0.25 mm., 5.1 per cent.; through 0.25 on 0.17 mm., 6.1 per cent.; through 0.17 on 0.13 mm., 7.3 per cent.; through 0.13 mm., 15.5 per cent. The wear of a wet ball mill crushing 75 tons in 24 hours was 0.0287 kg. of balls per ton of feed, 0.03047 kg. of lining, or 0.0583 kg. total.

The dry ball mill without sieves, as made by Pfeiffer Brothers, is cylindrical and has the feed (up to 150 mm. size) introduced through a hollow trunnion. The ends of the cylinder are special chilled castings; the circumference is of crucible steel and is divided into several segments so arranged as to make a series of steps all around the circles. Between the segments are slots for the discharge of the crushed material which is collected in a housing and delivered to an elevator. These ball mills are made in eight sizes having a total weight of balls of from 200 to 5000 kg. and requiring from 3 to 70 h.p.

The product of the ball mill is elevated to a Pfeiffer dry centrifugal separator. This resembles very much the Mumford & Moodie separator (see "Ore Dressing," Richards, p. 818) except that there are two revolving plates, a smaller one above a larger one. The material is thus thrown outward twice by centrifugal force through a rising current of air which removes the fine dust to a collecting chamber and leaves the coarser particles to fall down by gravity into the ball mill to be recrushed. These separators are used in cement grinding, etc., and have the sizes shown in the accompanying table:

SIZES OF PFEIFFER SEPARATORS.

Diameter in mm......	1,100	1,500	1,800	2,100
Hight in mm.........	2,100	2,700	3,100	3,400
Revolutions per min.	300	250	210	200
Horse power.........	$\frac{3}{4}$	1	$1\frac{1}{2}$	2
Capacity, Kg. per hour	400 to 800	1000 to 2000	2,000 to 3,000	4,000 to 6,000

The fineness of the separation can be regulated down to stuff which will pass a sieve with 5000 meshes per square centimeter. This separator has

been used in magnetic separation to replace fine sieves for removing dust and fines.

The Barthelmuss dry crushing mill resembles the Huntington mill in principle but it has only two revolving mullers. A fan attached to the central axis creates a current of air and carries the crushed material through the surrounding sieve.

Tube Mills.[1]—The Standard mill at Bodie, California, has a 5x22 ft. tube mill for regrinding sands. The mill makes 26 r.p.m. and is charged with about 12 tons of Greenland pebbles. Originally white cast iron liners were supplied but they gave trouble from dropping and had short life. Soft wood blocks on end, 6 in. long had a life of 10 days. Mountain mahogany had about the same life. Silex linings lasted well but gave trouble when partly worn out from the replacing of worn out blocks, and there was too much delay from the putting in of new sets of blocks. Finally wrought iron plates $\frac{7}{8}$x8 in. were adopted. These were cut into 7 and 15 ft. lengths and bolted through the shell. These lasted about 100 days grinding 4800 tons of sand. The lining consumed per ton of ore was 2.44 lb. and the pebble consumption 6.47 lb. The wrought iron plates wear very thin and are quickly replaced.

An article on tube mills at El Oro[2] contains very full results on the experience with tube mills at El Oro, Mexico. Valuable information is given on the following points, viz., foundation, rotation, leaks, feeding, arrangement of platform, liners.

In tube mill work a very important point is a good classification to remove the fine material from the feed.[3] Thick pulp makes for better grinding and for less wear on the pebbles and liners. Manganese steel liners are not so satisfactory as silex. Comparative data are as follows, the figures for manganese steel being first stated with the figures for silex in brackets immediately afterward: feed of sand in tons per 24 hours, 232 (229); ratio of water to one part of ore in feed, 1.16 (1.00); on 60 mesh before grinding, 43.88 per cent. (43.52); on 60 mesh after grinding, 13.97 per cent. (6.16); through 60 on 90 mesh before grinding, 29.48 per cent. (30.36); through 60 on 90 mesh after grinding, 29.85 per cent. (19.66); through 90 mesh before grinding, 26.64 per cent. (26.12); through 90 mesh after grinding, 56.18 per cent. (74.18); tons reduced from over 60 to under 60 in 24 hours, 69 (85); tons increase in under 90 in 24 hours, 68 (109). Steel costs several times as much per pound as silex. Two thicknesses of silex blocks have been used, viz., 4-in. and $2\frac{1}{2}$-in. With the $2\frac{1}{2}$-in. blocks the cost of liner installed ready for use is \$550; the days run are 52.56; the cost per day is \$10.45. With the 4-in. blocks, the figures are respectively \$760—\$88.75—\$8.60.

[1]R. G. Brown, *Min. and Sci. Press*, 1906, XCIII, p. 261.
[2]Charles Butters, *Min. and Sci. Press*, 1906, XCII, p. 344.
[3]W R. Dowling, *Journ. Chem. Met. and Min. Soc. of South Africa*, 1906, VI, p. 308.

The discharge half of the mill wears longer than the feed half. The end liners, funnel-shaped castings and discharge screens last three to four months. The running time of the mills averages 88.98 per cent. of the total time; 6.58 per cent. of the stoppages are due to relining and the balance to sundry causes. French and Danish flint pebbles are used, supplemented by 3- to 6-in. pieces of hard banket ore. A full load of ore in the mill gives best results and a proper speed of the mill is important. Full details of the method of lining the mill are given.

Tube Mill Lining.[1]—S. D. McNicken describes a tube mill lining made of hard iron segments placed around the inside of the mill and held in place by longitudinal bars 2½in. square which are bolted through to the outside. These bars serve the further purpose of holding the flint stones so that wear on the lining is much reduced. The segments are 4 ft. long and 1½ in. thick and there are 12 of them around the 4-ft. circle. At the Komato Reefs mine, New Zealand, such a lining lasted 18 months while the smooth lining without the bars lasted only six to eight months. The mill treats 70 tons per day of material from stamps having 10-mesh screen and grinds it to equal the product of stamps having a 35-mesh screen. The cost of lining is 1.4c. per ton of sand ground.

At the Waihi mine in New Zealand tube mills for regrinding the product from stamps were formerly lined with silex from Iceland.[2] It cost $400 to line one mill and it lasted three months. Now Mr. Barry, the superintendent, uses segments of his own design. Each segment consists of a rectangular iron frame filled with hard Waihi ore and cement all curved to suit the inside of the mill. Such a lining lasts six months and costs $200.

A. E. Drucker reports that a tube mill lining made up of hard quartz laid in cement was unsatisfactory and was replaced by cast steel liners.[3]

Pebbles for Tube Mills.[4]—The flint pebbles for tube mills are derived from beds of Cretaceous chalk on the sea coast of Greenland and France. In the United States pebbles are found in Texas, California, Florida, Kansas and Colorado. The silex or flint blocks for lining come principally from Sweden and Belgium. The amount of pebbles used in a tube mill is usually five or six tons and a size of from 2 to 3½ in. is preferred. The Danish (Greenland) pebbles were quoted in 1905 at $16 to $20 per long ton f.o.b. New York. French pebbles brought $9.50 to $12 per long ton under the same conditions.

Tube Mills for Diamond Washing.[5]—An experimental tube mill test has been made on the blue ground of the Premier diamond mine, South Africa. The mill was 22 ft. long, 5½ ft. diameter and contained three tons of pebbles. When supplied with a slotted screen with holes 1¼ in. wide and 3 or 4 in. long

[1] *Min. and Sci. Press*, 1906, XCIII, p. 538.
[2] *Min. and Sci. Press*, 1906, XCIII p. 108.
[3] *Min. and Sci. Press*, 1906, XCIII, p. 594.
[4] C. C. Schnatterbeck, "Mineral Resources of the United States," 1905, p. 1073.
[5] *Min. Journ.*, 1906, LXXX, p. 184.

it crushed 23 loads per hour. The results appear to have been promising although they have not been made public. Further experiments are planned.

Fine Crushing Machines.[1]—Primarily these articles are detailed descriptions of the crushing, amalgamation and cyanide plants of the Kalgoorlie district but they are of interest to the ore dresser on account of the details of crushing machinery.

The Sons of Gwalia mill uses 1000-℔. stamps and regrinds coarse sand in Wheeler pans. Each pan requires 5⅓ h.p. and treats 33 tons per day running at 45 r.p. m. Each pan uses 33.3 gal. of water per minute. Treating chloritic schist, a new set of shoes and dies, costing about $190, lasts six weeks. The following sizing tests show the quality of the work, the feed being stated first, and the product in brackets: On 40 mesh 50.5 per cent. (12.5); through 40 on 60 mesh, 30.0 per cent. (27.5); through 60 on 100 mesh, 17.5 per cent. (24.0); through 100 on 150 mesh, 1.5 per cent. (7.5); through 150 mesh, 0.5 per cent. (28.5).

The Kalgoorlie Gold Mines mill first reduces the ore in No. 5 Krupp ball mills and regrinds in Wheeler pans. The feed to the ball mills has been broken to 1½ in. and a ball mill crushes 40 tons of this schistose material per day to pass through a 32-mesh screen. The product contains 7.96 per cent. on 40 mesh, 10.92 per cent. on 60 mesh, 21.67 per cent. on 100 mesh, 6.65 per cent. on 120 mesh, 5.0 per cent. on 150 mesh and 47.8 per cent. through 150 mesh.

The Great Boulder Perseverance mill employs Griffin mills run dry for the first grinding and has grinding pans for the reduction of the roasted coarse sands. The No. 11 Griffin mills have die rings 30 in. diameter and mullers 15 in. diameter. Treating schistose material through 1¼-in. ring and fitted with 14-mesh (22 I. S. W. G.) discharge screens, each mill treats 37 tons per day using 18 h.p. The vertical shaft makes 198 r.p.m. Unexploded charges of dynamite in the ore explode in the mill and cause fractures in the pan. Die rings last 18 days; roll tires, nine days; ploughs, nine days; roll bodies, six months; shafts, six months; bodynuts, 18 days; follower nuts, three months; ball and body trunnions, three months; thrust rings, 18 days; feeder shaft, six months; cast iron pans, a few explosions at most; cast steel pans, 12 to 20 explosions; balls for bearings, in good shape after six months; ball rings of cast steel, four months.

The grinding pans in the Great Boulder Perseverance mill are 8 ft. diameter and 3 ft. 4 in. deep. A set of 12 shoes, each weighing 220 ℔. lasts from three to six months, while a set of 12 dies, each weighing 283 ℔. lasts 12 months. The abrasion of metal amounts to 13.23 oz. per ton of sands ground. Running at 27 r.p.m. 118.4 oz. are abraded per square foot of shoe and die surface per month while at 31 r.p.m. the abrasion is

[1]Robert Allen, *Min. Journ.*, 1906, LXXIX, pp. 213, 313, 344, 380, 409, 615, 686, 853. From *Monthly Journal* of Chamber of Mines of West Australia.

163.8 oz. On the assumption that abrasion is proportional to grinding effect the pans at 31 r.p.m. do 38.3 per cent. more work than at 27 r.p.m., but the power required is increased from 6 to 9 h.p. The pans treat 1.4 tons of sand per horse power per day or one ton per square foot of grinding surface per 24 hours.

SIZING TESTS AT GREAT BOULDER PERSEVERANCE MILL.

	14 Mesh Screen No. 22 Wire.	10 Mesh Screen No. 21 Wire.	10 Mesh Screen No. 20 Wire.
On 20 mesh..	0.30 per cent.	3.50 per cent.	2.50 per cent.
Through 20 on 30 mesh..	1.00	5.15	4.00
Through 30 on 40 mesh..	2.20	4.75	4.50
Through 40 on 60 mesh..	5.50	8.75	7.15
Through 60 on 80 mesh..	5.35	5.25	5.35
Through 80 on 100 mesh..	4.15	4.00	4.85
Through 100 on 120 mesh..	2.00	2.65	2.25
Through 120 on 150 mesh..	7.35	5.00	6.00
Through 150 on 200 mesh..	4.05	2.15	1.65
Through 200 mesh.........	68.10	58.80	61.75

The Great Boulder Proprietary mill has No. 8 Krupp ball mills and 30-in. Griffin mills (both run dry) for the first crushing and modified Wheeler pans for regrinding. The ball mills have drums 8 ft. 10 in. diameter and 4 ft. wide. Each mill has a load of 38 cwt. of balls, is driven at 23 r.p.m. and has a capacity upon the hard schistose ore of 90 tons per 24 hours of material which has been through a 1⅛-in. ring and is crushed through 30-mesh screen having No. 28 wires (I. S.W.G.). The plates, end liners and balls are of Krupp cast steel and the consumption of metal in balls is 0.75 lb. per ton crushed. A set of solid grinding plates crushed 10,760 tons before they were discarded.

SIZING TESTS AT GREAT BOULDER PROPRIETARY MILL.

Mesh.	Griffin Mill.	Ball Mill.	Mesh.	Griffin Mill.	Ball Mill.	Mesh.	Griffin Mill.	Ball Mill.
On 30 mesh....	1.07	0.22	60—80 mesh	4.60	6.64	120—150 mesh	10.60	6.32
30—40 mesh...	1.01	5.21	80—100	5.99	8.07	150—200	4.38	5.80
40—60 mesh...	3.36	15.74	100—120	1.27	1.67	Through 200 mesh.......	67.72	50.33

The Griffin mills are 30 in. in diameter and are driven at 200 r.p.m. Each requires 20 h.p. and has a capacity of 30.8 tons per day when crushing through a 15-mesh screen (No. 21 wires I. S. W. G.). The wearing parts last as follows, the life being stated in terms of tons crushed, with the cost of the parts at mine given in brackets: Tires, 9-10 ($19.25); dies, 27-30 ($28.00); body rolls (made at mine) 90 ($10.00); followers, 9-10 ($8.00); shafts, rough made at mine, broken ones welded many times, 90 ($11.25); shafts, machined, 90 ($20.00); ball and trunnions, 100 ($32.25); thrust rings, brass, 90 ($4.40); thrust rings, steel, 180 ($—); feeder shaft, 160 ($2.50); cast iron pans, 90 ($28.00).

Principles of Crushing.[1]—In discussing the principles of rock crushing, O. H. Howarth points out that crushing a lump of ore by direct pressure first causes a grinding of the points of contact to a fine powder followed by a final rupture of the lump. In most crushing appliances, however, there is some slip which causes a grinding action and increases the amount of fine powder. All crushers make use of vertical pressure or horizontal rolling, or a combination of these in varying proportions. A careful study of the character of the rock to be crushed should be made to determine the proper machine to use in each individual case.

In connection with cement grinding, F. H. Lewis discusses[2] the whole crushing problem. The most efficient crushing process is the one which removes the crushed particle from the field of action at the moment it is ground to the desired size; in other words it is a combined crushing and separating process. If the crushed particles are not so removed they remain to hinder the work of the machine in its action upon the remaining particles which need further crushing. We have familiar examples of this combination of crushing and separating in the plants for reduction of ores, cements, etc., where material is fed to a crushing machine, such as rolls, ball mill, tube mill, etc., and afterward is treated by some form of screen or pneumatic separator which takes out the fines and returns the coarse for further crushing. In such a case it is economy to have the machine crush somewhat coarser than the size of the desired product. In many cases the functions of crushing and separating are combined in one machine. Since the installation of separating machines is somewhat expensive involving the use of conveyors and elevators it follows that it is not always the greatest economy to adhere strictly to the preceding principles. In other words the most efficient plant may not always be the most economical one.

Standard for Crushing.[3]—S. H. Pearce and W. A. Caldecott suggest that the proper standard of comparison of efficiency of crushing in tube mills is by computation of surface area of particles before and after crushing as explained in Richards' "Ore Dressing," p. 305. Some such calculations are made in this paper.

Sizing and Classifying.

Impact Screen.[4]—A Wilfley impact screen has been installed at the Wheal Kitty for treating the tailings from Wilfley tables. The screen is 3 ft. square, covered with 60-mesh brass wire cloth, wires 26 gage, sloping 45 deg. and receiving 300 blows per minute through a cam and spring mechanism. The oversize amounts to about one ton per hour and assays

[1]*Mines and Minerals*, 1906, XXVI, p. 441.
[2]*Eng. Mag.*, 1906, XXXII, p. 210.
[3]*Journ. Chem. Met. and Min. Soc. of South Africa*, 1906, VII, p. 72.
[4]J. H. Collins, *Trans.* I. M M., 1906, XV, p. 524.

$1\frac{1}{2}$ to 2 ℔. of tin oxide per ton; the undersize amounts to about 0.5 ton per hour and contains 6 to 12 ℔. of tin oxide per ton. After 300 hours' work the screen shows little wear and has never choked.

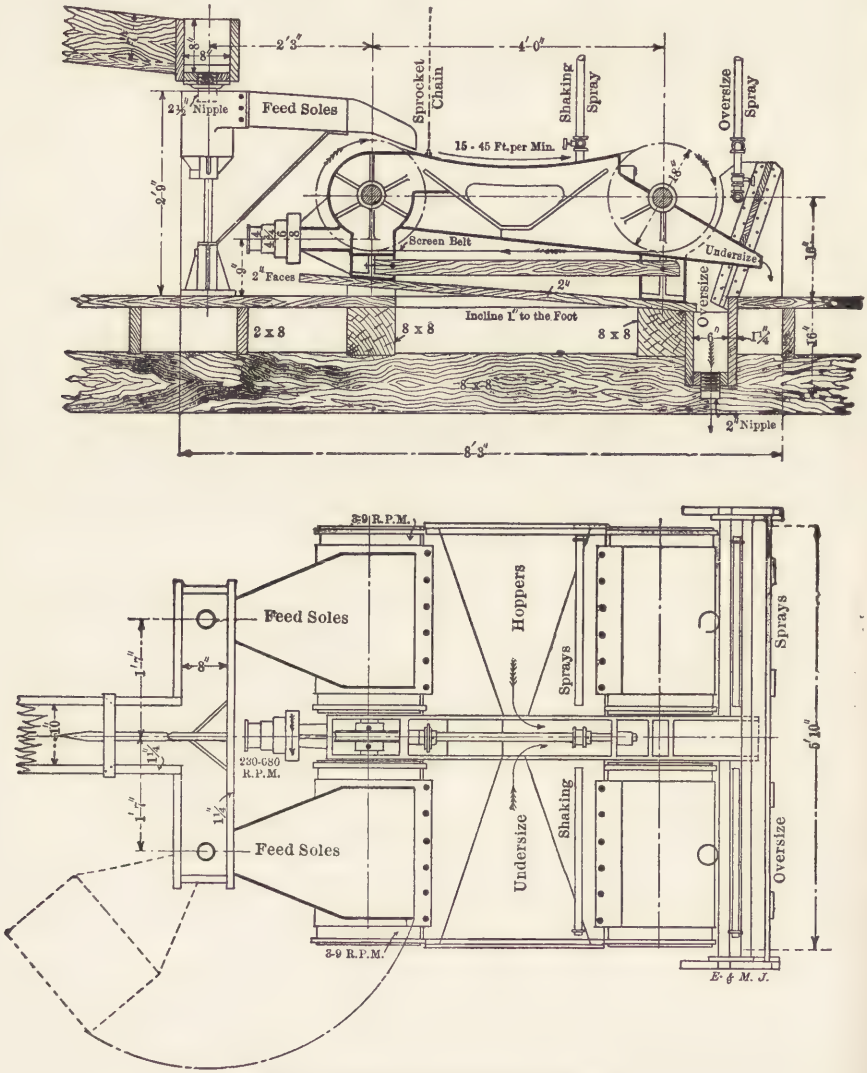

FIG. 1.—CALLOW TRAVELING-BELT SCREEN.

Traveling-belt Screen.[1]—With ordinary trommels the minimum limit of wet screening in concentration mills is about $2\frac{1}{2}$ mm. The Callow screen reduces this limit to 200 mesh or even further. As shown in Fig. 1,

[1] J. M. Callow, *Eng. and Min. Journ.*, 1906, LXXXI, p. 468.

this consists of an endless screen running on two end rollers at a speed of from 15 to 45 ft. per minute. Two such screens are usually mounted side by side. The feed is distributed over feed soles which deliver the coarse particles ahead of the fine. The movement of the belt brings the material under a shaking spray of clear water which washes the undersize through the screen into the side discharging hoppers below. The oversize passes over the end roll and is washed off by a spray into the oversize hopper.

The screens are attached through rubber connectors to sprocket driving chains. The rollers have no outboard bearings, which allows the screens to be changed readily. The capacities per 24 hours for each foot of width are as follows: 50 to 60 tons of feed for 30-mesh screen; 40 to 50 for 40 mesh; 30 to 40 for 60 mesh; 25 to 30 for 80 mesh. These figures are based on a speed of 20 ft. per minute and an assumption that the feed will be split into half oversize and half undersize. The feed water per foot of width may be as low as 4:1, or as high as 250 gal. per minute with 30-mesh screens or 30 gal. per minute with 80-mesh screens. The shaking spray consumes 3 to 5 gal. per minute per foot of width and the oversize jet about half that amount. The wear of the screen amounts to not over 2c. per ton even on the finest sizes. These machines are working successfully at the mill of the Basin Reduction Works in Montana and at the Bunker Hill & Sullivan mill in Idaho.

Bronze Screens.[1]—At Lautenthal, bronze wire screens with from 3 to 1 mm. holes have replaced copper plate. They have longer life and do better screening.

The Fitzgerald Dewatering Screen.[2]—This is used at the Humphreys mill, Creede, Colo. for handling all the jig tailings of a mill treating 275 tons in 24 hours. Two are used, each consisting of a wooden launder 24 ft. long, 20 in. wide and 5 in. deep. The last 16 ft. of each has 25-mesh perforations in $\frac{1}{4}$ in. steel plate and the water falls through to a launder below while the oversize is delivered over the end. The screen has an inclination of about $\frac{3}{4}$ in. per foot and is connected at the head end to a Wilfley table mechanism which gives it a bump of 4 to 6 in., 50 times per minute thus progressing the coarse material forward over the discharge end into Chile mills.

Screens for Sizing.—Further contributions on this subject have appeared during the year. As expressed in last year's review, we are of the opinion that screens should be designated by their actual size of opening in inches or millimeters, preferably the latter.

E. A. Hersam[3] has computed five tables showing the actual size of the holes in screens with varying number of meshes per linear inch and different sizes of wire for each mesh number. In detail the tables are as fol-

[1] *Zeit. f. B. H. u. S.;im Preuss.*, 1906, LIV, p. 278.
[2] H. J. Baron, *Mining Reporter*, 1906, LIV, p. 306.
[3] Bulletin No. 9, A. I. M. E., May, 1906, p. 423.

lows: (1) Sizes of wire screens (usually iron or steel) from 3-in. mesh down to 100 mesh having wires of the Worcester (Washburn & Moen) (Roebling) standard gage. (2) Sizes of wire screens (usually brass) from 3-in. mesh down to 200 mesh having wires of the Old English or London gage. (3) Sizes of wire screens (mostly of brass in the coarse sizes) from 3-in. mesh down to 100 mesh having wires of the Birmingham (or Stubs) gage. (4) Sizes of wire screens (usually copper) from 3-in. mesh down to 100 mesh having wires of the American (or Brown & Sharpe) gage. (5) Sizes of wire screens from 3-in. mesh down to 100 mesh having wires of the Trenton standard gage. It is possible, by referring to these tables, to obtain a screen which shall very closely approximate almost any size of opening. A set of reference letters indicates in each table those screens which are nearest to certain standards such as Rittinger's standard millimeter sieve scale and others.

The use of designations which shall give exact sizes of openings in millimeters or inches is advocated for many reasons. The common method of designating a screen by the number of meshes per linear inch is objectionable because a computation is necessary to determine the size of the opening.

The character of the product of a given sieve is affected by the following conditions: (1) the fracture or cleavage of the ore, which causes a variation in the ratio of the mass to the maximum dimension of the grain. (2) The percentage of moisture which retards screening, especially in fine sizes. (3) The shape of the hole, whether round, square or slotted. (4) The thickness of the metal in punched screens and the diameter of the wire in wire screens.

For measuring fine screens a quick and accurate method is to project an image of the screen in question upon the wall by a stereopticon. Parallel rays of light are passed through the screen and then through a magnifying lens.

Geo. T. Holloway[1] considers the disadvantages of the use of "mesh"-designation. Tables of wire gages are given; also tables showing the actual sizes of holes corresponding to the trade numbers of needle slot screens, and the meshes and sizes of wire in wire cloth screens which correspond to round or square hole punched screens.

The Committee on Standardization of Screens[2] reports that the method of designating screens by the number of meshes per linear inch or per square inch is unsatisfactory for the following reasons: (1) Meshes per linear inch and per square inch are used indiscriminately, thus leading to confusion. (2) No indication of the exact size of the hole is given by this method. (3) The number of apertures per linear inch varies from

[1] *Trans.* I. M. M., 1906, Bulletin No. 5.
[2] *Journ. Chem. Met. and Min. Soc. of South Africa*, Supplement, June, 1906.

the quoted number. (4) The same nominal description is applied to more than one kind of screening. (5) The same sample of screening may have wires of more than one gage. It seems desirable to denote screens by actual size of aperture. Tables are given showing different meshes and sizes of wire for the following apertures in inches: 0.250, 0.200, 0.175, 0.150, 0.125, 0.100, 0.090, 0.080, 0.070, 0.060, 0.050, 0.045, 0.040, 0.036, 0.033, 0.030, 0.029, 0.028, 0.027, 0.026, 0.025, 0.024, 0.023, 0.022, 0.021, 0.020, 0.019, 0.018, 0.017, 0.016, 0.015, 0.014, 0.013, 0.012, 0.011, 0.010.

In commenting on the report of the South African Committee, Philip Argall[1] shows that for commercial work it is not necessary to take irregular

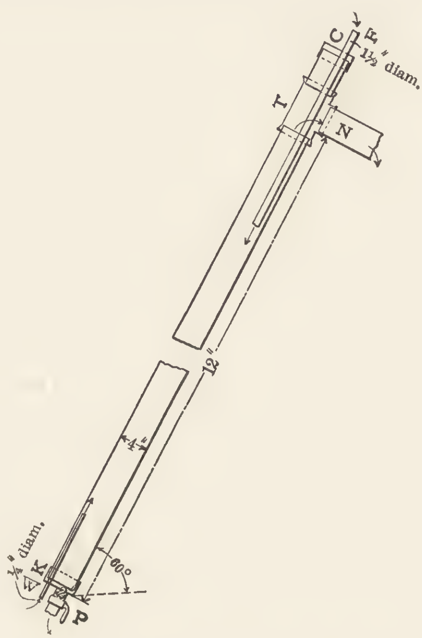

FIG. 2.—DE KALB'S SAND AND SLIME SEPARATOR.

number of apertures per inch in order to obtain a standard series of size apertures.

The Weller Pulp Thickener.[2]—This device used at the Jackson mill, Idaho Springs, Colo., consists of a cylinder 46 in. long and 30 in diameter with sides and ends covered by muslin (about 80 mesh). This cylinder is mounted on a hollow perforated shaft and revolves in a tank of water having a spigot in the bottom. Tailings from the fine jigs are fed into this tank. The excess water is strained by passing inward through the muslin and finding its exit through the hollow shaft. It goes to a Wilfley

[1]*Min. and Sci. Press*, 1906, XCIII, p. 654.
[2]*Min. Rep.*, 1906, LIII, p. 608.

slimer. The thickened pulp is discharged through the spigot and treated on Wilfley tables.

DeKalb Classifier.[1]—A device for separating sand and slime is shown in Fig. 2 in which *P* is the discharge spigot, *K* is the hydraulic water pipe, *F* is the feed pipe and *N* is the overflow. The capacity is not over four tons per 24 hours. Some idea of its work may be gained from the following sizing tests:

RESULTS OF DEKALB CLASSIFIER.

Size.	Feed.	Spigot.	Overflow.	Size.	Feed.	Spigot.	Overflow.
	Per Cent.	Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.
On 0.15 mm..........	4.77	15.03	0.47	0.076—0.051 mm....	1.32	2.05	1.49
0.15—0.13 mm.......	1.67	7.34	0.93	0.051—0.025 mm....	11.84	8.33	10.73
0.13—0.10 mm.......	5.21	5.55	2.24	Through 0.025 mm...	74.83	59.77	83.02
0.10—0.076 mm......	0.38	1.93	1.12				

Settling Box for Recovering Water.[2]—At the American mill, Goldfield, Nev., the overflow from the classifiers and the leaching vats amounts to about 50 gal. per minute and contains about 1.5 per cent. solid matter. To recover the water this material is pumped to a V-shaped box 96 ft. long 6½ ft. wide at the top and 7 ft. deep, tapering to an edge at the bottom. The box is made of two layers of tongue and groove lumber 3 in. wide and 1 in. thick with a layer of canvas between. This is supported by 4x6-in. timbers at intervals of 3½ ft. which are held from spreading by ¾-in. iron bolts at the top and bottom. Every other one is reinforced by a brace of 4x6-in. timber notched into the cross piece upon which the box rests. Originally the box was only 48 ft. long and had five baffle plates coming up to within 3 in. of the surface of the water, but this did not give sufficient settling capacity so the length was doubled and the baffle plates discarded. The settled slime is drawn off through several 2-in. cocks along the bottom. It contains 35 to 50 per cent. solid matter and goes back to the cyanide plant.

The Callow Tank[3] is a conical settling tank with vertical downward central feed, with overflow around the rim at the top, with an annular launder to collect and convey away the overflow and with a spigot in the form of a goose neck to discharge the settlings at a point 12 to 18 inches below the water level in the cone. The central feed pipe is 12 in. in diameter and dips into the water 12 in. The feed is admitted and the currents broken up by a feed box, a cone and a disc float. A tank with 60 deg. sides and a top diameter of 6 ft. is found to adapt itself best to mill work. These tanks are finding extensive application. The following results obtained in working Butte slimes will serve as an example of their

[1]*Eng. and Min. Journ.*, LXXXII, 1906, p. 206.
[2]E. J. Sweetland, *Eng. and Min. Journ.*, 1906, LXXXII, p. 348.
[3]J. M. Callow, private communication.

work. Fifty-seven tanks each supplied with 31.4 gal. of pulp per minute yielded 5.2 gal. of spigot product per minute.

TEST OF CALLOW TANK.

	Total Gallons per Minute.	Grams of Solid per Gallon.	Tons per 24 Hours.	Assay.		Total Contents in 24 Hours.	
				Per Cent. Copper.	Ounces Silver.	Pounds Copper.	Ounces Silver.
Feed	1792.7	41.15	117.16	2.8	2.81	6,559	329
Overflow	1495.0	16.25	38.45	1.815	2.36	1,394	90.8
Spigot Product	297.5	154.5	73.13	3.5	3.34	5,106	244.3

These tanks recovered 62.5 per cent. of solids reduced to 16.6 per cent. of the volume and carrying 77.75 per cent. of the copper and 74.25 per cent. of the silver. The spigot product, treated on Wilfley tables, yields 80,000 lb. of copper and 3500 oz. of silver per month, all of which formerly went to waste.

Removal of Wood in Ore Dressing.[1]—At the Butte Reduction Works, pieces of wood often elude the vigilance of the feeder man and find their way into the breakers and choke them either partially or completely. To obviate this trouble the ore falls into a hopper full of water. The ore sinks and is taken out by an elevator with perforated buckets. The wood floats off one side with the overflow water and falls down over a perforated plate through which the water passes. From 20,000 tons of ore 40 tons of wood have been removed, and it is estimated that the capacity of the crushing plant is increased 15 per cent.

Another difficulty, from talcose ore packing in the breakers to such an extent that the pitman shaft was bent, was remedied by removing fine material from the feed by a grizzly placed in front of the breaker.

Principles of Classification and Jigging.[2]—This important paper contains the results of a large amount of experimental work with quartz, galena magnetite and blende upon the behavior of these minerals in jigs and classifiers.

Concentrating Machines.

Hancock Jigs.[3]—Hancock jigs are being installed at the Mine la Motte mill and at the St. Joseph Lead Company's mill in Missouri.

Wall Jig.[4]—This jig is used with good results at the Cactus mill, Newhouse, Utah. It is similar to the Hancock jig except that each compartment is separate allowing an independent stroke and water supply. It has a large screening capacity.

[1]A. H. Wethey, *Eng. and Min. Journ.*, 1906, LXXXII, p. 743.
[2]G. C. Bring, *Jern Kontorets Annaler.*, 1906, p. 321.
[3]*Canadian Mining Review*, 1906, XXVII, p. 24.
[4]*Eng. and Min. Journ.*, 1906, LXXXII, p. 60.

Improved Jig.[1]—For the concentration of ores containing sulphides of copper, iron and zinc a specially constructed jig is used. This has five to seven compartments and differs from the ordinary jigs in having deflectors after each of the last three compartments. These dam the water and force fine floating slimes down into the beds whence they are sucked down into the hutches. The first hutch is copper and iron pyrites, the second and third are blende mixed with pyrites and the later compartments are slime. The separation of the blende from the other sulphides in all the hutches after the first is accomplished by a roasting and magnetic process.

Deister Concentrator.[2]—This is a riffle table of the Wilfley type. The reciprocating motion and the direction of the riffles are both at right angles to the longer axis of the table, and the feed end is elevated 1 in. above the tailings end. Some of the riffles are long enough to reach all the way across the table. Between two successive long riffles are several low riffles which extend only part way across the table. The feed is distributed across the feed end. The concentrates come off the side and the tailings off the lower end. The table runs at 320 r.p.m. and requires 0.2 h.p. It gives best results on material under 20 mesh. At the Baltic copper mill, Lake Superior, it has done excellent work on the fine overflow of settling boxes.

Acme Combined Concentrating Table.[3]—This table has been the subject of experiment at Dolcoath, Cornwall. The table consists of an annular revolving convex round table surrounded by an annular concave revolving round table. Both tables are attached to the same frame and are separated by two circular launders. The overflow material from a classifier is fed upon the outer table and separated into concentrates and tailings. The tailings are waste; the concentrates are collected and elevated by a raff wheel to be reconcentrated on the inner table which makes finished concentrates and tailings which are elevated to join the feed to the outer table. The outer table is 19 ft. outside diameter and 15 ft. inner diameter and slopes $3\frac{3}{8}$ in. in 2 ft. The inner table has corresponding diameters of 11.5 and 7.5 ft. and a slope of $3\frac{7}{8}$in. in 2 ft. Samples were taken and assayed, low grade samples by vanning and concentrates by the cyanide method. The following results were obtained: Feed to table, 8.44 per cent. tin; concentrates, 68.30 per cent.; tailings, 1.12 per cent.

Feeders.—The reader is referred to the article on "Gold Milling," elsewhere in this volume.

Sand Launders.[4]—The movement of coarse sand in launders depends upon the transporting power of the water and the friction of the bottom. This friction in turn depends on the slope of the launder, the smoothness

[1]Eric Hedburg, *Mining World*, 1906, XXIV, p. 214.
[2]*Eng. and Min. Journ.*, 1906, LXXXII, p. 316.
[3]L. H. L. Huddart, *Trans.* I. M. M., 1906, XV, p. 299
[4]Thos. T. Read, *Eng. and Min. Journ.*, 1905, LXXX, p. 1119.

of the bottom, the size and shape of the sand particles and their specific gravity. For maximum efficiency the transported particles should move at a velocity which is one-half that of the transporting current.

Picking Belts.[1]—For sorting ore a speed of 35 ft. per minute is satisfactory. A somewhat higher speed may be used where the material is coarse and easily distinguished, while very close work requires reduced speed. The width of the belt is commonly between 24 and 36 in. The capacity varies with the size of material since fine material must be distributed in a thinner layer than coarse. With 3-in. lumps a 36-in. picking conveyor running 35 ft. per minute will have a capacity of 35 tons per hour of ore weighing 100 lb. per cu.ft. Under similar conditions a 20-in. conveyor will have a capacity of 20 tons. For a given width of picking belt the capacity may be taken within reasonable limits as directly proportional to the speed and to the size of the lumps.

For troughed belt conveyors where picking is not employed, the following figures apply:

Size of ore, inches............	12	8	6	4	3	2
Width of conveyor, inches......	30	24	20	16	14	12
Tons per hour of ore weighing 100 lb. per cu.ft., at 400 ft. per min........................	560	360	250	160	120	80

The size of conveyor required is determined by the size of the lumps rather than by the tons per hour to be delivered. This is because the feed chute must be wide enough to prevent choking and the conveyor wide enough to prevent spilling. In special cases the speed may go up to 700 ft. per minute. The inclination may go up to 20 deg. under ordinary conditions, and up to 26 deg. with fine material under favorable conditions. Where the belt is correctly run the cost of belt renewals will approximate 0.1c. per ton of ore delivered to the belt. The cost is greater on small conveyors than on large ones. The power will average about 0.00015 h.p. hour per ton per foot of horizontal distance carried plus 0.001 h.p. per foot of hight elevated.

Ferraris Magnetic Separator.[2]—This is the third separator invented by E. Ferraris, of Monteponi, Sardinia. It is designed to treat strongly magnetic material. It consists of a drum revolving on a horizontal axis. This drum is made up of magnet poles so arranged that poles of alternate polarity alternate around the circumference. A conveying belt brings the feed to the drum to which the magnetic material is attracted while the non-magnetic falls down by gravity. The magnetic material is carried upward by the revolution of the drum and removed by a belt which passes around the drum. The separator which has a belt 16 in. wide,

[1]E. H. Messiter, *Eng. and Min. Journ.*, 1906, LXXXI, p. 1139.
[2]*Eng. and Min. Journ.*, 1906, LXXXII, p. 1129.

operating with a current of 1.5 amperes at 110 volts, treats about 500 kg. of roasted ore per hour.

Ding's Magnetic Separator.[1]—In this machine a feed hopper delivers the ore in a thin sheet at the head end of an inclined tray 16 in. wide which is given a shaking motion to cause the material to move slowly down the tray. Just over the tray is an annular revolving wheel whose diameter is somewhat greater than the width of the tray. This wheel has cylindrical studs on its under side which are 1 in. in diameter and project 1 or 1½ in.; on the upper side is a U-shaped groove running around the circle. The axis of the wheel is at right angles to the plane of the tray. Above the wheel is a horse shoe electromagnet whose pole pieces are two arcs of circles which project downward into the U-shaped groove on opposite sides of the circle so that they are almost in contact with the revolving wheel. These pole pieces cover two arcs of the wheel which lie between the sides of the tray. The action of the electromagnet is to induce magnetism in the parts of the wheel lying beneath the poles. Magnetic particles are thereby lifted from the tray as they pass beneath the wheel. The revolution of the wheel carries them beyond the side of the tray and at the same time outside the field of magnetic influence (that is, out from beneath the poles) when the particles fall off by gravity into a chute. The non-magnetic material continues on over the end of the tray. The tray is supported on a heavy steel plate which acts as a magnetic armature. The revolving wheel is made of aluminum with laminated steel to form the grooved projections above and the studs below. Each stud and the corresponding segment of the grooved projection above together form a single piece which is insulated from all other studs and segments.

Wetherill Magnetic Separators.[2]—There are several forms of Wetherill magnets for treating weakly magnetic material. The well known cross-belt is much used in Germany. Another form called type V. has an armature revolving between the two poles of powerful electromagnets. This armature is made up of a number of soft iron annular discs insulated from one another and mounted on a shaft. The ore is fed on the top of this armature and the magnetism induced in it holds the magnetic particles until they drop off when, by the revolution of the armature, they are carried out of the magnetic field.

A form of low power magnet has a vertical core surrounded by a coil. The coil in turn is enclosed at the ends and at the circumference by an iron housing which is connected to the core and, together with the core, forms a continuous magnetic circuit. Ore is dropped in front of this housing all around its periphery. Magnetic particles are deflected inwards and fall

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[1]*Eng. and Min. Journ.*, 1906, LXXXI, p. 749.
[2]*L'Electricien*, 1906, XXXI, pp. 100, 120.

into a central hopper while non-magnetic material falls vertically into an annular hopper.

WETHERILL SEPARATOR: RESULTS OF TESTS ON ROASTED BLENDE.

Size of Material.	Per Cent. of Total.	Per Cent. Zinc.	Kg. per Hour.	Magnetic Product. Per Cent. of Weight.	Magnetic Product. Per Cent. Zinc.	Non-magnetic Product. Per Cent. of Weight.	Non-magnetic Product. Per Cent. Zinc.	Extraction. Per Cent.
4 to 2.4 mm.	30.29	34.25	1,000	10.18	6.1	20.11	48.5	94.01
2.4 to 1.2 mm.	33.05	34.15	1,050	13.23	6.5	19.82	52.6	92.37
1.2 to 0.6 mm.	14.18	35.25	1,000	5.34	5.7	8.84	53.1	93.91
0.6 to 0 mm.	22.48	32.69	400	8.05	6.5	14.43	47.3	92.86
	100.00	34.00		36.80	6.27	63.20	50.15	93.21

Magnetic Separating Table.[1]—The Lenschner magnetic table is an ordinary convex revolving table which is made of wrought iron and has one pole of an electromagnet placed just above the surface while the other pole is placed beneath the table. These poles are near the circumference of the table and are long and narrow, the length being sufficient to extend about one-tenth the distance around the table. In separating blende and spathic iron which are of the same specific gravity the magnetism induced in the iron holds the spathic iron while the blende is washed off. Several of these tables have been installed at Friedrichssegen and the Humboldt company has one in its experimental plant.

High Grade Magnetic Concentrates.[2]—No. 1 mill of Witherbee, Sherman & Co., Port Henry, N. Y., recently made some high grade magnetic concentrates running 71.57, 71.20, 71.16, 71.10 and 71.02 per cent. iron, 0.89 per cent. silica and 0.006 per cent. phosphorus. Sizing tests of this material showed the following: On 8 mesh, 0 per cent.; 8-16 mesh, 3.71 per cent.; 16-20 mesh, 10.64 per cent.; 20-40 mesh, 44.23 per cent.; 40-60 mesh, 13.08 per cent.; 60-80 mesh, 8.91 per cent.; 80-100 mesh, 7.69 per cent.; through 100 mesh, 11.74 per cent.

Dry and Wet Electromagnetic Separation at Budapest.[3]—The Hernadthaler Iron Industrie Aktien Gesellschaft at Budapest, Hungary, has a plant at Krompach with a capacity of 360 tons in 20 hours. The ore consists of spathic iron and quartz and slate containing finely disseminated pyrite and chalcopyrite and fahlore. It analyzes 27.37 per cent. iron, 0.911 copper, 1.511 sulphur and 22.88 insoluble. The dry separator for treating material over ¼ mm. is shown in Fig. 3 and the wet form for material under ¼ mm. in Fig. 4. In both forms the feed is delivered upon a cylinder revolving rapidly in the direction shown by the arrows. This cylinder consists of a center shaft with five flanges upon it. Between

[1]*Zeit. f. B. H. u. S. im Preuss*, 1906, LIV, p. 278.
[2]*Iron Trade Review*, April 5, 1906, XXXIX, p. 35.
[3]C. Blömeke, *Metallurgie*, 1906, III, p. 721.

these flanges the coils of the electromagnets are so wound that the flanges become magnetic poles of opposite polarity. These flanges widen outward so that at the circumference adjoining flanges are only a little distance apart, being separated by a ring of non-magnetic substance. The feed is distributed at the four points where these rings occur along the length of the cylinder. Centrifugal force first throws off the non-magnetic particles. The weakly non-magnetic follow next while the strongly magnetic remain in contact with the cylinder until they come in proximity to the pointed wheel shown at the left. The magnetism induced upon these points causes the magnetic particles to hop over to the points and the revolution of the wheel carries them out of the magnetic field when the particles fall off by gravity. An arrangement of partitions divides the products into concentrates, middlings and tailings. A dry machine with four rings treats 0.75 ton per hour of material between 2½ and ¼ mm. while the wet machine treats 0.4 ton of material below ¼ mm. About 70 per cent. of the feed is magnetic.

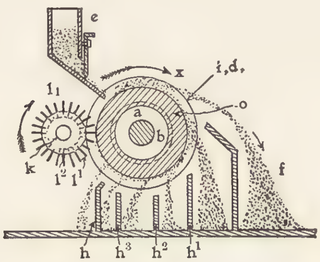

FIG. 3.—DRY SEPARATOR.

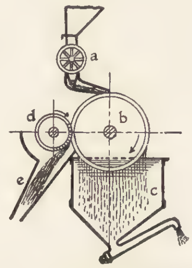

FIG. 4.—WET SEPARATOR.

The plant at Krompach contains 24 dry and four wet separators. The spathic iron product contains 33.5 per cent. iron and 0.17 per cent. copper. It is roasted and used as an iron ore. The middlings product from the dry separators amounts to about 0.47 per cent. of the total quantity and is ground in ball mills to ¼ mm. and treated on the wet separators. The non-magnetic product consisting of quartz, slate and sulphides is concentrated by classifiers and tables. Results are given of successful tests of these machines separating tin and wolframite and also an iron conglomerate. The dry separator on Krompach ore uses 1400 watts of electricity and 0.5 h.p. for driving it. The wet separator requires 1000 watts and 0.25 h.p.

Sutton-Steele Dielectric Separator.[1]—This machine differs from other static machines, in that it does not depend upon the difference in electric conductivity of the substances to be separated, but upon inducing dielectric impedance in the particles to be separated. This is induced by several

[1]R. C. Canby, *Eng. and Min. Journ.*, 1906, LXXXI, p. 893.

different methods of interrupting or reversing the current, in conjunction with a convective discharge through the mass of material on to the separating roller. The essential parts of the machine are the feeders, the revolving separating rollers, 5 ft. long and $1\frac{15}{16}$ in. diameter, the rows or combs of points so arranged as to discharge onto the roller, brushes for removing the adhering particles from the opposite side of the roller, and screw conveyors for removing the separated products.

Dry Concentration.[1]—This process has been extensively used in Western Australia, Queensland and New South Wales. The Draper crushing and blowing machine is mentioned as doing good work.

Dry Washing at Manhattan, Nevada.[2]—A cut is shown of a Freeman dry washing machine.

Sutton Steele Dry Process.[3]—The Sutton Steele table is a bumping table which resembles the Wilfley, except that it is operated dry. The table top is pervious so that air under 0.5 oz. pressure is continually discharged up through the ore, thereby giving the ore great mobility and allowing a ready separation through the action of the slope of the table and the bumping action. The table makes 400 throws per minute, and it is claimed to have three times the capacity of a wet table and to give a sharper separation. For heavy lead ores a better separation has been obtained by using riffles running the whole length of the table instead of ending on a diagonal as in the Wilfley. Some results of tests seem to indicate better results than can be obtained with wet tables.

Friction Process of Ore Dressing.[4]—The claim is made that when an ore consisting of hard gangue (as quartz) and a soft mineral (as galena) is crushed, the mineral remains on the points of the particles of gangue and may be removed by a gentle rubbing. The plant is as follows:—1. Graded crushing to desired size. 2. Treatment with iron in a slowly revolving cylinder having a hard rough lining. Water may or may not be used. 3. Screening of the product to remove most of the valuable fines. 4. Magnetic treatment to remove the iron to go back to the cylinder. 5. Settling in a spitzkasten yielding spigot to waste and overflow of slime, which is collected in a settling tank and saved. Some experimental results are given, together with a full discussion of the principles of this process, and the causes of the formation of soft points.

Attraction of Some Common Minerals for Residuum Oils.[5]—Blocks of minerals having a surface of 1 or 2 sq. cm. and 0.2 or 0.4 cm. thickness were tried at 60 deg. F. on oil having a specific gravity of 0.925 and a viscosity of 12,241 at 65 deg. F. The oil was covered with a layer of water.

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[1]*Mining Reporter*, 1906, LIV, p. 260.
[2]C. T. Rice, *Eng. and Min. Journ.*, 1906, LXXXII, p. 581.
[3]R. C. Canby, *Eng. and Min. Journ.*, 1906, LXXXII, p. 893.
[4]Otto Witt, pamphlet by Craz & Gerlach, Freiberg, Saxony. Abstracted in *Oest. Zeit.*, 1906, LIV, p. 209, and *Mining Magazine*, 1906, XIII, pp. 325, 484.
[5]J. F. Hamilton, *Proc. Can. Min. Inst.*, 1904, VII, p. 185.

The blocks were allowed five seconds of contact with the oil, and then a weight of three grams usually was used on a pulp balance to pull the block free from the oil. The relative times required to pull the different minerals away from the oil were measured by a stop-watch. Results are shown in the accompanying table:

ATTRACTION OF MINERALS FOR OILS.

	Size Sq. cm.	Time of Rest. Seconds.	Weights Used. Grams.	Average Result of Time. Seconds.	Size Sq. cm.	Time of Rest. Seconds.	Weights Used. Grams.	Average Result of Time. Seconds.
Pyrrhotite	1	5	3	3.1	2	5	6	1.95
Pyrite	1	5	3	3.25	2	5	6	2.2
Mispickel	1	5	....	2.6	2	5	6	1.8
Chalcopyrite	1	5	3	2.9	2	5	6	2.3
Gray copper	1	5	3	2.8	2	5	5	2.7
Galena	1	5	3	2.6	2	5	5	2.7
Zinc blende	1	5	3	2.9	2	5	5	2.5
Quartz	1	5	2.3	3.3	2	5	4	2.1
Mica schist	....	....	2.1	2.2	2	5	3	1.9
Calcite (pink)	....	....	3	2.2	....	....	....	....
Calcite (white)	....	....	2	3.7	....	....	....	....
Hematite	....	....	3	3.25	....	....	....	....
Magnetite	....	....	3	3.2	....	....	....	....

Flotation Processes.—Considerable has been written on this subject during the past year. The reader is referred to the article on "Zinc," elsewhere in this volume.

General Milling Practice.

Milling Practice in Colorado.[1]—A series of articles is devoted to a description of the milling practice in several districts in Colorado. Each district is discussed in considerable detail. The following list shows the districts covered and the scheme in each.

Idaho Springs: First crushing by stamps or rolls; jigging of coarse material; regrinding in stamps, Huntington or Kinkead mills; fine concentration on tables and vanners.

Summit County: Stamping, amalgamation and fine concentration.

Telluride: Stamping, amalgamation and fine concentration followed in some cases by regrinding in Huntington mills or tube mills, a second concentration or amalgamation and finally cyaniding.

Creede: Rolls, trommels, jigs and tables. Middlings recrushed in Chile mills. Canvas plant for slimes.

Silverton: Rolls, trommels, jigs and tables; canvas plant or slimers; recrushing of middlings in Chile mills or stamps. Some mills use only stamping and fine concentration.

Aspen: Rolls, trommels, jigs, tables and vanners with Huntington mills for recrushing. Some mills omit the trommels and jigs.

[1] *Mining Reporter*, 1906, LII, p. 636; 1906, LIV, pp. 229, 286, 341, 386, 410.

The Dives-Pelican and Seven-Thirty Mill at Silver Plume, Colo.[1]—The ore contains silver, lead and zinc minerals in a quartz gangue. It is dumped upon (1).

1. Grizzly with 1¼-in. spaces. Oversize to (2); undersize to (3).
2. Two Blake breakers, 10x20 in. crushing to 1¼ in. To (3).
3. Bucket elevator and distributing belt conveyor. To (4).
4. Two-compartment ore bin. By belt conveyors to (5).
5. One pair of rolls, 16x36 in., crushing to ⅜ in. To (6).
6. Elevator. To (7).
7. No. 1 trommel with 4-mesh wire cloth. Oversize to (8); undersize to (9).
8. One pair of rolls, 16x36 in. To (6).
9. No. 2 trommel 36x72 in., with 5-mesh wire cloth. Oversize to (16); undersize to (10).
10. One No. 3 trommel, 36x72 in., with 10-mesh wire cloth. Oversize to (17); undersize to (11).
11. One No. 4 trommel, 36x72 in., with 16-mesh wire cloth. Oversize to (18); undersize to (12).
12. One No. 1 hydraulic classifier. Spigot to (21); overflow to (13).
13. One receiving tank. Spigot to (15); overflow to (14).
14. Four hydraulic classifiers in series. Spigots to (22); overflow of last to waste.
15. Four hydraulic classifiers in series. Spigots to (22); overflow of last to waste.
16. Two double four-compartment Harz jigs. Concentrates; tailings to (19).
17. One double four-compartment Harz jig. Concentrates; tailings to (19).
18. One double four-compartment Harz jig. Concentrates; tailings to (20).
19. Two unwatering screens. Oversize to (20); undersize to waste.
20. Three 5-ft. Huntington mills with 30-mesh screens. By three Frenier sand pumps to (13).
21. Three Cammett tables. Concentrates; tailings; water to (20).
22. Six Wilfley tables. Concentrates; middlings of four coarse tables to (23); tailings.
23. One Wilfley table. Concentrates; tailings.

The mill treats 150 tons in 24 hours. The crushing plant is run only eight hours out of the 24. Three men are required per eight-hour shift. Electric power is supplied through eight motors; one 50-h.p. motor for the Blake breakers, one 15-h.p. for the first bucket elevator, one 5-h.p. for the distributing conveyor over the bins, one 2-h.p. for the feed belt conveyor from the bins, one 50-h.p. for the rolls and trommels, one 50-h.p. for the Huntington mills, sand pumps, unwatering screens and a large triplex pump for water, one 20-h.p. for the jigs, one 7½ h.p. for the tables.

When treating silver-lead ore three products are made: A silver-lead concentrate, a zinc concentrate and a sand product containing silver. The ore analyzes a trace of gold, 12 to 15 oz. silver per ton, 1 to 2 per cent. lead and 1½ to 3¾ per cent. zinc. From 8 to 20 tons are concentrated into one. The silver-lead concentrates assay about 200 oz. silver, 0.05 oz. gold, 20 to 25 per cent. lead and 15 to 20 per cent. zinc. The zinc product contains 35 per cent. zinc and 75 oz. silver. The sand product from (23) runs about 20 per cent. iron, 25 per cent. silica and 40 oz. silver. Tailings assay about 3 to 4 oz. silver. The extraction is about 60 per cent.

The zinc-lead ore assays 10 to 12 per cent. zinc, 5 per cent. lead and 8 to 10 oz. silver. The lead product from this assays 50 per cent. lead, 5 to 10 per cent. zinc and 30 oz. silver. The zinc product contains 40 per cent. zinc, 3 to 5 per cent lead and 15 to 20 oz. silver. The concentration is 4:1 or 5:1. The extraction on this class of ore is 53 per cent. of the silver, 95 per cent. of the lead and 87 per cent. of the zinc.

The Hanna Mill.[2]—In the Hanna M. & M. Company's 125-ton mill at Capitol City, Colo., the ore which contains quartz and massive sulphides

[1]*Mining Reporter*, 1906, LIII, p. 556.
[2]*Mining Reporter*, 1906, LIV, p. 82.

of lead, copper, zinc and iron is brought by aerial tramway, weighed and dumped into (1).

1. Two bins holding 50 tons each. By gates to (2).
2. Belt picking conveyor. Shipping ore to (3) separately and thence to smelter; residue to (3).
3. Breaker crushing to $\frac{3}{4}$ or 1 in. To (4).
4. Mill bins holding 50 tons. By plunger feeders to (5).
5. Coarse rolls crushing to $\frac{1}{4}$ in. To (6).
6. No. 1 trommel, 3-mesh. Oversize to (5); undersize to (7).
7. No. 2 trommel, 4-mesh. Oversize to (10); undersize to (8).
8. No. 3 trommel 6-mesh. Oversize to (10); undersize to (9).
9. Dewatering box. Spigot to (10); overflow to (12).
10. Three Harz jigs, three-compartment each. Lead concentrates; zinc-iron middlings to (20); tailings to (11).
11. One 5-ft. Huntington mill with 12- and 16-mesh screens. By bucket elevator to (12).
12. Sherman classifying tank. Spigot to (13); overflow to (15).
13. Four Wilfley tables. Lead concentrates; zinc-iron middlings to (20); lead-zinc and zinc-silica middlings returned to table; tailings to (14).
14. One Huntington mill with 20-mesh screens. To (15).
15. Four Sherman classifying tanks in series. Spigots to (17); overflow to (16).
16. Eight Sherman settling tanks in series. Spigots to (18); overflow elevated as clear water for the mill.
17. Four Wilfley tables. Products like (13) except tailings of first two tables to waste and of last two to (19).
18. One Wilfley slimer. Lead concentrates; zinc-iron middlings to (20); tailings to waste.
19. Four Wilfley slimers. Product like (18).
20. One Wilfley table. Concentrates to (21); tailings to waste(?)
21. Bartlett-Snow rotary drier treating 20 tons in 10 hours. To (22).
22. Blake static separators. Zinc product; copper-iron product.

An average analysis of the milling ore is 0.5 to 1.25 per cent. copper, 5 to 10 per cent. lead, 5 to 10 per cent. zinc, $2 gold and 4 to 10 oz. silver per ton. The lead product from the jigs contains 54 to 61 per cent. lead, 0.5 to 0.75 oz. gold and 12 to 16 oz. silver. The zinc product of the Blake machines runs about 37 per cent. zinc, 5 to 10 per cent. iron, 2 to 3 per cent. copper, 5 to 10 oz. silver and 0.15 to 0. 20 oz. gold per ton. The copper-iron product contains 11 per cent. copper, 20 to 30 per cent. iron, 0.3 to 0.5 oz. gold and 8 to 10 oz. silver.

The Gold Prince Mill, Near Silverton, Colorado.[1]—The ore contains sulphides of lead, zinc, iron, and copper carrying gold and silver values. At the mine it is crushed in a No. 6 K Gates breaker and passes to two steel bins having a capacity of 500 tons each. Thence it goes by aerial tramway 2.37 miles to (1).

1. Bins holding 1000 tons. By plunger feeders and two belt conveyors to (2).
2. Two trommels with 1.5-in. holes. Oversize to (3); undersize to (4).
3. Two Farrel jaw breakers, 13x30 in., crushing to 1½ in. To (4).
4. Two belt elevators and distributing conveyor to (5).
5. Ten stamp bins of steel holding 50 tons each. By Challenge feeders to (6).
6. One hundred stamps, with 16- to 30-mesh screens. To (7).
7. Twenty amalgamated plates, 4½x12 ft., to (8).
8. Twenty traps. To (9).
9. Ten tanks. By Frenier sand pumps to (10).
10. Ten two-compartment spitzkasten. 1st spigot to (11); 2d to (12); overflow to (17).
11. Twenty Card tables. Lead concentrates; zinc-iron concentrates; tailings to (13).
12. Ten Card tables. Lead concentrates; zinc-iron concentrates; tailings to (13).
13. Ten cone classifiers. Spigots to (14); overflow to (17).
14. Five tube mills 14 ft. long, crushing 100 tons in 24 hours to 80-mesh. To (15).
15. Ten two-compartment spitzkasten. Spigots to (16); overflow to (17).
16. Card tables. Lead concentrates; zinc-iron concentrates; tailings to waste.
17. Card slimers. Lead concentrates; zinc-iron concentrates; tailings to waste.

The concentrates are dried in Argall multi-tubular driers and it is planned to treat the zinc-iron concentrates by magnetic separators. The stamps have concrete mortar beds, Homestake mortars and chrome steel parts except the stems which are special steel. The stamps weigh 950 lb. each

[1]H. J. Baron, *Mining Reporter*, 1906, LIV, p. 204.

and crush five tons per day, dropping 5 to 7 in., 100 to 105 drops per minute. Electricity is used for power, several small motors having been installed for the various parts of the mill. The mill buildings are sheathed with 1-in. boards covered with heavy building paper which in turn is covered with No. 22 corrugated steel siding. Concrete floors are used. The building is heated by a mechanically forced hot air draft plant from two coal fired marine type 100-h.p. boilers. Tests indicate a recovery of 10 to 40 per cent. of the gold and silver by amalgamation and the total recovery by amalgamation and concentration will be 85 per cent. The magnetic plant is expected to increase this to 90 per cent.

Boston Consolidated Mill.[1]—This mill is now under construction at Bingham, Utah, for treating the low grade porphyry containing copper sulphides. Ore is brought in railroad cars to (1).

1. Crude ore bin holding 12,000 tons. By 40 feeding gates to two belt conveyors to (2).
2. Two No. 6 K gyratory breakers. To (3).
3. Two trommels with 1.25-in. holes. Oversize to (4); undersize to (5).
4. Two No. 5 gyratory breakers crushing to 2.5 in. To. (5)
5. Two belt conveyors. By distributing conveyors to (6).
6. Crushed ore bin holding 12,000 tons. By feeders to (7).
7. Two hundred and twelve Nissen stamps, weighing 1500 lb. and crushing 10.5 tons each in 24 hours through a 28-mesh (No. 28 wire) screen. To (8).
8. Classifiers. Four products to (9).
9. Two hundred and eighty-six Wilfley tables. Concentrates; tailings of Wilfleys treating two coarser products to waste; tailings of Wilfleys treating two finer products to (10).
10. Callow cone settling tanks. Settlings to (11).
11. Two hundred and thirty-four Johnston vanners. Concentrates; tailings.

Tests indicate that Wilfley tables alone will give an extraction of 65 to 67 per cent. of the copper. The Johnston vanners increase this to 75 or 80 per cent. The ratio of concentration will be 16:1 up to 20:1.

Mill of Utah Copper Company.[2]—The low-grade porphyry in Bingham, Utah, contains chalcocite and bornite and a little chalcopyrite. The scheme of the first mill is as follows:

1. 1000-ton ore bin. By feed hopper to belt conveyor to (2).
2. Grizzly with 1-in. spaces. Oversize to (3); undersize to (4).
3. One No. 7½ Gates breaker. To (4).
4. One 16-in. belt elevator. To (5).
5. Trommel with 1½-in. holes. Oversize to (6); undersize to (7).
6. One No. 4 Gates breaker. To (4).
7. One 16-in. belt elevator and an inclined conveying belt. To (8).

From this point the mill is in two halves and only one half is described.

8. An inclined shaking screen with ⅜-in. holes. Oversize to (9); undersize to (11).
9. One 250-ton bin. By plunger feeder to (10).
10. One pair of 36x15-in. belted Gates rolls. To (12).
11. One 250-ton bin. By plunger feeder to (12).
12. One 16-in. belt elevator. To (13).
13. Two trommels with 0.09-in. holes. Oversize to (15); undersize to (14).
14. Two trommels with 0.021-in. holes. Oversize to (16); undersize to (17).
15. Fine rolls. To (12).
16. Three 6-foot Chile mills with 0.015-in. screen. To (17).
17. One three-cone classifier. 1st spigot to (18); 2d to (20); 3d to (20); overflow to (22).
18. Three-compartment Harz jigs. Hutches to (19); tailings to (21).
19. Wilfley tables. Concentrates; light tailings to waste; heavy tailings and slimes to (21).
20. Wilfley tables, Products like (19).
21. Pump to cylindrical classifier. Spigot to (16); overflow to (22).
22. Two classifiers with three spigots each, each classifier followed by 12 settling tanks. Classifier spigots to (23); tank spigots to (24); overflow clear water for mill.
23. Wilfley tables. Concentrates; tailings.
24. Twenty-seven Frue vanners. Concentrates; tailings.

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[1] L. H. Beason, *Eng. and Min. Journ.*, 1906, LXXXII, p. 1216.
[2] *Eng. and Min. Journ.*, 1906, LXXXII, p. 434.

This mill was built three years ago and treats 800 tons in 24 hours. The ore averages 2 per cent. copper and the concentrates contain 30 per cent. copper. A 6000-ton mill along the same lines is in process of erection.

Wall Concentrating Mill.[1]—In this custom mill in Bingham Cañon, Utah, are several special pieces of apparatus. The general scheme of the mill is as follows: Ore from wagons is dumped into (1).

1. Ore bin holding 200 tons. To (2).
2. Grizzly with 1-in. spaces. Oversize to (3); undersize to (4).
3. Blake breaker, 9x15 in., crushing to ⅝ in. To (4).
4. One pair of Wall corrugated rolls. By cross-belt elevator to (5).
5. One trommel, 5-mesh No. 14 wire. Oversize to (6); undersize to (7).
6. Wall jig with three compartments. Concentrates to bin; tailings to waste.
7. Shaking screen having 10-, 20- and 30-mesh screen. All over 30-mesh to (8); undersize to (9).
8. Two Wall jigs and one Harz jig. Lead concentrates to bin; tailings to waste.
9. Two Wilfley and two Wall tables. Concentrates to bin; tailings to waste.

The tailings of the mill run about 0.3 per cent. lead and it is claimed that 88 per cent. of the lead is saved in the concentrates. The mill treats 41 to 46 tons in 8 hours. Power is supplied by a 60-h.p. boiler and a 40-h.p. engine. About 1500 ℔. of coal are burned per day. The labor required is one man in bin at $2.50, one engineer at $3.25, one feeder at $2.75, one wheeler at $3, one jigman at $3.50 and one foreman at $5.50.

The Wall rolls are 16x16 in. and have helical corrugations, about 2 in. deep, meshing together. They are claimed to give increased capacity and decreased fines. Crossed belt elevators have the advantage that falling grit is thrown off the lower pulley and that the discharge spout may be placed lower and to better advantage. The Bemis shaking screen has a downward movement by a cam and an upward movement by springs and bumper. The Wall jig is a movable sieve jig in which strong pulsion and weak suction are obtained by means of a cam and lever or tappet. Each sieve is run independently. The Wall table is a riffle bumping table 8x5 ft. Its mechanism is similar to the old Gilpin county bumper except that it bumps with the spring in tension intead of compression. A space of 2 ft. 4 in. at the discharge end is covered with rubber having ⅛-in. corrugations. The riffles are ¾ in. wide and ¾ in. apart.

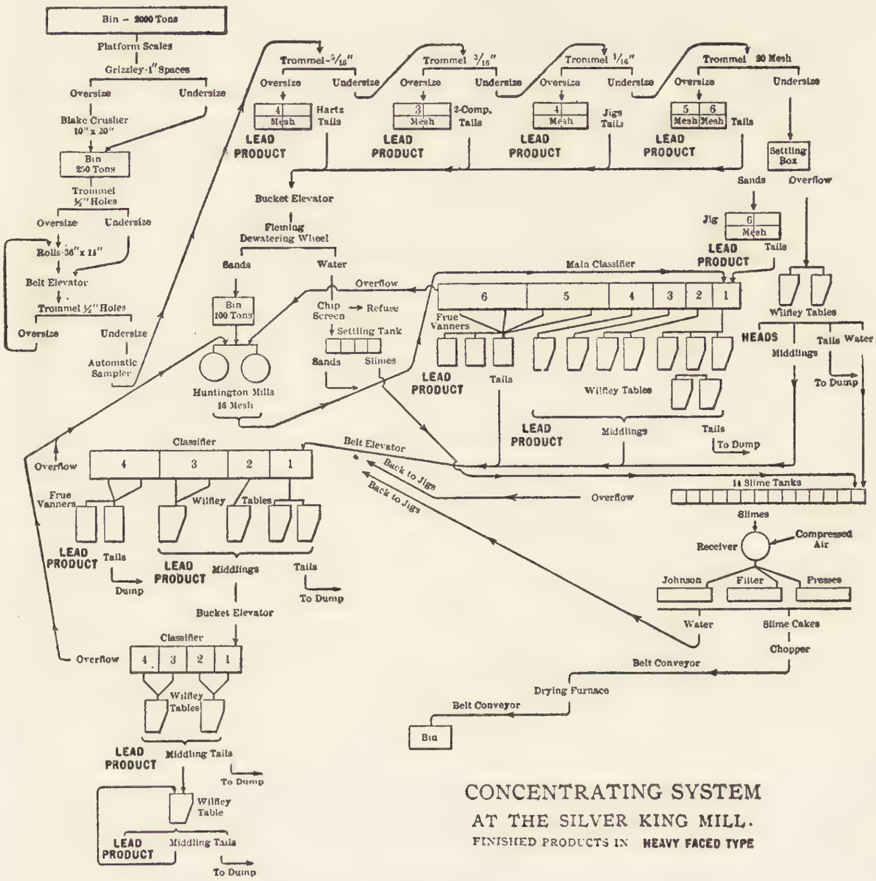
Cactus Mill.[2]—This was described in THE MINERAL INDUSTRY, Vol. XIV, under head of Newhouse mill. This later description is somewhat fuller.

Silver King Mill.[3]—The scheme of treatment is shown in the accompanying tree (Fig.5). The 14 slime tanks are some V-shaped and some rectangular. They are 30 to 35 ft. long, 3 ft. deep and about 3 ft. wide at the top. Some difficulty is found in getting the slime thick enough for the filter presses. The receiver is of steel, 6 ft diameter and 10 ft. high. Compressed air under a pressure of 100 ℔. per sq.in. forces the slime into the filter press. Two of the filter presses are made of forty-four 26-in circular discs while the

[1]C. T. Rice, *Eng. and Min. Journ.*, 1906, LXXXII, p. 1009
[2]L. A. Palmer, *Mines and Minerals*, 1906, XXVI, p. 337; *Min. Magazine*, 1906, XIII, p. 322.
[3]*Eng. and Min. Journ.*, 1906, LXXXII, p. 202.

third is rectangular and has 30 plates 24 in. sq. The pressed slime contains about 18 per cent. moisture. When dry it assays about 8 per cent. lead, 25 to 30 oz. silver per ton and $4 gold.

Power is furnished by two tubular boilers and two Corliss engines. In the concentration plant on two shifts are two jigmen, two look-outs and oilers, two Huntington mill men, two table men and four shovelers.



CONCENTRATING SYSTEM
AT THE SILVER KING MILL.
FINISHED PRODUCTS IN **HEAVY FACED TYPE**

FIG. 5.

The slime plant and engine room are run in three shifts and require three mill men and three engine men. All receive $3 per day except the shovelers. Additional men are one machinist at $4, one helper at $3, one repair man at $4, one helper at $3, and night shift boss at $4.

Daly Judge Mill.[1]—The ore is received in two ore bins, each of 450 tons capacity, and goes thence by car over scales to (1).

[1] *E'g. and Min. Journ.*, 1906, LXXXII. p. 248.

1. Grizzly. Oversize to (2); undersize to (3).
2. Blake breaker. To (3).
3. No. 1 rolls. Sample taken every fifteen minutes. To (4).
4. No. 2 rolls. To. (5).
5. A 3-mesh screen. Oversize to (6); undersize to (8).
6. No. 3 rolls. To (7).
7. A 2-mesh screen. Oversize to (6); undersize to (8).
8. Elevator. To (9.)
9. Three trommels in series, 3-, 4- and 6-mesh. Over 3-mesh to (10); through 3 on 4-mesh to (10); through 4 on 6-mesh to (10); through 6-mesh to (16).
10. Three Harz jigs, three compartments each. Lead concentrates; tailings by elevator to (11).
11. A Fleming dewatering wheel. Pulp to (14); overflow water to (12).
12. An 8-mesh trommel. Oversize to (14); undersize to (13).
13. Settling tank. Settlings to (18); overflow clear water to (15).
14. A 50-ton bin. By three plunger feeders to (15).
15. Three Huntington mills with 16-mesh screens. To (16).
16. Classifier with 11 compartments. Spigots to (18); side overflow clear water to jigs and other machines; end overflow to (17).
17. Classifier with eight compartments. Spigots to (19); overflow clear water to other machines or to waste.
18. Ten Wilfley tables. Galena-pyrite concentrates; galena-blende-pyrite middlings by Frenier sand pump to (20); tailings to (22).
19. Two Wilfley tables and one Wilfley slimer. Lead concentrate; blende pyrite middlings; tailings to waste.
20. A four-compartment settling tank. Spigots to (21).
21. Four Wilfley tables. Lead concentrates; blende-pyrite middlings; tailings to (22).
22. Frenier sand pump. To (23).
23. A three-compartment settling tank. Spigots to (24).
24. Three Wilfley tables. Lead concentrates; blende-pyrite middlings; tailings to waste.

All blende-pyrite middlings are stored awaiting a favorable market. The mill treats about 200 tons in 24 hours. The feed averages 10 per cent.

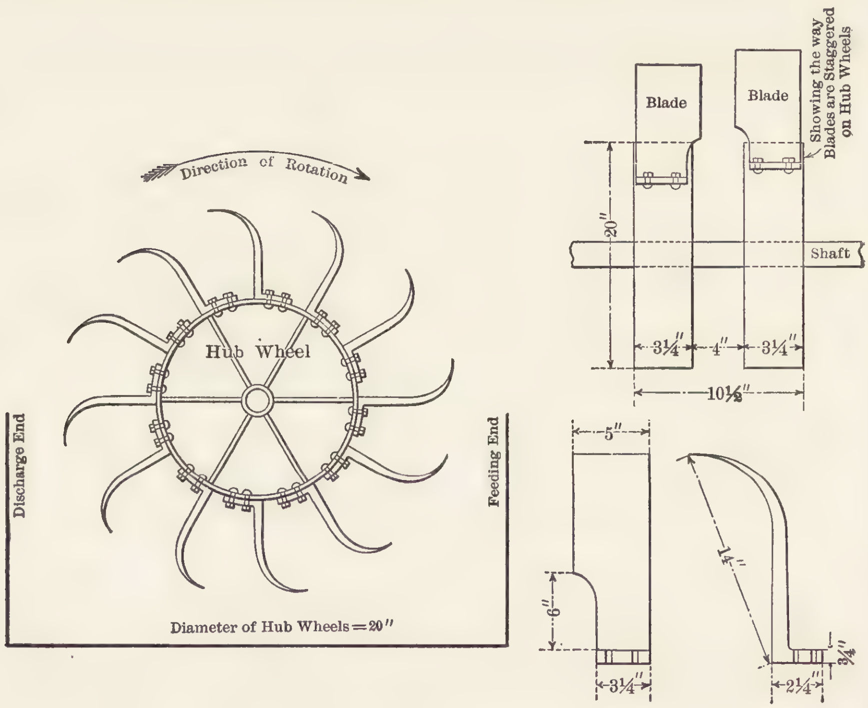


FIG. 6.—FLEMING DEWATERER.

lead, 9 per cent. zinc and 7 oz. silver per ton from the contact deposits, and 6 per cent. lead, 5 oz. silver and no zinc from the lower fissure deposit. The two classes of ore are treated separately, first one, then the other.

The lead concentrates average 33 per cent. lead, 7 per cent. zinc, 21 per cent. iron, 3 per cent. silica, 18.5 oz. silver and 0.11 oz. gold. The zinc concentrate averages 20 to 25 per cent. zinc, 2 to 4.5 per cent. lead, 15 per cent. iron, 12 per cent. silica, 9 oz. silver and 0.04 oz. gold. The tailings carry 0.7 per cent. lead, 3 per cent. zinc and 1.5 oz. silver.

Power is furnished by one 150-h.p. marine boiler (two 60-h.p. return tubular boilers in reserve). Four to six tons of coal are burned in 24 hours. The exhaust steam heats the mill. Eleven men per eight-hour shift are required: one engineer at $3, one trammer at $2.75, one breakerman at $2.75, one oiler and look-out at $2.75, one jig man at $3, one Huntington man at $3, two table men at $3, three shovelers at $2.50.

The Fleming dewaterer consists of a wheel revolving in a tank (see Fig. 6). This pushes the sands out of the water while the water overflows the side. The wheel revolves nine times per minute.

The 11-compartment classifier is shown in Fig. 7. The other classifiers are similar.

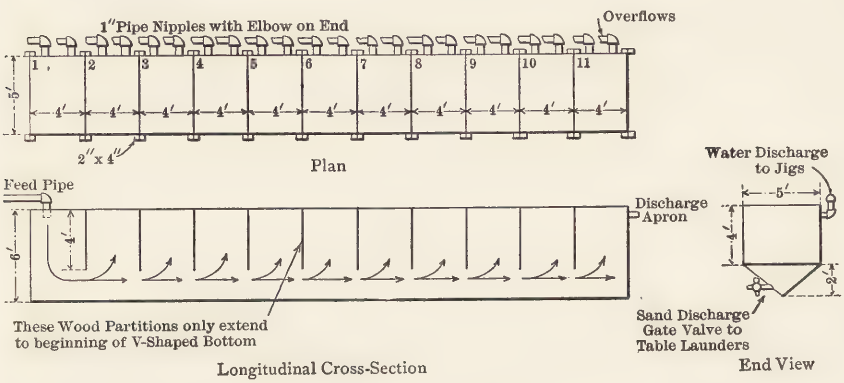


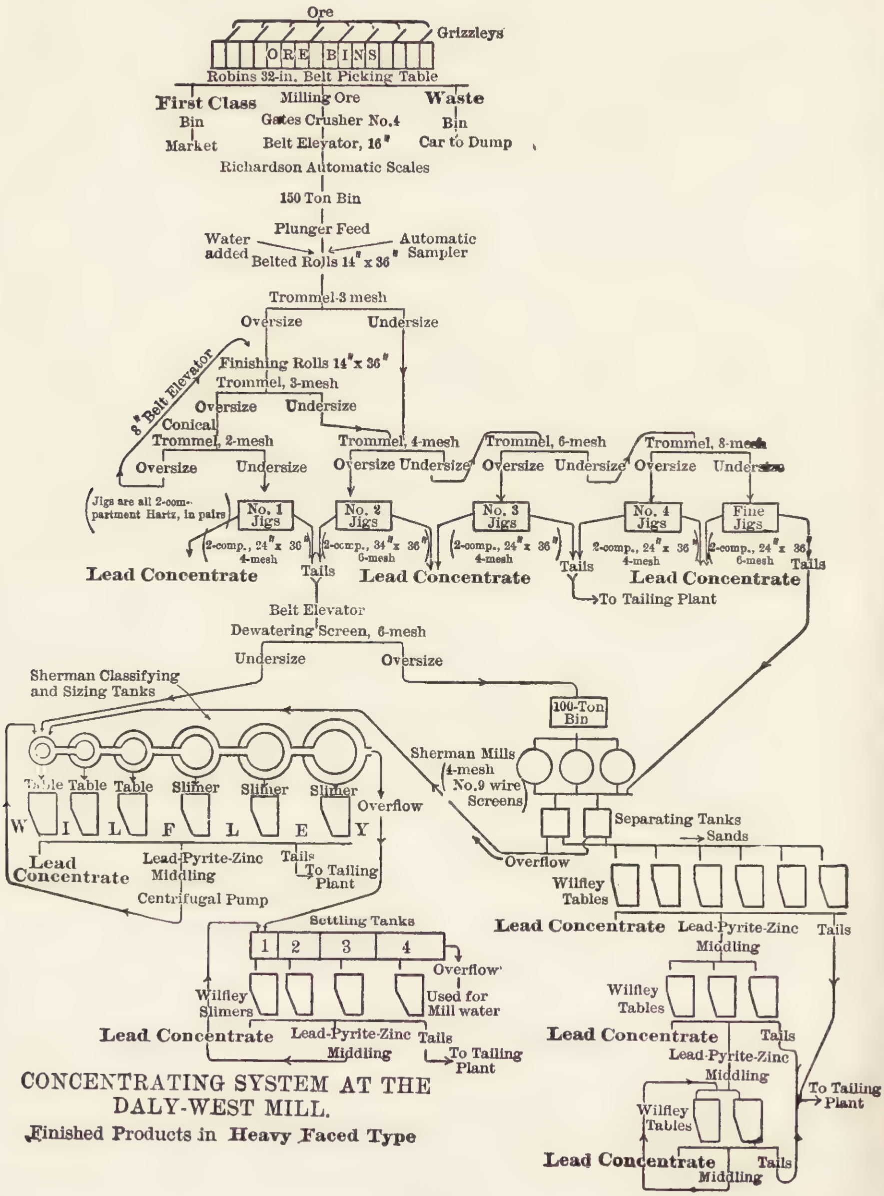
FIG. 7.—SETTLING TANK.

Daly West Mill.[1]—The scheme of this mill is shown in Fig. 8. In the tailings plant the table tailings come to (1) and the fine jig tailings to (2).

1. One 8-mesh trommel. Oversize to waste; undersize to (2).
2. Settling tank. Spigot by bucket elevator to (3); overflow to (6).
3. One 8-mesh unwatering trommel. Oversize to (5); undersize to (4).
4. One Harz jig, 2-compartments, each 24x36 in. Concentrates; tailings to waste.
5. A 50-ton bin supplying a 5-ft. Sherman centrifugal mill with 4-mesh screen. To (6).
6. Six Sherman classifying tanks in series. 1st, 2d, and 3d spigots to three Overstrom tables (7); 4th and 5th to two Wilfley slimers (7); 6th to Sherman slimer (7); overflow to (8).
7. Three Overstrom tables, two Wilfley slimers and one Sherman slimer. Concentrates; middlings pumped to (6); tailings to (10).
8. Nineteen Sherman settling tanks in series, each 6 ft. diameter and 5 ft. deep. Settlings to (9); overflow pumped back to be used as wash water.
9. Wilfley slimer. Concentrates; tailings.
10. Four settling tanks in series. Coarse settlings to waste; fine settlings (mostly under 200-mesh) by centrifugal pump to (11).
11. Distributing tank and two Sherman slimers. Concentrates; middlings to (10); tailings to (12).
12. Settling tank. Settlings to waste; overflow pumped back as wash water.

[1]*Eng. and Min. Journ.*, 1906, LXXXII, p. 53.

The mill has a capacity of 400 tons per 24 hours. The ore is galena and blende in gangue. The mill saves 95 per cent. of the lead, 75 to 80



CONCENTRATING SYSTEM AT THE DALY-WEST MILL.
Finished Products in **Heavy Faced Type**

FIG. 8

per cent. of the silver, and 50 per cent. of the zinc on ore running 4.5 per cent. lead, 11.3 oz. silver per ton and 5.2 per cent. zinc. The concen-

trates contain 30 per cent. lead, 15 per cent. zinc, 6 to 7 per cent. iron, 50 oz. silver and 14 per cent. silica. The tailings assay 0.3 per cent. lead (wet assay), 2 per cent. zinc and 2 to 3 oz. silver. The concentration is 7:1. The first set of tables in the tailings mill save $10,000 per month and the second set $3000 per month. The horse power used is 300. The water flowing in the system is 350 gal. per minute; the actual loss of water in the tailings is 40 gal. per minute.

The special feature of the mill is the regrinding and retreatment of tailings. The Sherman mill is an improved Huntington mill in which the muller stems rotate in an oil bath and also project upward and are held lightly against the die ring by tension springs which makes the mullers wear round. The Sherman classifier has an annular space between an inside and outside cylinder. The outer cylinder has a conical bottom. Material enters the inner cylinder, passes down through a screen (to remove sticks) and rises in the annular space to overflow the rim of the outer cylinder. The sectional area of the inner cylinder is half that of the annular space. The following classifier has twice the area of the one preceding. The Sherman table has 24 trays like the Wilfley slimer, each 1 ft. wide and 5 ft. long. The first four trays are given a bump and jar toward the head of the tray and the other seven have a vanning motion. The first trays have the head end higher than the tail, the middle trays are level, while the last trays have the head end depressed. The catching surface on the trays is canvas.

Milling Plants of Yavapai County, Arizona.[1]—For crushing, gravity stamps are commonly used. There are also several Merralls, Tremain and Nissen single mortar stamps and a Wood steam driven stamp. They are followed by amalgamating plates and concentrating tables. Some mills use roller mills, as the Huntington, Chile, Lane and Elspass. Occasionally canvas plants have been installed.

Concentration at Old Dominion Mill, Globe, Arizona.[2]—The capacity of this mill is 500 tons per day on ore containing 3 to 5 per cent. copper in the form of sulphides and native copper, concentrating about 3½ tons into one. The ore is crushed by breakers to about 1¼ in. and delivered by belt conveyor to (1).

1. Mining box, receiving water from the mine. By distributing box to (2).
2. Two trommels with ¾-in. holes. Oversize to (3); undersize to (5).
3. One pair rolls, 36x16 in. To (4).
4. Elevator. To (5).
5. Two trommels with ⅜-in. holes. Oversize to (8); undersize to (6).
6. Two trommels with 6-mm. holes. Oversize to (11); undersize to (7).
7. Two trommels with 2.5-mm. holes. Oversize to (12); undersize to (18).
8. One double one-compartment bull jig. Concentrates to (37); hutches to (4); tailings to (9).
9. Dewatering box with shovel wheel. Coarse material to (10); overflow to (32).
10. One pair rolls, 36x16 in. To (4).
11. One jig, two-compartment. Coarse concentrates to (37); hutches to (13); tailings to (13).
12. Two jigs, two-compartment. Products like (11).
13. One dewatering box. Spigot to (14); overflow to (32).
14. One 6-ft. Chile mill with 16-mesh slotted screens. To (15).

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[1] *Mining Reporter*, 1906, LIV, p. 435.
[2] H. J. Baron, *Mining Reporter*, 1906, LIV, p. 334.

15. Two classifiers with two spigots each. 1st spigot to three Wilfley tables (17); 2d to two Wilfley tables (17); overflow to (16).
16. One spitzkasten. Spigot to three Wilfley tables (17); overflow to (28).
17. Eight Wilfley tables. Concentrates to (36); middlings to (28); tailings to (39); muddy water to (28).
18. Two hydraulic classifiers each with two spigots. Spigots separately to (21); overflow to (19).
19. Two spitzkasten in series. Spigots divided between Wilfley tables (27); overflow to (20).
20. Three pulp thickeners holding 10,000 gal. each. Spigots to nine vanners (31); overflow is wash water for tables and vanners.
21. Four jigs, two two-compartment and two three-compartment. Concentrates to (37). tailings to (22).
22. One dewatering box. Spigot to (23); overflow to (32).
23. One 6-ft. Chile mill with 1.5-mm. screen. To (24).
24. Two two-spigot classifiers. 1st spigot to three Wilfley tables (26); 2d spigot to two Wilfley tables (26); overflow to (25).
25. One spitzkasten. Spigot to four Wilfley tables (26); overflow to (28).
26. Nine Wilfley tables. Products like (17) except that the concentrates and middlings from five tables are elevated by two Frenier pumps.
27. Nine Wilfley tables. Concentrates to (37); middlings to (22); tailings to (39); muddy water to (32).
28. One elevator. To (29).
29. One dewatering box. Spigot to (23); overflow to (30).
30. Three pulp thickeners holding 10,000 gallons each. Spigots to nineteen vanners (31); overflow to (34).
31. Twenty-eight 6-ft. Frue vanners. Concentrates to (36); tailings to (39).
32. Six slime settling tanks. Settlings to (30); overflow clear water to (33).
33. Receiving tank. By pump to storage tank for use in jigs and classifiers; overflow, if any, to waste.
34. Receiving tank. By centrifugal pump to (35).
35. Slime settling bins. Settlings to bins for briquet plant; overflow to (33).
36. Elevator. To (37).
37. Four concentrates bins in parallel. Overflow to (38).
38. Two overflow bins. Overflow is clear water for tables and vanners.
39. Automatic sampler. Tailings to waste.

Milling in Arizona.[1]—The Longfellow concentrator of the Arizona Copper Company was described in THE MINERAL INDUSTRY, Vol. XIV. A new mill for treating ores from the Humboldt mine is being built at Morenci and has the following flow sheet: The ore is delivered from cars into four 250-ton steel tanks and goes to (1).

1. Two shaking grizzlies. Oversize to (2); undersize to (3).
2. Two Blake breakers, 18x36 in. To (3).
3. Two grizzlies. Oversize to (4); undersize to (5); smelting ore picked out.
4. Two pairs of rolls. To (5).
5. Inclined belt conveyor 318 ft. long. To (6).
6. Two 50-ton steel tanks. By two feeders to (7).
7. Three trommels. Each trommel has $\frac{5}{8}$- and $\frac{3}{4}$-in. holes. Over $\frac{3}{4}$ to (8); $\frac{3}{4}$ to $\frac{5}{8}$ to (9); through $\frac{5}{8}$ to (10).
8. Two pairs of rolls, 16x42 in. By elevators to (7).
9. Harz jigs. Concentrates; tailings to (11).
10. Two Hancock jigs. Concentrates; middlings returned to jig; tailings to (11).
11. Four 6-ft. Huntington mills with 5-mm. screens. By elevators to (12).
12. Two Hancock jigs. Concentrates; middlings returned to jig; tailings to (13)
13. Three Huntington mills with 1.5 mm. screens. To (14).
14. Two Hancock jigs. Concentrates; middlings to (15); tailings to (16).
15. Wilfley tables. Concentrates; tailings.
16. Classifiers. Coarse to (17); fine to (18).
17. Two Huntington mills. To (18).
18. Two elevators To (19).
19. Classifiers. Coarse to (20); fine to (21).
20. Seventeen Wilfleys. Concentrates; tailings.
21. Thirty-three vanners. Concentrates; tailings.

It is proposed to treat all Wilfley and vanner products once more. The mill tailings pass through an automatic sampler into (22).

22. Classifier. Coarse to (24); fine to (23).
23. Twenty-four concrete settling tanks each 16 ft. deep and 9.5 ft. inside diameter. Spigots to (24); overflow clear water for the mill.
24. Settling reservoir 11 ft. deep and 130 ft. inside diameter, divided into four compartments holding 4,000 tons each and having filter bottoms. Clear water goes back to mill. Pulp excavated, dried and goes into the mine for filling.

Power is supplied by three 250-h.p. Stirling boilers. A 250-h.p. Nordberg cross compound engine runs the Hancock jigs, Huntington mills, elevators and recrushing rolls. Another Nordberg engine is directly connected to a 240-kw. generator for running the various motors. The lights and crane are operated by a high speed Ball engine direct connected to a 75-kw. generator. The mill will be lighted throughout by Nernst

[1] D. E. Woodbridge, *Eng. and Min. Journ.*, 1906, LXXXII, pp. 103 150.

lamps. The capacity of the mill is 700 tons per 24 hours treating sulphide ores with 3 per cent. copper.

At the Detroit mill, settling tanks of reinforced concrete are cheaper than wood. The water used at this mill is 225 gal. per ton of ore.

Concentration at Cananea, Mexico.[1]—The Greene Consolidated Copper Company has two mills treating 2400 tons daily. The ore is a lean silicious material carrying 1 to 4 per cent. copper, chiefly in the form of chalcocite. The mills save 73 to 80 per cent. of the copper and the tailings contain 1.25 per cent. copper. A complete flow sheet of this mill is given in the original article.

San Martin Mill, Rayon District, Chihuahua, Mexico.[2]—This mill treats a complex lead-silver-copper ore containing considerable argentiferous gray copper. The mill treats 40 tons in 24 hours. Rich smelting ore is picked out by hand and the remainder goes to (1).

1. Jaw breaker. To storage bin and thence by plunger feeder to (2).
2. One 4-ft. Bryan mill with 6-mesh screens. To (3).
3. Two impact screens making four products. Two coarse products to (4); two fine sizes to (5).
4. Two Bartlett tables. Concentrates; tailings to bin and thence to (6).
5. Two Wilfley tables. Concentrates; tailings to waste or to (8).
6. One 4-ft. Bryan mill with 20- or 30-mesh screens. To (7).
7. Hydraulic classifiers making two products. To (8).
8. Two Wilfley tables. Concentrates; tailings to waste.

At El Poterito Camp in Western Chihuahua, Mexico,[3] there is a 10-stamp mill for treating quartz ores containing native gold and silver minerals. About 80 to 85 per cent. of the gold is saved and 40 to 50 per cent. of the silver. The ore is dumped on (1).

1. Ore floor. Shovelled to (2).
2. Blake breaker. To (3).
3. Bins. By feeders to (4).
4. Ten stamps, 850 lb., 40-mesh wire screens, wet crushing, no amalgamation. To (5)
5. Settling tanks. Settlings shovelled to (7); overflow to (6).
6. Settling reservoirs. Settlings shovelled to (7); overflow to waste.
7. Four 5-ft. amalgamating pans. To (8).
8. Two settlers, 8 ft. diameter. Amalgam to strainer; pulp to (9).
9. Two Wilfley tables. Concentrates to smelter; tailings to (10).
10. Tail sluice with riffles. Sand wheeled to (7); tailings to waste.

Washoe Plant of the Anaconda Copper Company.[4]—The concentration plant is described in detail. See Richards' "Ore Dressing," pp. 986, 987. For recovering coke from the ashes of the reverberatory furnaces in the smelter these ashes are flushed to (1).

1. Grizzly with 1.5-in. spaces. Oversize to waste; undersize to (2).
2. One V-shaped settling tank 20 ft. long and 6 ft. wide divided in half by a transverse partition. Overflow to waste; settlings through two gates opened automatically eight times per minute to (3).
3. Two single-compartment jigs, each 24x36 in. with 9-in. overflow, 4-in. bed and a No. 4 screen. Discharge (cinders) to waste; overflow (coke) to (4).
4. Trommel with 0.25-in. holes. Oversize (coke) by elevator to bin to be used in mixing briquets for smelter; undersize (dirt and ashes) to waste.

Concentration at Butte.[5]—In concentrating Butte ores 0.5 h.p. per ton of ore is required and 7680 gal. of water are used per ton of ore. The Hancock jig requires 919 gal. of water per ton of ore while the Evans

[1] D. E. Woodbridge, *Eng. and Min. Journ.*, 1906, LXXXII, pp. 623, 965.
[2] Kirby Thomas, *Mining World*, 1906, XXV, p. 373.
[3] W. S. Hutchinson, *Eng. and Min. Journ.*, 1906, LXXXI, p. 418.
[4] L. S. Austin, Bulletin No. 10, A. I. M. E., July, 1906, p. 529.
[5] C. W. Goodale, Bulletin Colorado School of Mines, 1906, III, p. 189; *Eng. and Min. Journ.* 1906, LXXXII p. 969

jigs require 3457 gal. Costs for concentration are: Labor $0.30; power, $0.10; supplies, $0.08; insurance, management, etc., $0.02; total, $0.50 per ton.

An average analysis of Butte ores is silica, 54.5 per cent.; alumina, 10.1 per cent.; iron, 10.9; sulphur, 13.9; copper, 4.14; silver, 1.68 oz. per ton; gold, 0.0127 oz. The ore is crushed to 0.1 in. size.

Concentration of Zinc Ores.[1]—This report contains much information on the dressing of zinc ores in general and of the British Columbia plants in particular. Not so much attention is given to wet concentration of raw ore as to the treatment of zinc concentrates for the elimination of impurities. The length of the report precludes the possibility of making a short abstract but the following list will give an idea of the ground covered. General principles of concentration. Breaking and hand picking on tables and belts. Roasting for porosity. Decrepitation and screening process. Roasting for magnetic separation in general. Roasting and magnetic separation practice at Friedrichssegen, Meiern and Lohmannsfeld and proposed scheme for British Columbia ores. Results of test on 19 lots of ore from various places in British Columbia, involving the use of wet concentration and magnetic separation. Theory of magnetic separators and description of the following kinds: Monarch or Ball Norton, Sautter, Siemens and Halske, Heberli, Ferraris No. 1, Payne, Wenstrom, Buchanan, Mechernich, Siemens, Delvik-Gröndal, Conkling, Ferraris No. 2, Rowand, Cleveland-Knowles, Finney, Ferraris No. 3, Courtney, Knowles, Wetherill type E, International, and Dings. Magnetic separating plants in British Columbia. Electrostatic separation by the Blake-Morscher machine. Flotation processes of Potter, Delprat and DeBavay with results of tests on British Columbia ores. Descriptions of the following mills in British Columbia: Cork, Whitewater, Jackson, Slocan Star, Ruth, Payne, Ivanhoe, Monitor, Wakefield and Saint Eugene. Discussion of proper milling practice for British Columbia zinc ores. Results of concentration test on 20 lots of ore and concentrates from various parts of British Columbia, involving the use of wet concentration, high and low power magnets with and without roasting, and electrostatic separation.

Concentration at Rossland.[2]—The LeRoi No. 2 mill treats 50 tons in 24 hours. The ore is reduced by Blake breaker, then by a Gates breaker to ½ in. and finally by two Trent Chile mills to 30 mesh. Three Jenckes sizers in series deliver spigot product to three Wilfley tables and overflow to a canvas slime table. The ore coming to the mill contains 0.4 to 0.5 per cent. copper and the tailings average 0.3 per cent. The mill saves about 50 or 60 per cent. of the gold and about 20 per cent. of the copper. The

[1]Report of Zinc Commission, Department of Interior, Dominion of Canada, 1906.
[2]R. W. Brock, Preliminary Report on Rossland, B. C., Mining District. Geological Survey of Canada.

concentrates contain about 2.5 oz. gold per ton, 2.5 oz. silver and 2.6 per cent. copper. The Elmore oil plant which was erected to treat the Wilfley tailings is still closed down.

The Rossland Power Company's 200-ton mill uses coarse concentration on jigs and recrushes the jig tailings in Trent mills and cyanides them. The White Bear mill has 30 stamps crushing to 20 mesh. Six Wilfley tables served for concentration and the Wilfley tailings went to an Elmore oil plant. The LeRoi mill treats 45 tons in 24 hours. The ore is crushed by Blake breaker and sized by a trommel with four sizes of holes. The coarser sizes are jigged on Harz jigs which make concentrates and tailings which are recrushed and sent back into the system. The undersize of the finest screen is classified and the spigot products are treated on Harz jigs while the overflow is divided among Wilfley tables. The Harz jig tailings are reground in Huntington mills and sent back to the classifier.

Concentration of Silver-Lead Ores.[1]—The mill of the Block 10 Company at Broken Hill, N. S. W., is described. It treats 600 tons of lead and zinc sulphide ore in 24 hours according to the following scheme:

The ore from the mine cars is dumped on (1).

1. Two grizzlies. Oversize to (3); undersize to (2).
2. Ore bin. To (6).
3. Ore bin. To (4).
4. Two Austin breakers, to 1¼ in. To (5).
5. Ore bins. To (6).
6. Traveling skip. To (7).
7. Four mill bins. By four feeders to (8).
8. Four shaking screens. Oversize to (9); undersize to (11).
9. Four No. 1 rolls. To (10).
10. Four trommels, 2½ mm. Oversize by four raff wheels to (9); undersize to (11).
11. Four No. 1 hydraulic classifiers. Spigot to (12); overflow to (20).
12. Four No. 1 jigs, four-compartment double jigs. 1st and 2d hutch to (43); 3d and 4th to (13); tailings to (18).
13. Four elevators. To (14).
14. Four shaking screens. Oversize to (15); undersize to (16).
15. Four ball mills. To (16).
16. Four No. 2 hydraulic classifiers. Spigot to (17); overflow to (22).
17. Four No. 2 jigs. Four-compartment double jigs. 1st and 2d hutch to (43); 3d and 4th to (13); tailings to (19).
18. Two shaking screens. Oversize to (46); undersize to (21).
19. Two shaking screens. Oversize to (45); undersize to (21).
20. Four No. 3 hydraulic classifiers. Spigot to (21); overflow to (22).
21. Four Wilfley tables. Concentrates to (43); middlings to (22); tailings to (46).
22. Slime thickening box. Spigots by two pumps to (23); overflow by pump to (33).
23. Distributing box. To (24).
24. Four spitzkasten with four compartments each. Sixteen spigots to (25), overflow to (35).
25. Sixteen Warren vanners. Concentrates to (43); middlings to (26); tailings to (47).
26. Two spitzkasten with four compartments each. Eight spigots to (27); overflow to (37).
27. Eight Warren vanners. Concentrates to (43); middlings to (28); tailings to (47).
28. Four V-settlers. Spigots to (29); overflow to (38) and to (39).
29. Four spitzkasten with three compartments each. Spigots to (30); overflow to (40).
30. Ten Warren vanners. Concentrates to (31); overflow to (32).
31. Concentrates bin. To smelter.
32. Tailings bin. Tailings to dump; overflow to (40).
33. Four settling tanks. Settled slime to dump; overflow by pump to (34).
34. Circular settling tank. Settled slime to (33); overflow to two water supply tanks for mill.
35. One V-settler Settled slime to (22); overflow by pump to (36).
36. One circular vanner water service tank. Settlings to (33); overflow clear water for mill.
37. One hopper settling box, also receives overflow from elevator boots. Settlings to (22); overflow to (28).
38. One baffle trough. Settled slime to dump; overflow to water service tank for mill.
39. One pump To (33).
40. One pump. To (41).
41. Four settling tanks. Settled slime by distributing box to two of the spitzkasten (29); overflow to (42).
42. Three settling tanks. Settled slime to dump; overflow to (39).
43. Concentrates bin. Concentrates to smelter; overflow to (44).
44. One hopper settling box. Settlings to (22); overflow to (28).
45. Middlings bin. Middlings to middlings dump; overflow to (44).
46. Tailings bin. Tailings to tailings dump; overflow to (44).
47. Vanner tailings bin. Tailings to vanner tailings dump; overflow to (44).

[1] V. F. Stanley Low, *Trans.* Aust. Inst. Min. Eng., 1906, XI, p. 164. Abstracted in *Eng. and Min. Journ.*, 1906, LXXXI, p. 349.

The Odling circular vanner is being tried with promising results. Some results of the Warren vanner work are given in the accompanying table:

RESULTS OF WARREN VANNER.

Size.	Feed.		Tailings.	
	Weight.	Lead Assay.	Weight.	Lead Assay.
On 20 mesh	....	....	0.5 per cent.	4.1 per cent.
20–40 mesh	1.4 per cent.	6.2 per cent.	3.3 per cent.	4.6 per cent.
40–60 mesh	1.6 per cent.	6.2 per cent.	2.7 per cent.	4.6 per cent.
60–80 mesh	6.4 per cent.	5.8 per cent.	11.9 per cent.	3.8 per cent.
80–100 mesh	1.4 per cent.	4.8 per cent.	2.0 per cent.	2.7 per cent.
Through 100 mesh	89.2 per cent.	15.0 per cent.	79.6 per cent.	6.5 per cent.

The slimes settling tanks have 6890 sq. ft. of area or 18,920 cu. ft. of volume. In addition, there is 1055 sq. ft. of surface area of circulating tanks or 14,800 cu. ft. of volume. The water in circulation is 70,000 gal. per hour. The loss of water is 110 gal. per ton of ore milled.

Very full details are given of the electric power plant. Three Parsons improved parallel flow compound turbines of 240 h.p. each are direct connected to three 160-watt generators at 500 volts. In the works are 39 motors ranging from 1 to 40 h.p. each. The accompanying table shows the estimates made in designing the mill and the actual measurements after the mill was running.

POWER REQUIREMENTS OF BLOCK 10 MILL.

Machine	H. P. Estimated.	H. P. Actual.	H. P. Total Actual.
2 Austin breakers	each 40	each 25	50
4 feed rolls 4 pairs Cornish rolls 4 shaking screens 8 parallel trommels	total 120	....	120
2 lines counter shafting	....	each 3	6
4 coarse jigs	each 2	each 1½	6
4 fine jigs	each 1½	each 1	4
4 Wilfley tables	each ¾	each ½	2
4 No. 3 Krupp ball mills	each 10	each 9.5	38
24 Warren vanners and 2 lines counter shafting	total 15	....	12
5 centrifugal circulating pumps	total 36	....	60.5
3 centrifugal slime pumps	total 10	....	19
Vanner extension centrifugal pump	....	....	16
10 vanners and counter shafting in vanner extension	....	....	5.5
Tangye return pump	....	....	4
Total			349.0

In addition motors in the mine and shops require 123 h.p., making a grand total of 472 h.p. used.

Milling at Fredericktown, Mo.[1]—The mill of the North American Lead

[1] R. B. Brinsmade, *Mines and Minerals*, 1906, XXVII, p. 149.

Company at Fredericktown, Mo., treats ore containing galena with sulphides of copper, nickel and cobalt in a limestone gangue. From the mine cars the ore is dumped on (1).

1. Grizzly, 8 ft. long, with 3-in. spaces. Oversize has waste picked out and goes to (2); undersize to (4).
2. One No. 5 D Gates breaker. To (3).
3. Two No. 3 Gates breakers. To (4).
4. One bin with 500 tons capacity. To (5).
5. Bucket elevator. To (6).
6. Two No. 1 trommels, 3x12 ft., with ½-in. round punched holes. Oversize to (7); undersize to (8).
7. One pair rolls. To (5).
8. Two No. 2 trommels with 10-mm. holes. Oversize to (13); undersize to (9).
9. No. 3 trommels with 7-mm. holes. Oversize to (14); undersize to (10).
10. No. 4 trommels with 3-mm. holes. Oversize to (15); undersize to (11).
11. Hydraulic classifier with two compartments. 1st spigot to (16); 2nd to (16); overflow to (12).
12. One V-settler. Spigot to (20); overflow to waste.
13. Eight No. 1 Harz jigs with 3 compartments each 24x36 in. 1st discharge and hutch concentrates; 2nd to (17); 3rd to (18); tailings to (23).
14. Six No. 2 Harz jigs like No. 1. Products like (13).
15. Eight No. 3 Harz jigs, like No. 1. Products like (13).
16. Two No. 4 Harz jigs. Concentrates; tailings to (23).
17. Two No. 5 Harz jigs. Concentrates; tailings to (23).
18. One Chile mill with 3-mm. slot punched screen. To (19).
19. Four No. 6 Harz jigs. Concentrates; tailings to (23).
20. Overstrom and Wilfley tables. Concentrates; tailings to (23); middlings and mud to (21).
21. One V-settler. Spigots to (22); overflow to waste.
22. Overstrom and Wilfley tables. Concentrates; tailings to (23).
23. Tailings tank. Settlings spouted to railroad cars for use in making concrete; overflow to settling pond from which clear water is pumped back to the mill reservoir.

Power is supplied by three 250 h.p. Sterling boilers under 150 ℔. pressure and a tandem compound Corliss engine.

The Madison Lead and Land Company has a new mill for treating galena in a clay matrix. The ore is dumped from cars into (1).

1. Ore bins. To (2).
2. Fritsch-Blake jaw breaker. To (3).
3. One pair Fritsch rolls 16x36 in. To (4).
4. Bucket elevator. To (5).
5. Two trommels with 10-mm. holes. Oversize to (6); undersize to (7).
6. One pair Fritsch rolls, 14x30 in. To (4).
7. Hancock cone classifier. Spigot to (8); overflow to (10).
8. Hancock jig with five hutches. First three hutches are lead concentrates; 4th fed back to jig; 5th to (9); coarse tailings to waste; fine tailings to (10); overflow water to waste.
9. One pair Fritsch rolls, 14x30 in. By centrifugal pump to (10).
10. V-settlers making three products to (11).
11. Three double deck convex round tables. Concentrates; middlings to (12); tailings to waste.
12. Five Overstrom tables. Concentrates; tailings.

The Hancock jig has screens 30 in. wide, treats 600 tons and consumes 5 h.p. The beds of Nos. 1 and 2 compartments are galena pebbles, while on Nos. 3, 4 and 5, cubes cut from ¼ to ½ in. steel flats are used. The Nos. 1, 2 and 3 hutch products contain 81, 80 and 78 per cent. lead respectively. No. 4 contains 40 per cent. and No. 5 contains 20 per cent.

Milling in Wisconsin.[1]—The scheme of a typical mill in this district is shown in Fig. 9. It is the custom in this district to use 15x7 in. Blake breakers followed by slow running Cornish rolls. One roll runs between flanges on each end of the other roll. Some mills have only one jig which has eight cells, and in mills where the ore carries considerable galena, a separate lead jig of three or four compartments is installed. Tables are used only where much lead is present. A mill with an eight-cell jig (capacity 40 tons of 20-per cent. ore per 10 hours) requires 40 h.p. and costs $6500 installed. A mill with two seven-cell jigs, 60 tons capacity, requires

[1] *Eng. and Min. Journ.*, 1906, LXXXII, p. 152.

60 h.p. and costs $6800. A mill with two seven-cell jigs and two tables, 65–70 tons capacity, requires 65-70 h.p. and costs $7000. Three men are required: one feeder ($1.75 per day), one jig man ($2.50 to $3 per day), and one wheeler ($1.75 to $2.50 per day). The zinc concentrates vary according to the iron sulphide present in the ore. They have about 2 per cent. lime and from a trace up to 3 per cent. lead. The lead concentrates are practically pure lead sulphide.

The iron sulphide is removed from the zinc concentrates, if necessary, by roasting and magnetic treatment. The Dings, Trego and Galena

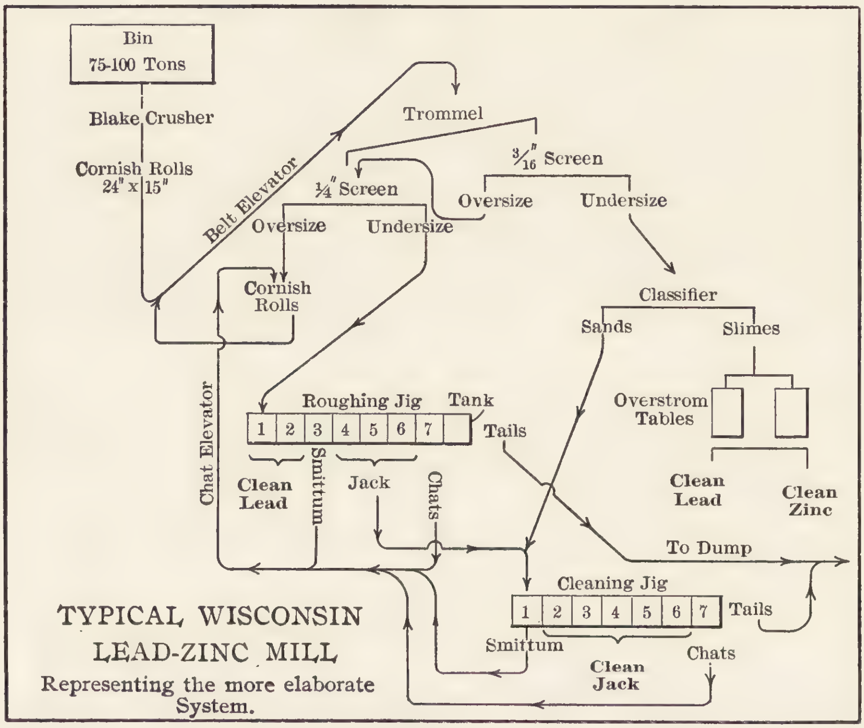

FIG. 9.

roasters are used. The first two give a quick high roast, while the last gives a slow low roast. The Dings works on the principle of the Stetefeldt shelf kiln. The Trego has a circular revolving hearth on which the ore is stirred by rabbles and treats the ore in 20 minutes. The Galena is a revolving cylinder 22 ft. long and 52 in. diameter and treats the ore for two and one-half to three hours. It is the most satisfactory. The aim in roasting is to drive off about one-third of the sulphur. After roasting, the ore is cooled by water spray, sized in a trommel having $\frac{1}{4}$ and $\frac{1}{8}$ in. holes. Each size is treated separately by either a Cleveland-Knowles or

a Dings magnetic separator. The final zinc product averages 59 to 60 per cent. zinc and about 3 per cent. iron; the middlings product averages 7 to 10 per cent. zinc and the tailings carry about 5 per cent. zinc. About 80 per cent. of the zinc is saved in the magnetic treatment and the percentage of lime is reduced. The cost of the magnetic treatment is from $1 to $2 per ton of finished product.

In the plant of the Joplin Separating Company at Galena, Ill.,[1] the ore is wheeled in barrows to (1).

1. Grizzly with 1½-in. spaces. Oversize sledged to (2); undersize to (2).
2. One 10-in. belt elevator. To (3).
3. Trommel with ¼-in. holes. Oversize to (4); undersize to (5).
4. One pair 24-in. rolls. To (2).
5. Impact screen with 20-mesh wire cloth. Oversize to (6); undersize to (7).
6. One seven-compartment jig with 30x36-in. screens. First and second compartments are lead concentrates; balance are pyrites-blende concentrates to (8); tailings to waste.
7. Two Wilfley tables. Lead concentrates; pyrite concentrates; blende concentrates; tailings to waste.
8. Elevator to four-ton hopper. By feeder to (9).
9. Roasting cylinder 32.5 ft. long and 5 ft. diameter, having capacity of 40 tons in 24 hours. Here one atom of sulphur is burned off from pyrite in 2½ or 3 hours. By scraping conveyor and wheelbarrow to cooling floor and thence by wheelbarrow to (10).
10. Belt elevator. To either of (11).
11. Two trommels, one having ¼ and ⅛ in. holes, and the other having ⅛ in. holes. Either may be used. Oversize to (12); undersizes separately to (13).
12. One pair of 10x12-in. rolls. To (10).
13. Two Cleveland-Knowles magnetic separators. Zinc concentrates; middlings to (14); tailings to waste.
14. One pair 10x12-in. rolls. By elevator to (15).
15. Screen with 3-mm. holes. Oversize to (14); undersize to (16).
16. One Cleveland-Knowles magnetic separator. Zinc concentrates; pyrite tailings.

Power is furnished by two 80-h.p. Atlas boilers, one run at a time, and a 10-h.p. engine for the dynamo and a 50-h.p. engine for the mill. The concentrator is run by day and the roaster by night. The day shift consists of a foreman, feeder, jigman, tableman, engineer and two or three wheelers. The night shift has engineer, fireman, feeder and wheeler. The average zinc concentrates from ¼-in. size contain 60.49 per cent. of zinc and 2.11 per cent. iron; those from the ⅛-in. size contain 57.07 per cent. zinc and 2.19 per cent. iron. The magnetic treatment recovers about 85 per cent. of the zinc.

The Platte 180-ton mill is a large plant and differs from those described in the preceding abstracts by having five jigs and no table.[2] One seven-compartment jig is used as a rougher; one seven-compartment jig serves as a sand jig for retreating coarse middling hutches of the rougher; one seven-compartment jig serves as a cleaner jig for the finer middling hutches of the rougher; one five-compartment chat jig treats the recrushed middlings of the sand and cleaner jigs; one four-compartment jig is used as a lead jig to retreat the lead hutches of the rougher, cleaner and sand jigs. The cost of this mill was $9925.82.

One mill in this district uses a picking belt for removing waste.

Magnetic Separation of Zinc Ore in Wisconsin.[3]—In the Wisconsin field the zinc ores are milled by the Joplin jigging system. The jig concentrates contain from 20 to 75 per cent. marcasite which is removed by roast-

[1] *Eng. and Min. Journ.*, 1906, LXXXII, p. 481.
[2] G. S. Brooks, *Eng. and Min. Journ.*, 1906, LXXXI, p. 1140.
[3] H. A. Wheeler, *Mines and Minerals*, 1906, XXVI, p. 368.

ing and magnets. The common form of roasting furnace is a revolving cylinder 25 ft. long and 5 ft. diameter. This treats 20 to 30 tons in 24 hours with a coal consumption of $1\frac{1}{2}$ to 2 tons. The Cleveland-Knowles magnetic separator which is commonly used raises the zinc contents of the concentrates up to 54-58 per cent. but the iron tailings contain 15 to 25 per cent. of the zinc.

The scheme of milling at the Enterprise Mine[1], Platteville, Wis. is similar to the typical mill previously described. In the magnetic treatment of the zinc concentrates the material is elevated to (1).

1. Galena roaster. Product by screw conveyor and elevator to (2).
2. Trommel with $\frac{1}{4}$- and $\frac{1}{8}$-in. holes. Over $\frac{1}{4}$ in. to (3); through $\frac{1}{4}$ on $\frac{1}{8}$ to (4); through $\frac{1}{8}$ to (4).
3. One pair of 8x12-in. rolls. By elevator to (1).
4. Two bins holding five tons each. Two sizes alternately to (5).
5. One Cleveland-Knowles magnetic separator. The voltage is 125 and the amperage on the first magnet is three; and five on the second. Zinc concentrates; middlings returned to jigs; tailings to waste.

The capacity of the roasting plant is 15 tons per 24 hours and it produces 2500 pounds of middling product and 11 tons of cleaned zinc concentrates. The crude concentrates fed to the roaster assay 30 to 45 per cent. zinc, 10 to 17 per cent. iron, 0.3 per cent. lead and 2.5 per cent. lime. The final zinc concentrates contain 59 per cent. zinc, 1.9 per cent. iron, and 1.8 to 2 per cent. lime. The middlings contain 7 to 10 per cent. zinc and the tailings 4.5 per cent. iron. The roaster is run in three shifts and requires three tons of coal per 24 hours. The cost exclusive of power is: Three men at \$2, 6; three men at \$1.50, 4.50; three tons coal at \$3.50, 11.50; total, \$22. Cost per ton, \$1.47. This does not include power, repairs, renewals, supplies, general expense, etc.

Milling at Lake Superior.[2]—The large Lake Superior copper mills require about 30 tons of water per ton of ore stamped. The Quincy has an Allis triple expansion pumping engine throwing upward of 16,000,000 gal. per 24 hours and a Worthington pump of 12,000,000 gal.; total, 28,000,000 gal. The Isle Royale mill has three steam stamps each crushing 550 tons per day. The blow is struck at a velocity of 25 ft. per second and the total weight of the stroke is from three to four tons. At the Quincy mill the chilled iron stamp shoes weigh 700 ℔. and wear to 400 ℔. in eight days when they are discarded. The dies wear from 800 ℔. down to 200 or 250 ℔. in six to eight months.

Regrinding at the Calumet & Hecla mills has been done in Huntington, Heberli, and Chile mills of both the Allis and Monadnock types. The rollers of the Chile mill have a crushing weight of 6000 ℔. and make 25 to 30 r.p.m., requiring 25 to 30 h.p. The cost of regrinding by Chile mills is as high as 35c. per ton in this plant. A huge Huntington mill has reduced the cost to as low as 10c. per ton.

The Calumet & Hecla tailings on conglomerate rock run about 0.7 per

[1]*Eng. and Min. Journ.*, 1906, LXXXII, p. 444.
[2]A. S. Atkinson, *Mines and Minerals*, 1906 XXVI, p. 346.

cent. copper; on amygdaloid rock containing 1 to 1½ per cent. copper the tailings run as low as 0.25 to 0.35 per cent. Quincy (amygdaloid) rock contains 1 per cent. copper and the tailings 0.26 per cent. Champion (amygdaloid) rock runs 1.5 per cent copper and the tailings 0.30 to 0.35 per cent.

The Calumet mill[1] has 11 steam stamps and the Hecla has 17. Each stamp treats 350 to 400 tons of conglomerate rock in 24 hours. The special features are: Woodbury mortar jig for steam stamp, Woodbury jigs with head plungers, Evans-Rawlings round tables coupled with Wilfleys, five sand wheels (Calumet 40 ft. and 50 ft. lift. Hecla 40, 50 and 65 ft. lift).

Graded Crushing at Lake Superior.[2]—The graded crushing plant at the Champion mill uses rock breakers, rolls, trommels down to ¼ in., classifiers, jigs and Wilfley tables. Huntington mills are employed for regrinding. The Nordberg steeple compound stamps with cylinders 15-24x 32 in. with steam at 150-lb. pressure crush 700 tons each per day. The old stamps were 20x24 and crushed 500 tons of rock per day with steam at 110-lb. pressure. Compounding has increased the tons of rock stamped per ton of coal 40 per cent. In the compound stamps steam is supplied to the low pressure cylinder only on the down stroke. On the up stroke both ends of the low pressure cylinder are connected to the receiver.

The Experimental Plant at the Champion Mill[3] has been dismantled. Its lack of success was due to difficulty in crushing rock sufficiently to release the fine particles of copper and in getting a character of material from which a proper middling product could be made for regrinding. The tailings losses under best conditions averaged slightly higher than the average from the stamps.

Crushing Lake Superior Iron Ore.[4]—Gates breakers No. 7½ or 8 are used. Breakers are placed below the car level and the product elevated or else the breaker is set up on a high foundation. The scheme of one plant, having a capacity of several thousand tons per day, is as follows:

Ore from railroad cars is dumped to (1).

1. No. 9 Gates breaker, crushing to 4 in. By elevator to (2).
2. Two trommels, 4x10 ft., with 1¼ in. holes. Oversize by belt conveyor to (3); undersize by belt conveyor to loading bin.
3. Storage bins supplying five No. 5 Gates breakers. To (4).
4. Five elevators. To (5).
5. Five trommels, 40 in. x 10 ft., with 1¼ in. holes. Oversize to (6); undersize to loading pocket.
6. Five No. 3 Gates breakers. To (4).

Individual electric motors are the best power for such a plant.

Washing of Mesabi Iron Ores at Canisteo, Minnesota.[5]—The iron ore is dumped from the cars into a bin and goes thence to (1).

1. Conical trommel with 2-in. holes. 14 ft. long, 2.5 ft. diameter at small end. It rotates on four rollers. Oversize to (2); undersize to (3).
2. Picking belt 28 ft. 8 in. long. Taconite and rock picked out. Residue to market.
3. Log washer. Iron ore concentrates to (4); tailings to waste.
4. Double trommel 12 ft. long. Oversize to market; undersize to waste(?)

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[1]A. L. Carnahan, *Mining World*, 1906, XXV, p. 515.
[2]J. Parke Channing, *Min. and Sci. Press*, 1906, XCII, p. 198.
[3]Report of Copper Range Company for 1906.
[4]R. E. Newhouse, *Iron Trade Review*, Jan. 25, 1906, XXXIX, p. 25
[5]*Iron Trade Review*, December, 1906, XXXIX, p. 27.

The capacity of this washer is 1000 tons per day. The iron ore has about 24 to 30 per cent. sand eliminated in the washing and the washed ore runs 50 to 58 per cent. iron.

Milling at Goldfield, Nevada.[1]—The Kinkead mill has three Kinkead mills and passes the pulp over amalgamated plates and three vanners. The tails are cyanided. The Western custom mill has two Vezin samplers, a five-stamp battery crushing to 30-mesh and a classifier which sends coarse material to a Wilfley table and fine to a Frue vanner. It treats 15 tons in 24 hours and saves 70 to 80 per cent. of the values. The Combination and the American mills use stamping, amalgamation and cyaniding. The former has gravity stamps and the latter has two Nissen stamps which treat six tons each in 24 hours.

Pyrite Dressing.[2]—At the Davis pyrite mine in Rowe, Mass., the ore from the skip amounts to about 100 tons per day and goes to (1).

1. Grizzly with 1½-(?)in. spaces. Oversize by car to (2); undersize to (3).
2. Breaking floor where the ore is spread out and broken and cobbed by hand to 3-in. maximum. Coarse ore removed by forks to lump shipping bin; fine ore shovelled to fines shipping bin; waste to dump. A little copper ore is picked out. Machine breakers have been tried but they make too many fines.
3. Storage bin. In three or four hours daily this is discharged to (4).
4. Concentric trommel with ¾- and ⅜-in. holes. Over ¾-in. to (5); through ¾- on ⅜-in. to concentrating mill; through ⅜-in. to fines shipping bin.
5. McLanahan-Stone picking table 10 ft. diameter. Waste to dump; residue is "nut" ore to shipping bin.

The concentrating plant is run only in the spring, summer and autumn. The material is fed by hand to (1).

1. Concentric trommel with ¾- and $\frac{5}{16}$-in. holes. Oversize to (2); undersize to (3).
2. Bacon-Farrell rolls. By elevator to (1).
3. Bin feeding a four-sieve Harz jig. First and second discharges and hutches are concentrates; third and fourth go back to jig; tailings to waste. A two-compartment Bacon jig is ready to be installed.

The mill is run only on day shifts and treats 1500 to 2000 tons in four or five months. The concentrates contain 44 to 46 per cent. sulphur and 1.5 per cent. copper and the tailings average 1 to 2 per cent. sulphur. Of the total ore mined 70 per cent. is shipped as lump ore.

Pyrites Concentration at Hermon, N. Y.[3]—A 250-ton plant is under construction. The ore is passed through a breaker in the rock house and sent by cars to (1).

1. Ore bins. By two 12x24-in. plunger feeders to (2).
2. Two shaking grizzlies with 1-in. spaces Oversize to (3); undersize to (4).
3. No. 5 K Gates breaker crushing to 1 in. To (4).
4. No. 4 Gates elevator. To (5).
5. Crushed ore bin holding 175 tons. By 12x24-in. plunger feeder to (6).
6. Shaking screen with 5-mm. round holes. Oversize to (7); undersize to (8).
7. One pair of rolls, 36x15-in., set about ½ in. apart. To (8).
8. One belt elevator, with buckets 12x15½ in. To (9).
9. Two trommels with 5-mm. round holes. Oversize to (10); undersize to (11).
10. One pair of 36x15-in. rigid rolls. To (8).
11. One Richards annular cone classifier with one spigot. Spigot (200 tons in 24 hours) to (12); overflow by centrifugal pump to (?).
12. One 25-ft. Hancock jig. First three hutches are pyrite concentrates to (16); 4th to (8); 5th to (13); 6th is tailings to waste.
13. One pair 36x15-in. rigid rolls. By 8-in. elevator to (14).
14. One Richards 13-in. annular classifier with one spigot. Spigot to (15); overflow by centrifugal pump to (?).
15. One 16-ft. Hancock jig. Concentrates to (16); tailings to waste.
16. One 10-in. elevator. To (17).
17. Five concentrates bins with a total capacity of 1200 tons.
18. Storage bins for fines (under 40-mesh). These have plugs at the bottom and in front for draining off excess water. Coils of steam pipe serve to dry the concentrates to prevent freezing in winter.

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[1] C. T. Rice, *Eng. and Min. Journ.*, 1906, LXXXII, p. 339.
[2] J. J. Rutledge, *Eng. and Min. Journ.*, 1906, LXXXII, pp. 673, 772.
[3] *Eng. and Min. Journ.*, 1906, LXXXI, p. 1192.

Concentrating Auriferous Antimony Ore.[1]—The ore from the mine of the Dominion Antimony Company at West Gore, Nova Scotia, contains auriferous stibnite and silicious gangue. Several concentration tests have been made on this ore. Wet crushing was found to cause too great a loss from slimes carried away in the water. Dry graded crushing and careful sizing to 24 or 30 mesh followed by wet concentration seems to give good results. In one test 6¼ tons were crushed with rolls to 24 mesh and treated on a Wilfley table which yielded concentrates, middlings returned to the table, sand tailings to be leached for gold, and slimes. Weights and assays are given in the accompanying table which shows a saving of 90 per cent. of the antimony.

STIBNITE CONCENTRATION AT WEST GORE, NOVA SCOTIA.

	Weight.	Per cent. Antimony.	Total Antimony.	Gold per Ton.	Total Gold.
Original ore......	6¼ tons	6.2	775 lb.	$7	$43.75
Concentrates.....	1,400 lb.	50.5	700 lb.	24	16.80
Sand............	10,665 lb.	1.0	100 lb.	4	21.34
Slimes...........	444 lb.	5.0	20 lb.	18	3.99

Concentration of Corundum.[2]—The mines of the Canada Corundum Company in Raglan township, Renfrew county, Ontario, contain corundum in a feldspar gangue. The ore is brought in cars to (1).

1. Flat bottomed bin holding 400 tons. By chute to (2).
2. One Farrell-Blake breaker 15x24 in., running at 250 r.p.m., crushing to 2½ in. By 18-in. Robins belt conveyor running at 300 ft. per minute and sloping upward 20 deg. to (3).
3. Two Farrell-Blake breakers 6x20 in. and one Gates A breaker all crushing to ¾ in. To (4).
4. Bin holding 400 tons. By Challenge feeder to (5).
5. Screen. Oversize to (6); undersize to (7).
6. Two pairs of coarse rolls 14x40 in. To (7).
7. Two trommels, 13x3 ft., running at 20 r.p.m., sloping 1 in. per foot and having 4-mm. holes. Oversize to (8); undersize to (9).
8. Two pairs of rolls 14x40 in. To (9).
9. Bucket elevator running at 350 ft. per minute. To (10).
10. Two trommels, each sectional, with 6 ft. of 4-mm. holes, 4 ft. of 6-mm. holes and 1½ ft. of 8-mm. holes. Oversize to (8); undersize of 4-mm. to (11); undersize of 6-mm. to (15); undersize of 8-mm. to (16).
11. Two trommels, each sectional, with 6 ft. of 2-mm. holes and 5 ft. of 2½-mm. holes. Oversize to (17); undersize of 2-mm. to (12); undersize of 2½-mm. to (20).
12. Two trommels with 1½-mm. holes. Oversize to (20); undersize to (13).
13. Two trommels with 1-mm. holes. Oversize to (20); undersize to (14).
14. V-box classifier. Spigot to (20); overflow to waste.
15. Two double three-compartment iron Harz jigs with screens 24x36 in. First hutch to (19); 2nd and 3rd to (18); tailings to waste.
16. Two double two-compartment wooden Harz jigs. First hutch to (19); 2nd to (18); tailings to waste.
17. One double three-compartment iron Harz jig. Products like (15).
18. One pair Gates rolls, 14x24 in. To (9).
19. One pair Colorado rolls, 6x30 in. To (22).
20. Twenty Overstrom and four Wilfley tables. Concentrates to (22); middlings to (21); tailings to waste.
21. Six Overstrom tables. Concentrates to (22); tailings to waste.
22. One bucket elevator. To (23).
23. One trommel with 12-mesh screen. Oversize to (19); undersize to (24).
24. Five storage and drainage tanks. By conveyor to (25).
25. One double-deck drier made of iron pipes 1¼ in. in diameter, heated by exhaust and live steam. A 4-mesh screen is on top and as the stuff dries it drops through and is carried by conveyor and elevator to (26).
26. Two magnetic concentrators, one of the cone and one of the drum type. Magnetic material to dump for further treatment; non-magnetic to (27).
27. Three screens (splitters) in series, 8-, 30- and 80-mesh. To (28).
28. Three sets of grading screens, dividing the corundum into the following sizes: 8, 10, 12, 14, 16, 20, 24, 30, 36, 46, 54, 60, 70, 80, 90, 100, 120, 150, 180, and 200 mesh. Steel wire cloth is used as fine as 30-mesh and silk screen below 30-mesh. Products to (29).
29. Storage bins. To (30) or (31).

[1]Report of Department of Mines of Nova Scotia for 1905.
[2]D. G. Kerr, *Trans.* Inst. Min. Eng., 1905, XXX, p. 143. Abstracted in *Can. Min. Rev.*, 1906, XXVII, p. 152.

30. Hooper air jig, used on concentrates from 24 to 70 mesh. First product (magnetic and pyrite) to storage dump; 2nd (corundum) to (33); 3rd (middlings) is rerun; 4th is tailings to waste.

31. Five Wilfley tables. First product (magnetic and pyrite) to storage dump; 2nd (corundum) to (32); 3rd (middlings) rerun; 4th to waste.

32. Conveyor to second deck of drier (25) and thence by conveyor to (33).

33. Elevator. To finishing magnetic separator and final magnetic separator making clean corundum which is sized by finishing splitters and graders like (27) and (28).

Power is supplied by three return tubular boilers burning 25 to 30 cords of wood per 24 hours, and by three engines. One Corliss engine of 125 h.p. runs the crushers and a Roots pump with a capacity of 1,000,000 gal. in 24 hours against a head of 60 ft. One auxiliary engine of 20 h.p. runs the machine shop and sometimes the dynamo. One Corliss engine of 125 h.p. runs the concentrating machinery and the dynamo for 220 lights of 16-candle power.

Analyses of products are as follows: Ore fed to mill, 10 per cent. corundum; jig concentrates, 50 per cent. corundum; jig middlings, 40 per cent. corundum; jig tailings, 3 per cent. corundum; table concentrates, 60 per cent. corundum; table tailings, 2 per cent. corundum; average magnet tailings 5 per cent. corundum; rewash-table tailings, 5 per cent. corundum; Hooper jig tailings, 5 per cent. corundum; final concentrates, 90-95 per cent. corundum; total mill tailings, 5 per cent. corundum; The original ore contains 6 to 7 per cent. magnetic iron and the final concentrates contain 1 to 2½ per cent. iron in the combined form. The total cost has not been below $40 per ton of final concentrates but with a well equipped mill treating 150 tons per 24 hours of an ore containing 10 to 12 per cent. corundum the cost should not exceed $30 to $35.

Corundum Washing.[1]—The hydraulic process used for washing rubies in North Carolina consists of a sluice box, sieve box and final cleaning by rocker and hand work. The methods of cleaning ordinary corundum are divided into the muller process (See Richards' "Ore Dressing," p. 1078) and the jig and table process exemplified by the mill of the Canada Corundum Company.

Concentration of Chrome Ore.[2]—At the mine of the Black Lake Chrome and Asbestos Company, Quebec, the mine ore is sorted into No. 1 grade, over 47 per cent. chromic oxide, No. 2 grade, 40 to 47 per cent. and No. 3 or milling material under 40 per cent. This last is crushed in a jaw breaker and fed to stamps having 22-mesh screens. The stamp pulp is concentrated on Wilfley tables. The best of the tailings go to another table which yields No. 2 concentrate containing 40 to 43 per cent. chromic oxide.

Concentration of Tungsten Ores.[3]—In a memoir on tungsten ores, the methods of treatment are described. The ores of San Juan county, Colorado, contain hübnerite coarsely crystallized. The treatment at the Great Western Exploration mill is as follows:

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[1]Bulletin No. 269, U. S. Geological Survey, pp. 99, 160.

[2]W. H. Edwards, *Eng. and Min. Journ.*, 1906, LXXXII, p. 584; from Canadian Mining Institute.

[3]H. R. Van Wagenen, *Mining Magazine*, 1906, XIII, p. 327. From Bulletin, Colorado School of Mines, 1906, III, p. 138.

1. Grizzly with 2-in. spaces. Oversize to (2); undersize to (3).
2. Blake breaker, 7x10 in. To (3).
3. Ten stamps weighing 1000 lb. each, crushing to 20-mesh. To (4).
4. Three classifiers in series. First spigot to (5); 2nd to (6); 3rd to (7); overflow to (8).
5. No. 1 Wilfley table. Concentrates; middlings to (9); tailings to waste; slime to (9).
6. No. 2 Wilfley table. Concentrates; middlings to (9); tailings by pump to (3); slime to (9).
7. No. 3 Wilfley table. Concentrates; middlings to (9); tailings to waste; slime to (9).
8. No. 4 Wilfley table. Products like (7).
9. Distributing tank. To (10).
10. Three slimers, much like vanners. Concentrates; middlings elevated to (3); tailings to settling tank.

The mill treats 12 tons in 12 hours and yields one ton of concentrates from 15 tons of ore.

The ores of Boulder county are generally wolframite finely disseminated and the concentration is more difficult. The scheme in the Boulder county mill is as follows:

1. Grizzly with 2-in. spaces. Oversize to (2); undersize to (3).
2. Blake breaker, 7x10 in. To (3).
3. Ten stamps weighing 750 lb. each and crushing to 12-mesh. To (4).
4. Two Wilfley tables. Concentrates; tailings by elevator to tank and thence to (5).
5. Two classifiers in series. First spigot to (6); 2nd to (7); overflow to (8).
6. One 4-ft. Frue vanner with corrugated belt. Concentrates; tailings to waste.
7. One 6-ft. Frue vanner with smooth belt. Products like (6).
8. One 6-ft. Frue vanner with smooth belt. Products like (6).

This mill treats 25 tons in 24 hours and obtains one ton of concentrates from 15 tons of ore. Details of several other mills are given.

Magnetic Concentration at Lyon Mt., N. Y.[1]—There are two mills, the old and the new. The old mill has a capacity of 1000 to 1200 tons in 24 hours while the new treats over 1200 tons. The latter is in three sections in series and treats the ore according to the following scheme:

Ore from cars is dumped to (1).

1. Crude ore bin with 700 tons capacity. To (2).
2. No. 1 grizzly with 2½-in. spaces between manganese steel bars. Oversize to (3); undersize to (6).
3. One 24x30-in. Blake breaker crushing to 5 in. To (4).
4. No. 2 grizzly with 1½-in. spaces. Oversize to (5); undersize to (6).
5. Gates breaker No. 6 K crushing to 2 or 2½ in. To (6).
6. No. 1 conveyor belt. To (7).
7. No. 3 grizzly with ¾-in. holes. Oversize to (8); undersize to (9).
8. One pair of Anaconda rolls, 40x30 in., crushing to 1 in. To (9).
9. No. 1 bucket elevator. To (10) or, if dry, to (12).
10. A vertical modified Rowand drier 40 ft. high, having horizontal T-iron baffle plates. To (11).
11. No. 2 bucket elevator. To (12).
12. Two storage bins, 450 tons capacity. By roll feeders to (13).
13. Two concentric trommels without center shaft. 16x4½ ft., ¾- and ¼-in. holes. Over ¾ and over ¼ separately to (14); under ¼ to (17).
14. Four pairs of Anaconda rolls, 40x15 in. By chutes to (15).
15. Two bucket elevators. To (16).
16. Two trommels with ¼-in. holes. Oversize to (14); undersize to (17).
17. Two bucket elevators. To (18).
18. Four bins, 350 tons capacity. To (19).
19. Four Ball-Norton double drum magnetic separators each treating 15 to 20 tons per hour. Concentrates to (25); middlings to (20); tailings to (21).
20. Two pairs of rolls, 22x16 in. To (22).
21. Two pairs of Anaconda rolls, 40x15 in. To (22).
22. Four magnetic separators like (19). Concentrates to (25); middlings to (24); tailings to (23).
23. One magnetic separator like (19). Concentrates to (25); middlings to (24); tailings to (27).
24. Bucket elevator. To (20).
25. One 20-in. belt conveyor, inclined 23½ deg. To (26).
26. Concentrates bin holding 400 tons.
27. One conveyor like (25). To (28).
28. Trommel with ⅛-in. holes. Oversize to (29); undersize to (30).
29. Coarse tailings bin holding 300 tons. Used for ballast, concrete, etc.
30. Fine tailings bin holding 300 tons. Used for locomotive sand.

The crude ore runs from 25 to 40 per cent. iron, average 32 per cent., and consists of finely disseminated magnetite. The tailings average 4 per cent. iron as magnetite and the concentrates average 63 per cent. iron.

[1]D. H. Newland, and N. V. Hansell, *Eng. and Min. Journ.*, 1906, LXXXII, p. 916, p. 981.

ANALYSES FROM LYON MT. MILL, JANUARY, 1906.

	Crude Ore.	Concentrates	Tailings.		Crude Ore.	Concentrates.	Tailings.
	Per cent.	Per cent.	Per cent.		Per cent.	Per cent.	Per cent.
Ferric oxide	31.48	60.128	4.57	Potash (K_2O)	1.438	.494	1.61
Ferrous oxide	15.81	28.850	3.60	Soda (Na_2O)	2.283	.777	2.99
Manganous oxide	.115	.107	.12	Moisture	.25	.040	.12
Silica	33.16	6.880	58.56				
Alumina	4.90	.900	10.72		99.823	99.960	98.910
Lime	4.96	.660	8.24				
Magnesia	2.10	.405	4.06	Iron total	36.50	64.72	9.70
Phosphoric acid	.043	.023	.064	Iron in magnetite	34.30	64.53	6.00
Sulphur	.027	.022	.033	Phosphorus	.019	.019	.028
Titanic acid	.427	.417	.457	Titanium	.089	.083	.096
Ferrous oxide (in gangue)	2.83	.257	4.76	Manganese	.256	.250	.274

Power is furnished by two 225-h.p., 1040 volts 3-phase current General Electric induction motors; 250 kw. are required when the mill is running full. The current for the separators is supplied by a 50-h.p. motor generator. Sixteen men are required per shift: four on crushers and rolls, three on the magnetic separators, one fireman for the drier, one motor man, one oiler, one foreman, five sampling, weighing, loading and dumping.

Concentrating Iron Sands of Lower St. Lawrence.[1]—Some experiments have been made on the recovery of magnetite from iron sands by the use of a Heberli drum separator. This has eight fixed magnets of alternate polarity arranged part way around the inner circumference of a revolving cylinder of brass. Fifty-two pounds of a sand containing 57 per cent. iron and 16.2 per cent. titanic oxide treated dry yielded 22 ℔. of concentrates with 70.46 per cent. iron and 1.91 per cent. titanic oxide. The tailings weighed 30 ℔. and assayed 45.30 per cent. iron and 23.30 per cent. titanic oxide. The capacity of the machine dry is about 300 ℔. per hour and it requires about 0.2 h.p. for the magnets and 0.2 h.p. to revolve the cylinder. When run wet the machine will do practically the same work that it does dry but the capacity is only about 100 ℔. per hour.

Decrepitation.[2]—The separation of blende from barytes cannot be accomplished by wet methods owing to the little difference in specific gravity and it becomes necessary to use special processes such as the electromagnetic, electrostatic, flotation or decrepitation methods. The last has been actually operated at Cole Camp, Benton county, Mo.

Dressing Tin Ores.[3]—At the Old Clitters mine near Gunnis Lake, East Cornwall, the ore contains tin and wolfram. The larger portion goes to (1) and a smaller portion to (5).

[1]J.F. Robertson, *Proc.* Can. Soc. Civ. Eng., Nov 30, 1905; *Can. Min. Rev.*, 1906, XXVI, p. 23; *Mining Reporter*, 1906, LIV, p. 643.
[2]*Eng. and Min. Journ.*, 1906, LXXXI, p. 1152.
[3]F Dietzsch, *Trans.* I. M. M., 1906, XV, p. 2.

1. Two grizzlies with 2-in. spaces. Oversize to (2); undersize to (3).
2. Blake-Marsden rock breaker, 15x10 in. To (3).
3. Storage hopper. By four Challenge feeders to (4).
4. Twenty stamps weighing 800 lb. each, dropping 95 times per minute and having 25-mesh gun metal woven wire screens. To (12).
5. Storage hopper. To (6).
6. Shaking screen. Oversize to (7); undersize to (8).
7. Rock breaker, 12x8 in. To (8), or to (11).
8. One pair rolls, 28x12 in. By elevator to (9).
9. Vibro screen. Oversize to (8); undersize to (10).
10. No. 6 ball mill with 30-in. screen. To (12).
11. Five stamps. To (12).
12. Three spitzlutten, two with three compartments each and one with two compartments. Spigots to (14); overflow to (13).
13. A 10-compartment spitzkasten 50 ft. long, 6 ft. wide on top and 5.5 ft. deep. Spigots to (15); overflow to (20).
14. Eight Buss swinging tables. Concentrates to (21); middlings elevated back to tables; tailings to (20).
15. Four double compartment distributing boxes. To (16).
16. Four double Lührig vanners. Concentrates to (21); middlings by centrifugal pump to (17); tailings to (20).
17. Small spitzkasten. Spigot to (18); overflow to (20).
18. Distributing box. To (19).
19. One double Lührig vanner. Concentrates to (21); tailings to (20).
20. One eight-compartment spitzkasten 40 ft. long, 6 ft. wide on top and 5.5 ft. deep. Lime is added to coagulate the slimes. Spigots to settling ponds. Overflow pumped back to water supply tank.
21. Calciner. The Buss concentrates contain only sulphides of iron, copper and arsenic and oxides of tin and wolframite and are roasted separately from the vanner concentrates which contain also iron oxide. After roasting, the former go to (24) and the latter to (22).
22. Convex Cornish buddles. Concentrates to (23); tailings of iron oxide to waste.
23. Reverberatory drier. To (24).
24. One Wetherill cross-belt magnetic separator with four fields. First product (iron oxide) to waste; 2nd product (copper and iron) to copper leaching; 3rd and 4th products (iron and wolfram) are cleaned by buddles and kieves, or kieves alone, and sold; 5th or non-magnetic (tin) product is kieved and goes to tin smelter.

There is also in the plant a No. 2 wet ball mill for crushing old rich middlings and sending them to the washing machines. At times it is used dry for recrushing the first and second Wetherill products when they are rich in tin and wolfram. Later it is proposed to use it for recrushing the Buss middlings to be retreated on the Lührig vanners.

The roasting aims to convert the iron in the sulphides of iron, arsenic and copper into magnetic oxides (Fe_3O_4 and Fe_6O_7), not into Fe_2O_3 which is weakly magnetic. The iron is thus largely eliminated as waste in the first Wetherill product.

Full tests have been made of the mill and these are given in the accompanying tables. Of the ore coming to the mill, 92.92 per cent., assaying 1.025 per cent. cassiterite and wolframite, is retained on 30-mesh; and 7.08 per cent., assaying 1.4 per cent., passes through 30-mesh. The average assay is 1.051 per cent.

The average value of tailings is calculated 0.11 per cent, and the average value of ore going to mill 1.23 per cent. The ore treated in July, 1903, was 2322 tons and the total calculated tin oxide and wolframite in this was 30.65 tons. The actual recovery was 9.29 tons of tin oxide (65 per cent. metallic tin) and 18 tons of wolframite (62.5 per cent. oxide of tungsten). This is equal to 8.96 lb. or 0.40 per cent. tin oxide per ton of ore crushed and 17.36 lb. or 0.77 per cent. of wolframite. The extraction is 88.6 per cent. All results are based on results of vanning assay which is acknowledged to give lower results than chemical analysis. Details of costs are given which total £345 4s. 2d. per month or 2s. 8.6d. per ton.

Experiment has shown that woven wire screens in the stamps are better than punched plates. Ball mills produce less slimes than stamps.

The satisfactory extraction is claimed to be largely due to the classification, and criticism is made of the lack of classification in many Cornish mills. Some difference of opinion was shown to exist as to the advantage of classification before fine concentration.

Sieve Scale. (mesh)	Pulp from 1st Compartment of Spitzlutten 39.94 Per Cent. of Total Pulp.		Pulp from 2nd Compartment of Spitzlutten 17.17 Per Cent. of Total Pulp.		Pulp from 3rd Compartment of Spitzlutten 17.04 Per Cent. of Total Pulp.		From Bottom Discharge of Spitzkasten (a) 16.79 Per Cent. of Total Pulp.	
	Per Cent. on Screen	Per Cent. Tin Oxide and Wolframite.	Per Cent. on Screen.	Per Cent. Tin Oxide and Wolframite.	Per Cent. on Screen.	Per Cent. Tin Oxide. and Wolframite.	Per Cent. on Screen.	Per Cent. Tin Oxide and Wolframite.
20–30	4.46	1.2	0.78	0.6	0.00	0.0	....	
30–40	26.64		8.12		1.61		....	
40–60	31.75		19.68		13.87	0.4	1.93	
60–80	13.51	1.6	14.39	1.0	8.06		4.98	0.2
80–100	6.75		13.59		24.84		8.38	
100–120	5.54	3.1	19.68	1.4	17.74	1.0	17.74	0.5
120–160	5.00		14.84		17.74		40.32	
Through 160	6.35	3.0	8.90	1.4	16.13	1.3	27.42	0.6
		Average, 1.82		Average, 1.19		Average, 0.955		Average, 0.51

(*a*) Goes to coarse vanners.

Sieve Scale. (mesh)	Pulp from Bottom Dicharge of Spitzkasten (b) 9.32 Per Cent. of Total Pulp.		Tailings from Buss Tables 68.82 Per Cent. of Total Ore.		Tailings from Lührig Vanners 24.13 Per Cent. of Total Ore.		Stamp Product.	
	Per Cent. on Screen.	Per Cent. Tin Oxide and Wolframite.	Per Cent. on Screen.	Per Cent. Tin Oxide and Wolframite.	Per Cent. on Screen.	Per Cent. Tin Oxide and Wolframite.	Per Cent. on Screen.	Per Cent. Tin Oxide and Wolframite.
20–30	....	....	....	....	....	....	7.17	
30–40	....	....	....	....	....	....	1.35	
40–60	....	....	16.42	0.10	1.41	0.00	19.38	
60–80	0.57		....	....	....	....	4.45	Not determined.
80–100	1.14	0.15	30.40(*c*)	0.10	6.62(*c*)	0.00	17.44	
100–120	3.42		....	....	....	....	6.39	
120–160	7.42	0.3	41.71(*d*)	0.05	35.00(*d*)	0.05	6.39	
Through 160	87.42	0.6	11.41	0.2	56.81	0.25	14.34	
		Average, 0.553		Average, 0.090		Average, 0.239	29.45	

(*b*) Goes to fine vanners. (*c*) Is through 60 on 100. (*d*) Is through 100 on 160.

Concentrating Tin Ore in Bolivia.[1]—In Bolivia tin ore is crushed by breakers, stamps, ball mills, Huntington mills and occasionally by rolls. Concentration is effected on Wilfley and other tables, the Cornish round buddles and revolving tables and on the old rectangular hand buddles.

Mount Bischoff Tin Mill in Tasmania.[2]—This plant was described in THE MINERAL INDUSTRY, Vol. XIII, p. 405.

Tin Washing in Malaya.[3]—At the Tambun mine the gravelly and clayey tin bearing ground is first passed through four puddlers 20 ft. in

[1]J. B. Minchin, *Eng. and Min. Journ.*, 1906, LXXXI, p. 810.
[2]*New Zealand Mines Record*, 1906, IX, p. 387.
[3]Ralph Stokes, *Mining World*, 1906, XXV, pp. 424, 572.

diameter. The concentrates from these are elevated to four trommels having 8, 4½, 3 and 2-mm. holes. The four undersizes are treated on four jigs. The oversize of 8-mm. is run through a rock breaker and Huntington mill and concentrated on a Wilfley table.

Other mines recover the tin by regular hydraulicking methods. The Redhills mill at Kinta uses hydraulic giants and puddlers. Coarse rock fragments are crushed by stamps and concentrated on Buss and Wilfley tables. A dredge is being installed at one deposit.

Treating Old Dumps.[1]—At Mine la Motte, Mo., the old dump contains tailings from material crushed to 8 mesh. This is now being reground in geared rolls and treated on Standard tables, at a profit of 50c. per ton.

Mill Experiences.[2]—The fact that sand launders wear rapidly especially where there is a drop or a bend is well known; likewise the efficiency of pockets at bends and drops to prevent this wear. Wear even in straight launders may be greatly reduced by the use of cross riffles capped with iron which device makes practically a sand bottom. Such a launder requires a little more slope than a smooth launder but a fall of one in twenty is sufficient to transport 8-mm. material.

The linings and rotors of centrifugal pumps used for elevating sand last only a very short time. Their life is prolonged by the use of white iron castings. Such castings cannot be machined and therefore the rotor has to be centered on the shaft by the use of wood wedges and babbitt. Such a rotor lasts three weeks whereas chilled iron has to be replaced in three days.

In centrifugal pumps there is a tendency for grit to cause wear between the shaft and casing sleeve. This may be obviated by forcing a tarry mixture composed of rosin dissolved in hot linseed oil into this space from a large grease cup. This prevents the entrance of grit.

Centrifugal pumps that have given difficulty from choking have been relieved by using smaller discharge pipes. Sections of the discharge pipe should abut tight against one another and be held by flange couplings. Long and easy bends should be used. Direct driving by motors of the enclosed type is preferable to the use of belts. To avoid the transmission of vibration from the pump to the motor the driving shaft may be made in two parts which are joined by a special flange coupling involving the use of old rubber hose to take up shock.

Coal Washing.

Coal Crusher.[3]—The Galland breaker for coke and coal consists of ordinary toothed rolls above which is a set of eight pointed bars or four bars on a side. Above the first set is still another set of eight bars. These bars

[1] *Min. and Sci. Press*, 1906, XCII, p. 232.
[2] R. D. O. Johnson, *Eng. and Min. Journ.*, 1906, LXXXI, p. 319.
[3] *Comptes Rendus Mensuels*, Société de l'Industrie Minerale, February, 1906, p. 52.

are given a horizontal reciprocating motion through cranks and connecting rods so that their action resembles that of a jaw breaker. In the upper set the bars are advancing toward each other while those in the lower set are receding and *vice versa.* These pointed bars make 250 strokes per minute and do the preliminary crushing for the rolls. This breaker is actually used in many coal plants. The chief advantage claimed is the reduced amount of dust formed.

Coal Screen.[1]—A device patented by W. O. Gunckel of Terre Haute, Ind., is a bar screen having round bars each of which is given a rotating motion back and forth for perhaps a quarter of a revolution around its axis. This device has all the advantages of shaking screens without the disadvantage of jarring or straining the mounting.

Removal of Slime from Coal.[2]—At Ewald coal washing plant there is an apparatus for removing slime which is in the form of a cylinder with conical bottom. A center revolving shaft has a horizontal sieve attached near the top and stirring arms near the bottom. Material fed below the sieve is subjected to the action of a rising current of water introduced from below. This lifts the slime to overflow while the coarse material remains behind and is discharged periodically through side and bottom doors.

Recovery of Water from Coal Washing.[3]—Washed coal after leaving the jigs is run through a disintegrator delivering to a long trommel. The water and undersize falls into the fourth compartment of a settling tank which is 30 ft. long, 9 ft. wide and 6 ft. deep. This tank has V-bottom and a screw conveyor 29 ft. long which transports all settlings to the boot of an elevator in the fourth hopper. The three vertical partitions extend down to within 1 in. of the screw; they also have several ½-in. holes about 2 ft. from their tops. The elevator for removing the settled sludge has perforated buckets for draining. Clear water is pumped from the first compartment. This apparatus will handle the waste water and sludge from 400 to 500 tons of coal in eight hours. Each ton of coal requires a ton of water. The Ramsay sludge tank is another device for accomplishing the same results. For a description see Richards' "Ore Dressing," p. 446.

Removal of Coal Dust.[4]—At the Serlo colliery, Saar district, experiments have been made on removing clay shale dust from small coal before it reaches the washing plant. The coal falls from an adjustable slot in the bottom of the hopper through a horizontal blast of air delivered through an adjustable slot 2 ft. long in a 1¼-in. iron pipe. The air is delivered under a pressure of 5 or 10 lb. per sq.in. and the best results are obtained when the air slot is 1½ in. wide.

[1]F. W. Parsons, *Eng. and Min. Journ.*, 1906, LXXXII, p. 925.
[2]*Zeit. f. B. H. u. S. im Preuss.*, 1906, LIV, p. 284.
[3]F W Parsons, *Eng. and Min. Journ.*, 1906, LXXXI, pp. 649, 1151.
[4]*Colliery Guardian* 1906, XCII, p. 1027. From *Zeit. f. B. H. u. S. im Preuss.*, 1906, LIV, p. 279.

Removal of Dust from Coal.[1]—At the coal washing plant of the Dechen mine, Saar district, the use of a fine water spray over the shaking screens did not give good results in settling the dust. The present arrangement consists of a suction fan connected by pipes to the hoppers in which the screens are placed. A 15-h.p. motor runs the fan which draws off 700 cu. m. of air per minute and delivers it into the canal with the mine water, which precipitates the dust. This system keeps the dust down satisfactorily.

Coal Washing Improvements.[2]—The improvements in coal washing during the last 25 years in the Ruhr district are described by O. Simmersbach. The older systems of F. M. Simmersbach and Schüchterman and Cremer are especially dealt with. The latest Baum washer treats coal, without first classifying, from 0 to 3 in., according to the fundamental rule of "first classify and then wash." The plant principally used consists of jigs driven by air, trommels with screens placed one over the other and an unwatering conveyor. The water flowing away from this is cleared and returned to the washer, the slimes being led to separate basins.

The Pine Hill Breaker.[3]—This breaker, located near Minersville, Penn., is made of reinforced concrete and a framed oak superstructure. The coal is first screened over bar screens set 5 in., 4 in. 3 in. and 2 in. apart. The various oversizes are picked for rock and slate and crushed in rolls to market sizes. Bone and impure coal are crushed in rolls and together with fines go to the main breaker. This contains 21 shaking screens, three sets of rolls and 24 jigs which will have a capacity of 1500 to 1800 tons. The total capacity will be 2500 to 3000 tons.

Anthracite Washeries.[4]—At the present day washing is applied to fine anthracite from the mine and also to old culm banks which contain fine coal and dirt and were formed in earlier years when there was little or no demand for the finer sizes of coal. The washing operation is the same for both classes of material. The delivery of material from the culm banks to the washery may be accomplished by a steam shovel and cars or by a system of hydraulicking into conveyors. At the Capouse washery of the Scranton Coal Company the culm is flushed into a 500-ft. movable conveyor which delivers to a 380-ft. fixed scraping conveyor. The latter carries the material to (1).

1. Feeding chute. Foreign material, as wood, iron, slate, etc., picked out. Large lumps broken up. To (2).
2. Bucket elevator, 65 ft. long. To (3).
3. Three balanced shaking screens run wet. The top screen, 27x6 ft., has 1.5-in. holes for 21 ft.; then cross angle. irons, 0.75 in. apart, for 2.5 ft. to allow slate to fall through; finally 2-in. round holes for the last 4 ft. The middle screen 20x6 ft., has $\frac{7}{8}$-in. holes and the bottom screen, of the same dimensions, has $\frac{3}{32}$-in. holes. Over 2 in. is hand picked into slate and coal to (5); through $\frac{3}{4}$ in. is slate to waste; through 2 in. to (6); through 1.5 on $\frac{7}{8}$ in. to (7); through $\frac{7}{8}$ on $\frac{3}{32}$ in to (4); through $\frac{3}{32}$ in. to (10).
4. Four balanced shaking screens. The top one is 30x6 ft. with 0.5-in. holes; the next is 26x6 ft. with $\frac{3}{8}$-in. holes; the next is 20x6 ft. with $\frac{1}{4}$-in. holes; the bottom is 20x6 ft. with $\frac{3}{32}$-in. holes. Over 0.5 in. to (8); through 0.5 on $\frac{3}{8}$ is No.

[1]*Zeit. f. B. H. u. S. im Preuss.*, 1906, LIV, p. 285.
[2]*Berg-und Hüttenmännische Rundschau*, 1906, No. 13. Abstracted in *Journ.* Iron and Steel Institute, 1906, II, p. 811.
[3]*Mines and Minerals*, 1906, XXVI, p. 241.
[4]G. W. Harris, *Trans.* A. I. M. E., 1905. XXXVI, p. 610. Abstracted in *Eng. and Min. Journ.*, 1906, LXXXI, p. 798.

1 buckwheat to bin; through $\frac{3}{8}$ on $\frac{1}{4}$ is No. 2 buckwheat tobin; through $\frac{1}{4}$ on $\frac{3}{32}$ is No. 3 buckwheat, or barley, to bin; through $\frac{3}{32}$ to (11).
5. Two pairs of rolls, one 24x24 in. and one 21x24 in., for breaking coal. To (2).
6 One Christ jig. Slate to waste; coal to (5).
7. Two Christ jigs. Products like (6).
8. Automatic slate picker. Slate to waste; coal elevated to (9).
9. Three Christ jigs. Slate to waste; coal through Pardee slate picker to pea coal pin.
10. No. 1 settling pond or dump.
11. No. 2 settling pond or dump.

The washer can produce 120 tons of washed coal per hour. The $\frac{3}{32}$-in. screens are of bronze to withstand the action of acid mine water; the balance of the screens are of steel. The screens (3) receive 165 to 170 throws per minute and the screens (4) 180 throws per miunte.

The outside force required is one foreman, two hose men, 10 or 12 men on conveyors, two men to run conveyor engines, one man at elevator and two men at the settling ponds. Inside the washery are one machinist, one carpenter, three jig men, eight slate pickers, one engine man, one elevator man, four loaders, four car repair men, six slate dump men, three firemen, one man to cart out ashes and one man to wheel coal to the boilers.

Other washeries differ somewhat in details. Some of them save coal up to stove or even egg size while others crush everything down to buckwheat.

Cleaning Anthracite in Colorado.[1]—At the present time the only producing anthracite mine in Colorado is at Floresta. Here the coal is brought in cars and dumped on (1).

1. Bar screen. Rock and waste picked from oversize. Balance of oversize to (2); undersize to (3).
2. Lump crusher.
3. Mud screens. Various sizes to picking tables where waste is removed and remainder goes to market.
4. Main screens which make various merchantable sizes to market.

Various forms of standard mechanical slate pickers are used but final reliance has to be placed on men or boys to remove waste by hand. The breaker has a capacity of 1200 tons per day. It is run dry.

Baum Coal Washing Plant.[2]—A plant built by the Baum company of Herne in Westphalia for the Anna mine of the Eschweiler Mining Company has a capacity of 100 tons per hour. The wagons are dumped either to (1) or (3).

1. Shaking grizzly with 80-mm. holes. Oversize to (2); undersize to (7).
2. Picking belt. Good coal to railroad cars; waste to cars to dump.
3. Shaking sieve with 80, 35 and 4-mm. holes. Over 80 to (4); between 80 and 4 to (7); under 4-mm. to hopper and thence by cars and elevators to (18).
4. Picking belt. Good coal to railroad cars or to (5); waste either to cars to dump or to (5).
5. Conveying belt. To (6).
6. Coal crusher and rolls. To (7).
7. Bin. By elevator to (8).
8. One Baum air driven jig. Coal and middlings to (10); waste by elevator to (9).
9. Conveying belt to waste dump.
10. One Baum jig. Clean coal to (11); middlings to (15).
11. One trommel with 50, 30, 20 and 10 mm. holes. The four oversizes either go over sieves and pockets to be loaded for shipment or they go by an elevator to *schleudermühle* and thence to (17); the undersize of 10-mm. goes to pump to (12).
12. Two Baum jigs. Coal to (17); middlings by elevator to (13).
13. One Baum jig. Middlings to bin or to (17); waste to (14).
14. Elevator delivering by conveying belt to waste dump.
15. Breaker. To (16).
16. One Baum jig. Middlings product to bin; waste to (14).
17. Unwatering conveyor. Coal to (18); water to (19).
18. Screw conveyor and distributor. To bins supplying coke ovens.
19. Settling tanks. Clear water pumped back to jigs. Slime pumped to (17).

[1]R. M. Hosea, *Eng. and Min. Journ.*, 1906, LXXXII, p. 399.
[2]*Metallurgie*, 1906, III, p. 630.

The plant is run by eight motors at 550 volts and giving a total of 270 h.p. The plant is considered to be well arranged for reducing the handling of the coal to a minimum, avoiding dust and giving a good separation of the coal from the ash.

Bibliography of Coal Washing.[1]—This very complete bibliography covers the literature from 1851 to 1905.

[1]Samuel S. Wyer, Bulletin No. 8, A. I. M. E., March, 1906, p. 285.

AUSTRALASIA.

In the following tables the production of minerals and metals in each of the Australian States and New Zealand is separately itemized. In the tables relating to foreign commerce, however, the States are not separately treated, the combined statistics of the Commonwealth now being officially reported.

MINERAL PRODUCTION OF NEW SOUTH WALES. (a)

(In metric tons or dollars; £1=$5) (b)

Year.	Alunite.	Antimony and Ore.	Bismuth Ore.	Chrome Ore.	Coal.	Coke.	Cobalt Ore.	Copper Ore.	Copper Matte, Ingot and Regulus.
1896	1,394	134	42	3,914	3,972,069	26,774	...	15	4,464
1897	736	172	3	3,433	4,453,729	65,229	...	169	6,458
1898	2,988	83	29	2,145	4,781,551	83,538	119	181	5,577
1899	935	332	16	5,327	4,670,580	98,074	193	445	5,574
1900	1,946	252	11	3,338	5,595,879	128,238	145	867	6,243
1901	3,196	90	21	2,523	6,063,921	130,944	112	655	6,184
1902	3,702	58	10	508	6,037,083	128,902	35	3,190	5,560
1903	2,524	13	23	1,982	6,456,523	163,161	155	1,750	8,094
1904	376	111	41	404	6,116,126	173,742	6	2,470	6,654
1905	2,745	394	56	53	6,738,252	165,568	*Nil.*	487	7,899
1906	1,886	2,490	25	*Nil.*	7,748,384	189,038	*Nil.*	(h)	9,911

Year.	Diamonds. *Karats.*	Gold. (b)	Lead. Argentiferous. (g) Ore.	Lead. Argentiferous. (g) Metal. (f)	Lead. Pig. (g)	Molybdenite.	Opal.
1896	8,000	$5,222,971	271,641	19,886	24	..	$225,000
1897	9,189	5,373,596	275,249	18,395	32	..	375,000
1898	16,493	5,847,680	394,676	10,270	1,745	..	400,000
1899	25,874	7,899,075	431,126	20,614	(e)4,896	..	675,000
1900	9,828	5,211,097	426,480	19,400	(e)6,807	..	400,000
1901	9,322	3,587,040	406,560	17,191	(e)3,394	..	600,000
1902	11,995	3,333,064	371,496	15,660	(e)4,685	16	700,000
1903	12,239	5,255,421	335,870	18,779	(e)3,561	31	500,000
1904	14,296	5,576,966	373,362	30,212	(e)5,977	26	285,000
1905	6,354	5,669,099	420,266	28,244	214	20	295,000
1906	2,827	5,249,762	377,890	(i)	60	34	282,500

Year.	Platinum. *Kg.*	Shale Oil.	Silver—*Kg.* (g)	Stone. Limestone Flux.	Tin. Ore.	Tin. Block.	Tungsten Ore.	Zinc. (d) (g)
1896	75.8	32,348	6,307	90,347	98	1,147	..	......
1897	61.2	34,635	4,666	68,671	14	799	..	29,303
1898	38.9	30,164	16,580	9,401	1	639	..	39,564
1899	19.8	37,307	21,525	1,016	5	749	..	50,677
1900	15.6	23,229	24,080	17,273	15	1,087	..	20,594
1901	12.1	55,650	13,950	26,995	11	659	..	642
1902	11.6	63,886	33,195	17,630	23	502	..	1,281
1903	16.5	35,332	34,195	24,205	556	949	9	21,086
1904	16.6	38,477	34,880	25,374	586	1,084	106	58,523
1905	12.4	38,838	12,987	15,180	726	817	228	105,189
1906	6.4	32,965	8,865	12,993	(h)	1,698	245	105,325

(a) From the Annual Report of the Department of Mines, New South Wales. (b) Where gold is reported £1=$4.866. (d) Spelter and concentrate. (e) Includes minor quantities of lead carbonate and chloride, the product of the leaching plant at Broken Hill. (f) Includes a small quantity of silver-sulphide. (g) Exported. (h) Included with metal. (i) Included with ore.

MINERAL PRODUCTION OF NEW ZEALAND. (*a*) (*b*)

(In metric tons or dollars; £1=$5.) (*c*)

Year.	Antimony Ore.	Chrome Ore.	Coal.	Coke.	Copper Ore.	Gold. (*c*)	Kauri-gum.	Manganese Ore.	Silver—*Kg.*
1896	21	....	805,537	107	..	$5,067,589	7,240	66	2,933.3
1897	10	....	854,164	..	..	4,769,673	6,748	183	5,719.8
1898	..	....	921,546	9	2	5,258,642	10,063	220	9,140.0
1899	..	....	990,838	18	..	7,363,100	11,294	137	10,865.6
1900	3	28	1,111,860	..	12	7,005,103	10,322	166	10,202.0
1901	30	....	1,259,521	..	3	8,533,908	7,662	211	17,762.0
1902	..	128	1,386,881	..	..	9,495,673	7,549	...	20,970.3
1903	..	....	1,542,953	..	6	9,916,086	9,507	71	28,364.3
1904	..	....	1,537,838	..	..	9,671,180	9,203	196	34,042.3
1905	..	....	1,585,756	15	4	10,189,093	10,883	55	36,695.0

(*a*) From New Zealand Mines Statement, by the Hon. James McGowan, Minister of Mines, Wellington. (*b*) The exports are stated to be identical with the production, with the exception of coal, the exports of which were as follows: In 1896, 80,796 tons; in 1897, 77,280 tons; in 1898, 57,333 tons; in 1899, 90,912 tons; in 1900, 116,216 tons; in 1901, 162,197 tons; in 1902, 191,696 tons; in 1903, 154,769 tons; in 1904, 165,220 tons; in 1905, 122,817 tons. (*c*) Where gold is reported £1=$4.866.

MINERAL PRODUCTION OF QUEENSLAND. (*a*)

(In metric tons or dollars; £1=$5.)

Year.	Bismuth Ore.	Coal.	Copper Ore.	Gems other than Opal.	Gold. (*d*)	Lead.	Manganese Ore.
1895	60	328,237	441	(*b*)$29,575	$13,056,414	369	361
1896	..	377,332	589	(*c*)	13,235,842	628	305
1897	1	364,142	293	(*c*)	16,699,477	391	403
1898	8	414,461	63	(*c*)	19,016,763	252	68
1899	2	501,913	164	(*c*)	19,571,662	57	747
1900	8	505,252	386	4,500	20,002,290	207	77
1901	20	548,104	3,110	30,000	12,367,276	570	221
1902	1	509,579	3,845	25,000	13,238,500	271	4,674
1903	11	515,950	4,995	35,000	13,818,653	3,856	1,341
1904	20	520,232	4,440	52,875	13,210,869	2,079	843
1905	15	537,795	7,337	26,275	12,249,157	2,464	1,541
1906	7	616,480	10,238	90,550	11,257,316	2,854	1,131

Year.	Molybdenite.	Opal.	Silver *Kg.*	Stone, Building. (*b*).	Tin Ore.	Tungsten Ore.
1895	....	$163,750	6,999	52,206	2,148	25
1896	....	116,500	8,687	(*c*)	1,579	3
1897	....	51,250	7,280	(*c*)	1,222	13
1898	....	33,225	3,235	(*c*)	1,041	79
1899	....	45,000	4,521	164,939	1,322	263
1900	....	37,500	3,514	152,484	1,133	193
1901	....	37,000	17,777	(*c*)	1,638	73
1902	(*e*) 42	35,000	21,813	139,338	2,118	56
1903	(*e*) 24	36,500	19,972	107,780	3,768	200
1904	(*e*) 22	17,750	20,370	72,841	3,986	1,564
1905	64	15,000	18,716	(*f*)	4,008	1,434
1906	114	15,000	24,357	(*f*)	4,897	784

(*a*) From *Annual Reports of the Under Secretary of Mines*, Queensland, when not otherwise stated. (*b*) From *Mineral Statistics of the United Kingdom*. (*c*) Not reported. (*d*) Where gold values are reported, £1=4.866. (*e*) Includes bismuth and tungsten. (*f*) Returns not available.

MINERAL PRODUCTION OF SOUTH AUSTRALIA. (*a*)

(In metric tons or dollars: £1=$5.) (*b*)

Year.	Copper.		Gold. (*b*)	Iron Ore.	Lead.	Limestone.	Salt.	Other Metals and Minerals.
	Ore.	Metal.						
1896	354	4,176	$69,827	....	45	....	....	$3,775
1897	554	4,267	189,871	....	74	....	....	14,340
1898	545	4,327	51,949	....	286	....	....	7,550
1899	2,938	4,985	75,822	....	330	....	....	6,785
1900	2,405	4,432	70,528	....	347	....	....	89,835
1901	1,896	6,140	80,839	....	....	....	....	67,830
1902	2,620	6,210	121,056	....	1,973	....	....	102,160
1903	7,182	5,886	139,411	86,291	653	....	40,640	10,855
1904	3,100	5,694	369,938	47,434	....	44,135	40,640	990
1905	2,604	5,939	223,121	85,835	47	45,210	33,020	6,305

(*a*) From *Review of Mining Operation* by Hon. L. O'Loughlin, Adelaide, 1906.
(*b*) Where gold is reported £1=$4.866.

MINERAL PRODUCTION OF TASMANIA (*a*)

(In metric tons or dollars; £1=$5.)

Year	Coal.	Copper Ore and Matte.	Gold. (*e*)	Iron Ore.	Lead-Silver Ore.	Stone.			Tin and Tin Ore.
						Limestone.	Freestone, Flagstone, and Building Stones. *Cubic Feet.*	Rubble or Metal.	
1896	44,286	52	$1,156,035	203	21,150	2,621	13,575	(*g*)4,556	3,867
1897	43,210	113,261	1,407,447	999	17,806	1,702	82,197	13,274	3,282
1898	49,902	(*b*)	1,369,706	1,296	196,707	45,324	19,560	70,701	2,882
1899	43,803	(*c*)60,985	1,593,834	6,726	424,552	71,747	14,400	12,060	3,333
1900	51,549	(*d*) 4,221	1,538,727	5,141	453,579	47,671	51,509	4,291	2,693
1901	49,963	(*d*)11,401	1,436,326	1,422	804,463	26,545	37,579	244,721	2,516
1902	49,647	(*d*) 8,630	1,467,454	2,424	47,226	(*f*)	(*f*)	(*f*)	1,989
1903	49,856	(*d*) 3,891	1,237,925	6,076	43,103	(*f*)	(*f*)	(*f*)	2,414
1904	62,090	8,826	1,362,587	6,950	51,959	(*f*)	(*f*)	(*f*)	2,104
1905	52,825	9,919	1,520,101	6,401	76,424	(*f*)	(*f*)	(*f*)	3,953
1906	53,742	11,114	1,240,650	2,642	88,513	(*f*)	(*f*)	(*f*)	4,545

(*a*) From *Statistics of the Colony of Tasmania.* (*b*) Included with lead-silver ore. (*c*) In addition there were produced 43 tons of copper bullion. (*d*) In 1900 there were produced 9343 tons of blister copper; in 1901, 10,141 tons; in 1902, 7,869 tons and in 1903, 6,791 tons. (*e*) Where gold values are reported £1=$4.866. (*f*) Not reported. (*g*) Represents cart-loads.

MINERAL PRODUCTION OF WESTERN AUSTRALIA. (*a*)

(In metric tons or dollars; £1=$5.) (*b*)

Year.	Antimony.	Coal.	Copper Ore.	Gold. (*b*) (*c*)	Iron Ore.	Lead Ore.	Limestone.	Silver. *Kg.*	Tin Ore.
1900	..	120,305	6,282	$29,233,035	12,448	272	16,183	894	836
1901	..	119,721	10,319	35,208,687	20,898	(*d*)21	18,501	1,893	746
1902	..	143,145	2,298	38,673,323	4,877	(*d*)36	5,162	2,590	630
1903...........	22	135,568	20,854	42,678,319	224	*Nil.*	1,301	5,229	830
1904...........	..	140,773	4,033	40,992,284	1,465	*Nil.*	13,612	12,416	869
1905...........	..	129,402	2,389	40,415,312	3,264	*Nil.*	9,291	11,189	1,096
1906	*Nil.*	152,151	7,548	55,800,690	1,300	*Nil.*	9,624	.....	1,518

(*a*) From the *Report of the Department of Mines of Western Australia.* (*b*) £1=$4.866. (*c*) The value of gold produced in 1895 was $4,280,855; in 1896, $5,200,821; in 1897, $12,481,176; in 1898, $19,418,735. (*d*) Silver-lead ore.

MINERAL PRODUCTION OF VICTORIA. (*a*)

(In metric tons or dollars; £1=$5.)

Year.	Coal.	Lignite.	Gold. (*c*)	Stone, Building, etc.	Tin Ore.
1896.	230,187	5,908	$16,640,997	$485	47
1897...........	240,057	4,894	16,799,824	(*e*)125,000	48
1898...........	246,845	2,915	17,305,547	100,000	87
1899...........	266,578	(*b*)	17,662,410	(*b*)	158
1900	215,052	(*b*)	16,767,261	175,000	71
1901	212,678	152	16,320,029	225,000	78
1902	228,777	(*b*)	14,899,876	266,975	10
1903...........	65,230	5,752	15,860,815	213,245	34
1904..........	123,695	*Nil.*	15,824,952	1,488,075	72
1905...........	157,648	*Nil.*	15,443,438	(*b*)	126

(*a*) From Annual Reports of the Secretary for Mines of the Colony. (*b*) Not reported. (*c*) Where gold values are reported, £1=$4.866. (*e*) Estimated value.

MINERAL IMPORTS OF AUSTRALIA. (*a*).

(In metric tons or dollars; £1=$5.) (*b*).

Year.	Alkalies.	Brass Manufactures.	Bricks. Fire and Glazed.	Britannia, Yellow Metal, Etc.	Cement.	Chemicals.	Chinaware and Earthenware.	Coal.	Coke.
1900...	$395,870	$322,230	$25,890	$217,765	53,625	$4,457,060	$1,347,070	7,714	44,169
1901...	418,955	336,170	49,755	142,240	64,519	4,495,020	1,616,645	10,141	36,814
1902...	401,225	218,980	50,115	162,510	48,729	4,237,505	1,358,275	5,149	9,846
1903...	427,540	99,600	22,645	135,425	43,292	3,926,610	940,170	389	4,294
1904...	450,085	72,425	35,525	120,110	25,452	4,126,245	1,151,520	398	4,270
1905...	557,370	109,735	28,900	63,920	31,791	2,571,755	1,068,090	7,866	5,553

Year.	Copper.		Glass and Glassware.	Gold. (*b*)				
	Ore.	Manufactures.		Ore.	Bullion.	Specie.	Foil. (*c*)	Total Value.
1900...	1,423	$249,735	$1,900,875	$ 14,880	$4,556,007	$ 78,888	$51,224	$4,700,999
1901...	659	294,870	1,807,805	37,473	3,709,848	18,053	34,704	3,800,078
1902...	1,326	335,105	1,508,530	2,375,513	3,834,510	505,899	30,028	6,740,950
1903...	....	338,915	1,257,860	66,908	5,935,800	6,530	38,680	6,047,918
1904...	....	325,560	1,398,735	68,309	5,684,164	6,297	43,215	5,801,985
1905...	4	226,625	1,379,490	103,709	7,067,534	422,127	52,144	7,645,514

Year.	Graphite.	Iron and Steel.						
		Bars, Rods, Girders, Sheets, etc.	Galvanized Plates and Sheets.	Pig and Scrap.	Tin Plate.	Manufactures.	Pipe and Tubes.	Railway Material.
1900	137	100,849	44,598	44,683	$1,520,585	$5,182,730	$1,500,850	$3,888,325
1901	155	94,395	41,075	33,220	1,262,720	7,865,560	1,284,830	6,395,705
1902	121	......	50,099	34,772	1,393,045	7,967,025	1,005,975	6,318,475
1903	252	54,940	40,207	44,898	805,810	3,802,190	1,193,540	2,472,940
1904	193	63,482	46,614	40,063	879,755	2,526,750	1,130,185	920,180
1905	199	44,264	50,506	88,110	748,835	7,407,215	1,293,390	1,030,455

Year.	Jewelry and Precious Stones.	Lead Mfrs.	Paints and Colors.	Petroleum Products.			Potassium Nitrate.	Platinum.
				Kerosene—*gal.*	Naphtha—*gal.*	Paraffin.		
1900........	$1,570,560	368	$1,570,880	11,125,905	48,863	1,275	369	...
1901........	1,990,570	166	1,551,270	20,924,640	114,092	1,040	297	....
1902........	1,756,450	376	1,293,960	10,399,931	116,170	1,913	361	....
1903........	2,024,275	380	1,092,420	15,009,609	127,445	2,163	211	$9,255
1904........	2,234,375	242	1,299,875	14,791,319	277,737	530	354	910
1905........	2,115,955	448	1,417,670	16,416,734	292,670	....	409	1,875

Year.	Quicksilver.	Salt.	Silver. (b)			Stone, including slate and marble.	Sulphur.	Zinc.	
			Ore.	Bullion. *Kg.*	Specie.			Bar and Old.	Spelter.
1900...............	63.2	22,061	9	190.4	$1,226,208	$364,705	4,972	$213,800	616
1901...............	91.0	25,422	743	14.9	772,020	330,185	4,501	154,555	648
1902...............	92.6	25,920	252	13.6	439,186	403,185	7,853	132,725	950
1903...............	87.5	14,180	...	14.2	160,111	353,465	8,196	158,940	644
1904...............	92.6	16,127	...	39.8	154,534	393,135	11,462	134,665	1,057
1905...............	82.1	22,370	...	3908.0	261,397	301,350	8,050	154,775	1,190

(*a*) From Trade and Customs Returns, Commonwealth of Australia, 1900. Previous to 1900 each Colony reported its own imports and exports. (*b*) Where gold, silver or platinum values are reported, £1=$4.866. (*c*) Includes a small quantity of silver foil.

MINERAL EXPORTS OF AUSTRALIA. (*a*)

(In metric tons or dollars: £1=$5.) (*c*).

Year.	Alunite.	Bismuth.		Cement.	Chrome Ore.	Coal.	Coke.	Cobalt. Ore.
		Ore.	Metal.					
1900..........	1,737	9	4	....	2,190	1,774,980	6,005	130
1901..........	2,853	45	5	......	1,861	1,750,066	4,465	100
1902..........	2,305	6	6	.....	453	1,687,621	6,080	34
1903..........	2,253	33	9	506	1,769	2,063,016	27,345	139
1904..........	344	87	..	1,193	360	1,637,113	2,771	8
1905..........	2,453	101	..	1,816	.....	2,058,190	2,316	60

Year.	Copper.				Glass and Glassware.	Gold. (*c*)			
	Ore.	Bullion.	Ingots and Matte.	Mfrs.		Ore.	Bullion.	Specie.	Total Value.
1900	4,108	....	15,886	(*b*)	$28,650	$2,379	$19,604,657	$41,898,304	$61,505,340
1901	10,505	....	17,643	(*b*)	30,855	65,341	22,416,198	43,233,515	65,715,054
1902	7,490	....	21,075	(*b*)	58,325	1,214,208	20,736,800	41,954,939	63,905,947
1903	2,792	....	27,949	23	61,360	80,591	29,691,889	53,634,629	83,407,109
1904	4,086	92.1	24,539	14	44,245	46,894	27,073,767	49,284,833	76,405,494
1905	789	77.4	28,701	...	117,680	49,507	25,788,574	27,523,288	53,361,369

Year.	Iron and Steel.		Jewelry and Precious Stones.	Lead.				Molybdenum Ore.	Paints and Colors.
	Bars, Rods, Scrap, etc.	Manufactures.		Ore.	Pig and Matte.	Argentiferous.	Manufactures.		
1900...	284	$16,965	$290,675	24	17,200	29,711	998	..	$2,990
1901 ..	(d)199	59,150	323,750	110	12,761	30,333	1,025	..	3,100
1902. .	144	29,305	345,195	193	16,590	28,950	790	7	4,870
1903...	(d)261	40,290	371,375	2,257	28,744	20,093	1,305	36	12,940
1904 ..	224	57,285	380,345	(e)	73,754	35,847	932	50	6,725
1905. .	6,619	80,635	1,013,595	(e)	59,130	34,187	1,588	63	58,120

Year.	Platinum.	Salt.	Shale Oil.	Silver.		Stone, including marble and slate.	Tin.		Zinc.	
				Ore.	Bullion, Ingot and Matte.		Ore	Block.	Bar and Old.	Spelter.
1900	.....	4,575	16,792	72,483	192,327.9	$5,470	309	3,273	$3,440	2 601
1901	.....	7,109	19,587	73,825	196,135.9	24,920	227	2,727	1,690	78
1902......	.....	10,802	27,896	65,084	189,703.3	35,545	466	2,876	2,980	202
1903......	$5,163	7,057	14,483	72,740	202,730.4	10,130	1,219	3,740	20,465	2,730
1904......	5,207	6,419	8,202	101,379	227,972.1	10,450	1,829	4,511	7,420	14,032
1905......	7,545	8 365	11,818	26.407	208,134.4	8,140	2,504	4,947	15,565	136,514

(a) From "Trade and Customs Returns," Commonwealth of Australia, 1904.—Note. Previous to 1900 each Colony reported its own exports separately. (b) Included with ingots and matte. (c) Where gold, platinum or silver values are reported £1=$4.866. (d) Includes a small quantity of scrap. (e) Included under silver ore.

AUSTRIA—HUNGARY.

In the following tables the mineral and metal productions of the two Kingdoms are reported separately, together with that of Bosnia and Herzegovina. In the tables relating to foreign commerce, the statistics are those of the combined Empire.

MINERAL AND METALLURGICAL PRODUCTION OF AUSTRIA. (a)
(In metric tons.)

Year.	Alum.	Alum and Pyritic Shale.	Antimony. Ore.	Antimony. Metal.	Asphaltic Rock.	Bismuth Ore.	Coal. Bituminous.	Coal. Lignitic.
1895	885	5,716	695	296	404	185.0	9,722,679	18,389,147
1896	919	25,184	905	422	390	*Nil*	9,899,522	18,882,537
1897	851	21,585	864	425	300	1.0	10,492,771	20,458,093
1898	1,037	28,914	679	343	643	*Nil*	10,947,522	21,083,361
1899	604	19,879	410	271	2,635	0.3	11,455,139	21,751,794
1900	620	3,004	201	153	887	4.0	10,992,545	21,539,917
1901	442	2,551	126	114	541	16.0	11,738,840	22,473,510
1902	62	2,866	18	24	897	8.0	11,045,039	22,139,683
1903	*Nil*	2,978	41	14	1,273	10.0	11,498,111	22,157,521
1904	*Nil*	2,337	103	36	1,435	1.7	11,868,245	21,987,651
1905	*Nil*	1,657	1,673	90	4,363	1.7	12,585,263	22,692,076

Year.	Copper Ore.	Copper Metal.	Copper Sulphate.	Copperas.	Gold. Ore.	Gold. Bullion.	Graphite.	Iron. Ore.	Iron. Pig & Cast
1895	7,435	865	246	160	104	$49,841	28,443	1,384,911	660,549
1896	6,823	1,001	265	170	416	46,386	35,972	1,448,615	693,188
1897	7,405	1,083	276	125	647	44,924	38,504	1,613,876	762,685
1898	6,791	1,041	209	360	448	47,515	33,062	1,733,649	837,767
1899	6,731	1,123	235	475	387	50,306	31,819	1,725,143	872,352
1900	5,825	881	234	474	227	47,183	33,663	1,894,458	879,132
1901	7,406	776	256	472	143	31,234	29,992	1,963,246	884,844
1902	8,455	914	248	271	74	4,652	29,527	1,742,498	991,827
1903	12,688	961	310	298	2,148	5,316	29,590	1,715,984	970,832
1904	10,701	889	808	414	12,653	47,183	28,620	1,719,219	988,364
1905	10,677	870	540	116	35,937	135,570	34,416	1,913,782	1,119,614

Year.	Lead. Ore.	Lead. Pig.	Lead. Litharge.	Manganese Ore.	Mineral Paint.	Petroleum.	Quicksilver. Ore.	Quicksilver. Metal.	Salt.
1895	12,919	8,085	2,034	4,352	3,164	188,634	86,683	535	278,875
1896	14,563	9,769	1,738	3,950	3,979	262,356	83,305	564	308,933
1897	14,145	9,680	1,626	6,012	3,653	275,204	88,238	532	331,084
1898	14,363	10,340	1,520	6,132	3,213	323,142	88,519	491	341,959
1899	12,820	9,736	1,526	5,411	2,055	309,590	92,323	536	342,059
1900	14,314	10,650	1,288	8,804	2,828	347,213	94,747	510	330,277
1901	16,688	10,161	1,317	7,796	1,701	404,662	97,360	525	333,238
1902	19,055	11,264	1,023	5,646	1,486	520,845	90,040	511	311,806
1903	22,196	12,162	923	6,179	1,691	672,508	83,321	523	359,015
1904	22,514	12,645	783	10,189	1,829	....	88,279	536	369,877
1905	23,339	12,968	865	13,788	798	....	86,856	520	343,375

Year.	Silver.		Sulphuric Acid.	Sulphur Ore.	Tin.		Tungsten Ore.	Uranium.		Zinc.	
	Ore.	Bullion. (*Kg*)			Ore.	Block.		Ore.	Salts.	Ore.	Spelter.
1895.......	18,113	40,081	7,431	830	24	60	35	31	4.5	25,862	6,456
1896.......	18,701	39,904	7,972	643	15	54	22	30	4.2	26,887	6,888
1897.......	20,628	40,026	8,515	530	16	48	31	44	4.4	27,463	6,236
1898.......	20,886	40,304	7,003	496	13	48	36	51	4.3	27,395	7,302
1899.......	21,554	39,564	7,814	555	54	41	50	49	7.6	37,100	7,192
1900.......	21,641	39,572	7,067	862	51	40	50	52	11.3	38,243	6.742
1901.......	21,363	40,205	7,073	4,911	42	49	45	48	13.0	36,072	7,558
1902.......	22,288	39,544	8,781	3,721	47	50	45	46	10.0	31,927	8,309
1903.......	21,958	39,812	9,105	4,475	57	34	49	45	6.0	29,544	8,949
1904.......	21,949	39,032	8,742	6,288	77	38	52	17	11.0	29,226	9,159
1905.......	21,047	38.453	1,007	8.407	52	53	55	16	13.9	29.983	9,326

(*a*) From the ***Statistisches Jahrbuch des K. K. Ackerbau-Ministeriums.***

MINERAL AND METALLURGICAL PRODUCTION OF HUNGARY. (*a*)

(In metric tons or dollars; 1 Crown=$0.203.)

Year.	Antimony.		Asphalt.	Asphaltic Rock.	Bismuth	Carbon Bi-Sulphide.	Coal.			
	Ore.	Regulus.					Bituminous (*e*).	Lignite (*e*).	Coke.	Briquets.
1895 .	1,240	465	2,285	....	....	237	1,068,046	3,517,901	12,033	29,421
1896. .	1,361	500	2,740	....	....	352	1,132,625	3,761,728	25,550	31,179
1897 .	1,800	523	3,057	....	4.7	432	1,118,024	3,870,530	(*d*)	27,022
1898. .	2,201	855	3,125	....	3.1	771	1,239,498	4,516,581	(*d*)	31,781
1899. .	1,965	940	3,060	....	3.0	1,120	1,238,855	4,292,584	10,336	31,137
1900. .	2,373	846	2,700	....	2.0	1,250	1,447,047	5,128,277	12,973	69,353
1901. .	(*b*) 323	706	2,878	25,161	1.6	2,087	1,365,270	5,179,829	10,975	40,182
1902 .	(*b*) 748	683	2,774	24,873	0.9	2,320	1,162,785	5,132,053	8,204	88,069
1903 .	(*b*) 205	732	2,422	21,552	1.5	2,357	1,233,410	5,271,781	9,442	101,197
1904 .	1,080	1,007	2,221	17,660	0.9	2,512	1,155,320	5,519,349	5,103	103,481
1905 .	949	756	173	19,372	1.4	2,760	1,088,087	6,088,578	69,303	144,697

Year.	Copper.	Copperas.	Gold.	Iron.			Lead.		Litharge.	Manganese Ore.
				Ore. (*e*).	Pig.	Cast.	Ore.	Pig.		
1895.......	286	521	$2,118,100	9,955,262	322,206	....	.....	2.277	615	3,525
1896	159	595	2,131,876	1,269,680	383,698	....	.....	1,911	465	2,101
1897.......	213	592	2,038,839	1,421,130	402,503	....	525	2,527	155	4,030
1898.......	153	745	1,839,474	1,666,837	448,621	20,784	771	2,305	188	8,087
1899	165	771	2,039,504	1,587,600	451,637	19,631	526	2,166	213	5,073
1900.......	181	700	2,173,079	1,666,363	432,817	22,738	612	2,030	201	5,746
1901.......	162	805	2,189,692	1,557,300	430,686	20,640	(*b*) 10	2,029	238	4,591
1902.......	89	909	2,260,135	1,562,238	416,835	18,569	(*b*) 20	2,244	219	7,237
1903.......	45	982	2,243,521	1,439,132	396,674	18,875	(*e*)3,698	2,057	257	5,311
1904......	63	1,277	2,437,998	1,524,036	370,297	17,203	(*e*)3,922	2,104	710	11,527
1905......	73	920	2.439,451	1.661,358	403,719	17,563	686	2,146	209	9,944

Year.	Mineral Paints.	Petroleum	Pyrites.	Quicksilver *Kg.*	Salt.	Silver-*Kg.*	Sulphur	Sulphuric Acid.	Zinc.	
									Ore. (*b*)	Spelter.
1895	371	2,083	69,195	1,129	169,395	20,432	102	4,223	(*d*)	..
1896.....	334	2,168	52,697	1,100	180,133	19,916	138	3,550	(*d*)	..
1897.....	460	2,229	44,454	700	171,711	26,790	112	3,397	30	..
1898.....	247	2,471	58,079	6,800	178,551	18,799	93	1,318	30	..
1899.....	394	2,125	79,519	27,000	182,593	20,991	116	1,463	1,197	..
1900.....	370	2,199	87,000	31,800	189,363	20,202	123	1,371	326	..
1901.....	305	3,296	93,907	33,003	184,083	23,636	137	1,464	693	14
1902.....	283	4,347	106,490	44,600	174,882	23,020	105	1,193	364	..
1903.....	263	3,010	96,619	43,700	183,328	19,281	135	1,543	46	26
1904.....	273	2,134	97,148	45,169	187,620	16,352	143	1,329	203	..
1905.....	196	471	106,848	36,000	195.410	15,946	135	1,410	173	..

(a) From the ***Annuaire Statistique Hongrois.*** (b) Includes only that part of the crude output that was not smelted into a refined product. (d) Not reported. (e) Total production.

MINERAL AND METALLURGICAL PRODUCTION OF BOSNIA AND HERZEGOVINA. (a)
(In metric tons.)

Year	Chrome Ore.	Copper. Ore.	Copper. Metal.	Iron. Ore.	Iron. Pig	Lignite	Manganese Ore.	Pyrites.	Quicksilver.	Salt.
1895....	707	(b)	105	(b)	2,569	195,422	8,145	....	(b)	12,758
1896....	443	(b)	206	(b)	10,120	222,724	6,821	....	(b)	13,720
1897 ...	396	3,847	135	37,095	15,606	229,643	5,344	....	(b)	13,919
1898....	458	3,760	156	57,935	15,263	270,752	5,320	3,670	4.0	14,496
1899....	200	3,980	180	67,030	13,730	303,000	5,270	....	3.3	15,030
1900....	100	3,008	141	133,454	38,960	394,516	7,939	1,700	6.7	15,791
1901....	505	3,696	199	122,569	39,296	445,007	6,346	4,570	9.3	16,865
1902....	270	3,657	166	133,348	43,992	424,753	5,760	5,170	7.2	17,348
1903....	147	1,073	191	114,059	39,833	467,962	4,538	6,589	8.1	18,459
1904....	279	640	115	127,297	47,678	483,617	1.114	10,421	8.1	18,021
1905....	186	670	39	122,540	43,074	540,237	4,129	19,045	10	(b)

(a) From *Oester. Zeits. f. B. H.* (b) Not reported.

MINERAL IMPORTS OF AUSTRIA-HUNGARY. (a)
(In metric tons or dollars; 5 Crowns=$1.)

Year.	Alum.	Aluminum and Alloys.	Aluminum, Sulphate and Chloride.	Ammoniacal Liquor.	Ammonium. Chloride and Sulphate.	Ammonium. Hydrate.	Antimony. Ore.	Antimony. Regulus—*Kg.*
1895..........	338	48	1,278	877	305	103	15	2,100
1896..........	359	50	1,128	507	323	71	16	700
1897..........	346	67	1,351	565	339	128	8	600
1898..........	338	101	1,822	230	430	80	12	28,200
1899..........	332	121	1,299	73	358	46	10	30,400
1900..........	430	154	1,435	176	573	65	46	23,000
1901..........	413	153	1,882	293	620	44	27	1,500
1902..........	537	151	2,161	402	438	22	40	18,200
1903..........	508	150	2,670	580	361	23	42	87,200
1904..........	602	231	2,346	426	388	20	64	21,000
1905..........	774	477	2,775	312	600	....	......	24,700

Year.	Arsenic, (b)	Asbestos. Crude (c).	Asbestos. Manufactures	Asphalt. Crude Rock.	Asphalt. Mastic and Bitumen.	Barytes.	Borax. Crude and Boric Acid.	Borax. Refined.
1895.......	293	432	108	2,410	872	5,098	1,908	62
1896.....	309	185	165	4,715	1,621	5,377	1,363	76
1897.......	259	625	134	5,824	1,309	4,947	1,206	63
1898.......	287	609	138	5,973	1,117	5,012	784	185
1899.......	284	866	1347	7,301	1,546	5,443	2,212	130
1900.......	320	1,085	1238	8,301	1,564	5,945	3,056	93
1901.......	351	1,678	1032	5,702	1,106	6,336	1,687	233
1902.......	351	2,038	798	5,732	1,273	6,266	2,168	174
1903.......	371	3,395	1221	5,871	1,272	7,057	2,192	150
1904.......	384	2,517	1240	8,211	1,064	6,238	2,752	142
1905......	342	5,962	208	8.553	1,139	6,187	3,099	205

Year.	Brass, German Silver and Tombac. Crude, Old and Scrap.	Brass, German Silver and Tombac. Bars, Sheets and Wire.	Brass, German Silver and Tombac. Wares.	Cement.	Chloride of Lime.	Chrome Ore.	Clay Products. Kaolin and Feldspar.	Clay Products. Manufactures.	Clay Products. Potters' and Other Clays.
1895....	2,742	131	510	32,012	2,538	1,827	6,532	194,476	27,493
1896....	3,118	113	526	35,290	1,989	1,891	7,425	213.208	30,072
1897....	2,660	152	549	32,479	1,820	1,109	6,913	186,297	28,925
1898....	3,232	182	607	30,745	2,851	2,206	7,991	183,822	31,905
1899....	2,699	168	588	21,410	3,749	1,874	8,152	177,119	30,799
1900....	2,654	54	579	25,747	3,326	2,823	6,847	179,699	30,419
1901....	3,771	121	577	23,559	3,326	860	7,687	190,585	36,448
1902....	3,744	424	625	18.658	2,596	2,668	9,085	209,734	28,985
1903....	3,716	628	658	23,256	2,791	2,121	9,940	234,008	32,015
1904....	4,761	498	769	20,259	3,407	1,209	10,854	250,568	33,281
1905...	4,379	638	795	21,950	1.847	2,305	13.656	246.004	36.890

Year.	Coal.		Coke.	Copper.			Copper Sulphate.	Cop-peras.	Cryolite.
	Bituminous.	Lignitic.		Ore.	Bullion and Scrap.	Bars, Sheets, Wire, etc.			
1895......	4,503,003	16,797	533,402	31	11,747	98	895	871	229
1896......	5,174,321	19,981	491,028	1	13,666	126	2,084	575	265
1897......	5,121,475	19,609	533,463	81	15,926	94	6,822	401	211
1898......	5,396,760	19,393	606,783	64	17,443	159	5,271	466	275
1899......	5,296,700	20,879	564,005	*Nil*	16,185	156	2,345	409	343
1900......	6,242,939	67,740	620,776	16	18,970	121	3,516	343	342
1901......	5,827,332	22,253	612,209	112	17,504	83	2,822	269	428
1902......	5,766,377	29,601	547,406	100	18,498	149	2,839	274	447
1903......	5,907,660	30,007	519,281	209	18,701	89	3,526	155	521
1904......	6,190,030	30,001	548,272	1,107	22,532	89	4,508	238	313
1905......	6,418,042	36,000	554,147	1,397	22,652	73	3,791	169	220

Year.	Fertilizers, Mineral.	Fluorspar.	Glass (all kinds).	Gold. (*d*)			Graph-ite.	Gypsum.	
				Bullion.	Coin.	Old and Dross.		Crude.	Burned.
1895....	....	3,528	...	$3,470,945	$17,384,964	$76,290	640	850	10,916
1896....	......	3,821	....	8,674,371	16,956,256	17,260	697	821	11,736
1897....	......	4,201	....	22,374,069	18,164,128	23,120	948	980	12,101
1898....	......	4,169	....	323,636	8,853,354	11,520	1,109	991	13,300
1899....	80,658	4,959	4,422	432,187	7,662,641	6,902	815	1,336	13,441
1900....	71,966	5,649	3,954	1,111,831	7,230,251	8,932	302	1,348	15,462
1901....	93,430	5.774	4,219	13,865,103	20,353,592	14,819	318	1,405	15,830
1902....	99,502	5,902	4,182	14,509,019	15,695,960	12,789	221	1,588	16,430
1903....	128,320	5,445	2,474	9,825,200	9,817,283	14,819	405	1,969	18,655
1904....	175,681	7,061	2,779	12,703,740	8,586,394	22,127	423	2,384	19,387
1905....	216,075	7,601	4.180	1,047,792	9,204,968	4,375	735	1,553	21,286

Year.	Hydro-chloric Acid.	Iron.				Lead.					
		Ore.	Pig and Old.	Manufac-tures.	Iron and Steel Bars, Sheets, Wire, etc.	Ore.	Pig.	Alloys, Crude.	Lith-arge.	Red and Yellow.	White.
1895....	467	117,600	175,400	$3,990,400	30,909	416	208	8,974	355	371	187
1896....	529	107,018	148,217	4,258,400	27,809	540	218	7,221	233	432	156
1897....	721	134,778	164,433	4,582,400	18,625	441	148	5,887	224	543	111
1898....	766	178,507	173,919	4,627,200	26,421	459	153	9,746	280	555	115
1899....	350	212,412	126.371	4,395,356	12,340	465	235	8,836	224	466	80
1900....	577	233,156	95,530	4,533,599	10,313	501	175	7,916	141	354	106
1901....	576	218,476	90,287	4,443,670	10,902	1,270	311	10,722	189	433	135
1902....	588	197,525	43,314	4,304,818	11,584	1,879	348	8,706	149	428	221
1903....	603	217,979	47,354	4,508,224	11,025	1,355	409	9,190	141	423	173
1904....	459	182,515	35.091	4,976,342	9,402	1,436	349	7,917	146	372	138
1905....	656	228,149	49,383	5,722,976	....	247	475	7,282	101	349	88

Year.	Magnesium Chloride.	Manganese Ore.	Millstones.	Mineral Paints.	Nickel.		Nickel and Cobalt Ores.	Nitric Acid.	Peat and Peat Coke.
					Old and Crude.	Sheet, Wire etc.			
1895.......	1,353	2,772	1,229	4,244	168	5	1,020	16	1,993
1896.......	1,333	7,371	1,205	4,362	161	5	719	21	2,002
1897.......	1,530	8,018	1,275	4,553	157	7	55	23	2,189
1898.......	2,096	5,396	1,429	4,979	137	9	510	22	1,511
1899.......	2,043	5,855	1,458	5,106	119	11	198	39	2,075
1900.......	2,100	7,016	1,672	4,958	258	8	406	36	2,664
1901.......	2,529	6,367	1,595	5,109	277	10	788	22	2,896
1902.......	2,621	15,595	1,410	4,831	265	8	225	90	3,234
1903.......	3,118	38,529	1,395	4,733	268	9	385	7	3,097
1904.......	2,997	35,357	1,282	5,563	402	11	656	24	2,676
1905.......	3,495	30,483	1,467	6,018	632	11	391	14	2,432

Year.	Petroleum Products. Crude Oil.	Refined Oil.	Paraffin.	Phosphorus and Phosphoric Acid.	Potassium Salts. Carbonate.	Chloride.	Chromate.	Pyrites.
1895	120,479	16,876	.....	226	285	2,679	29	54,610
1896	69,013	17,943	.....	224	987	2,475	34	50,691
1897	70,573	21,249	.....	209	333	2,206	34	49,462
1898	58,580	22,299	.....	209	300	2,258	3	52,282
1899	75,885	21,823	6,968	221	526	3,264	1	54,844
1900	20,813	22,963	5,080	204	1,029	3,633	11	60,317
1901	22,545	18,067	5,294	222	1,442	4,356	21	54,202
1902	24,830	15,864	4,238	225	485	3,377	11	60,235
1903	19,710	19,382	2,598	237	197	3,727	9	73,835
1904	20,110	22,715	1,470	193	222	3,557	3	65,397
1905	18,974	24,961	888	222	154	3,864	5	86,338

Year.	Quicksilver *Kg.*	Salt.	Silica, Quartz and Sand.	Silver. Bullion, *Kg.*	Old and Dross, *Kg.*	Specie.	Slag and Slag Wool.
1895	4,200	40,396	58,494	49,370	60	$90,353	981
1896	1,300	53,680	59,150	138,420	550	137,244	240
1897	1,000	46,057	61,532	99,900	1,000	75,944	4,717
1898	6,300	41,870	70,296	15,400	3,000	103,424	9,655
1899	2,600	37,883	71,279	28,900	600	112,056	5,665
1900	1,300	39,822	77,930	29,300	2,700	199,955	4,679
1901	2,600	39,625	83,401	41,800	1,700	207,669	3,068
1902	1,300	46,128	92,617	177,900	1,500	237,104	4,176
1903	1,600	48,793	94,492	150,400	5,700	250,299	3,850
1904	2,500	94,103	97,364	36,700	1,700	420,413	4,716
1905	2,400	......	104,195	36,100	300	143,152	4,094

Year.	Slate and Other Roofing.	Sodium Salts. Bi-Sulphate.	Carbonate.	Carbonate. (Calcined).	Hydrate.	Nitrate.	Sulphate.
1895	15,667	137	40	551	1,163	43,059	6,617
1896	15,932	144	57	1,332	835	33,086	4,678
1897	16,758	91	45	2,787	1,450	39,600	2,879
1898	16,025	89	53	2,408	1,498	41,773	4,476
1899	15,562	85	62	1,123	1,669	47,301	5,394
1900	13,047	73	104	1,141	1,836	54,559	5,110
1901	11,555	98	77	911	1,280	63,283	4,452
1902	14,378	17	97	312	1,030	39,958	5,997
1903	11,531	13	110	327	956	54,896	6,116
1904	9,170	103	103	1,109	659	54,887	5,409
1905	8,852	167	168	965	475	66,740	5,258

Year.	Stone. Lithographic.	Marble.	Paving.	Not Elsewhere Specified.	Sulphur.	Sulphuric Acid.
1895	684	1,886	5,459	79,869	14,709	1,566
1896	647	2,347	8,476	87,796	15,221	3,522
1897	524	2,333	16,961	118,848	21,406	5,877
1898	786	2,769	11,234	99,193	20,655	9,724
1899	611	2,850	27,067	82,878	23,504	10,245
1900	640	3,133	5,781	81,347	27,795	10,643
1901	616	2,908	11,299	77,695	25,300	11,712
1902	363	3,184	10,203	88,214	23,878	12,474
1903	668	3,132	14,266	74,960	22,625	16,148
1904	679	4,016	12,177	89,791	30,505	19,878
1905	766	4,122	11,513	135,632	30,227	17,320

Year.	Tin. Ingot, Crude, Old, etc.	Salts	Whetstones.	Zinc. Calamine and Other Ores.	Spelter.	Bars, Sheets Wire, etc.	White.
1895	3,038	46.0	3,559	7,691	17,156	611	510
1896	3,344	27.7	3,851	9,022	17,539	552	590
1897	3,467	22.3	4,151	7,863	16,599	356	577
1898	3,769	20.9	3,490	14,112	17,471	453	697
1899	3,005	30.2	3,717	12,730	15,225	481	750
1900	3,439	23.9	3,643	14,181	17,844	667	875
1901	3,671	24.9	3,445	18,403	16,921	579	718
1902	3,638	31.7	3,599	20,723	17,034	651	636
1903	3,564	31.3	3,774	22,344	17,973	746	698
1904	3,528	54.8	4,272	24,039	20,787	731	840
1905	3,845	60.8	4,376	22,890	21,874	568	972

(*a*) From *Statistik des Auswaertigen Handels des Oesterreichisch-Ungarischen Zollgebiets.* (*b*) Includes arsenious acid and sulphide. (*c*) Includes burned asbestos. (*d*) The values of gold are figured at the rate of one crown=$0.203

MINERAL EXPORTS OF AUSTRIA-HUNGARY. (a)

(In metric tons or dollars; 5 crowns=$1.)

Year.	Alum.	Aluminum, Sulphate and Chloride.	Ammonium. Chloride and Sulphate.	Ammonium. Hydrate	Ammoniacal Liquor.	Antimony. Ore.	Antimony. Regulus.	Arsenic, Arsenious Acid, and Orpiment.
1895...	60	231	876	92	413	193	369	36
1896...	47	267	2,524	70	604	218	441	26
1897...	70	210	4,188	39	592	289	359	16
1898...	83	253	4,886	23	724	266	679	29
1899...	54	233	7,576	42	734	562	240	47
1900...	44	164	7,004	82	942	247	276	65
1901...	55	211	8,824	113	705	179	385	80
1902...	102	135	11,777	93	717	174	290	89
1903...	77	14	11,478	155	814	128	249	63
1904...	38	2	12,386	130	654	200	673	72
1905...	68	34	15,191	114	472	...	774	42

Year.	Asbestos. Crude.	Asbestos. Manufactured.	Asphalt. Rock and Earth.	Asphalt. Mastic and Bitumen.	Barium. Sulphate. (b)	Barium. Chloride.	Brass, German Silver and Tombac. Crude, Old and Scrap.	Brass, German Silver and Tombac. Bars, Sheets, Wire, etc.	Chloride of Lime.
1895....	122	10	145	1,183	..	.....	1,115	459	267
1896....	48	10	134	1,692	..	.....	1,469	399	114
1897....	56	19	102	2,593	..	.....	1,699	680	111
1898....	150	28	183	2,126	..	.....	1,879	839	113
1899....	71	60	1,143	2,619	65	.....	1,824	910	203
1900....	47	168	1,218	2,177	23	.....	1,811	1,256	192
1901....	36	165	198	1,909	55	4,098	2,006	1,054	738
1902....	65	275	520	301	64	4,552	2,298	1,098	426
1903....	89	495	921	483	52	5,091	2,576	853	674
1904....	290	1,582	403	728	74	4,233	2,715	1,033	254
1905....	330	1,397	1,060	457	26	4,626	3,137	1,173	978

Year.	Cement.	Chrome Ore.	Clay Products. Manufactures.	Clay Products. Kaolin and Feldspar.	Clay Products. Potters' and other Clays.	Coal. Bituminous.	Coal. Lignitic.	Coke.
1895...	12,804	385	51,767	56,203	37,667	640,963	7,143,234	119,051
1896...	16,721	142	67,270	67,381	41,276	658,368	7,562,721	116,608
1897...	19,786	153	55,895	68,609	46,968	701,919	8,108,975	145,056
1898...	23,989	121	66,005	74,003	52,851	824,730	8,351,955	194,289
1899...	38,193	53	72,104	78,537	61,898	879,337	8,662,788	252,971
1900...	46,761	22	75,672	103,178	66,869	815,097	7,864,410	262,793
1901...	44,723	62	62,916	97,037	57,571	748,802	8,076,575	303,651
1902...	39,920	51	56,244	100,546	52,664	691,680	7,888,218	234,911
1903...	40,239	100	65,004	110,181	50,371	754,957	8,027,347	280,395
1904...	43,110	36	73,379	127,984	57,143	815,570	7,588,555	353,695
1905...	52,830	46	87,714	137,125	61,637	903,156	8,035,718	287,790

Year.	Fertilizers.	Fluorspar.	Glass. (All kinds.)	Copper. Ore.	Copper. Crude and Old.	Copper. Bars, Sheets Plates, etc.	Copper Sulphate.	Copperas.	Cryolite.
1895.....	.....	44	......	17	151	354	162	301	11
1896.....	.....	40	......	12	228	189	47	392	2
1897.....	.....	27	......	0.1	159	180	14	648	10
1898.....	.....	22	......	12	173	266	29	539	23
1899.....	5,468	309	55,649	74	534	298	67	808	101
1900.....	5,385	45	57,281	801	471	200	57	748	237
1901.....	5,628	6	55,409	1,042	435	334	23	548	231
1902.....	5,554	42	56,125	1,018	436	381	44	857	363
1903.....	10,850	12	61,210	1,308	1,226	451	45	898	521
1904.....	10,147	36	63,643	574	747	577	50	1,170	574
1905.....	3,717	5	72,360	2,328	1,253	746	49	8.36	638

Year.	Gold. Ore.	Gold. Bullion.(e)	Gold. Old and Dross.(e)	Gold. Specie.(e)	Graphite.	Gypsum. Crude.	Gypsum. Calcined.	Hydrochloric Acid.
1895	1	$203,352	$236,487	$ 8,885,815	11,923	1,496	1,439	1,460
1896	45	253,194	235,935	13,555,706	13,091	899	1,376	1,246
1897	37	158,827	357,798	18,598,931	14,229	662	1,804	1,439
1898	13	17,943	377,336	23,779,858	17,109	718	2,163	1,614
1899	67	17,864	313,432	12,711,454	19,451	634	1,539	1,495
1900	1.2	120,988	316,883	11,582,571	18,995	502	1,723	1,659
1901	0.2	42,427	352,408	6,880,888	14,900	461	1,206	1,632
1902	..	22,939	359,107	13,485,087	16,771	550	1,041	791
1903	3.7	10,150	368,445	11,052,944	17,302	342	1,510	3,530
1904	64.1	5,278	370,272	9,649,605	17,430	392	1,510	3,722
1905	1059.9	9,338	311,605	10,995,089	18,535	363	1,652	4,085

Year.	Iron. Ore.	Iron. Pig and Old.	Iron. Manufactures.	Iron. Iron and Steel. Bars, Sheets, Wire, etc.	Lime. Hydraulic and Caustic.	Magnesium. Chloride and Glauber Salts.	Magnesite (Calcined.)
1895	165,402	9,786	18,698	9,993	34,698	661	(c)
1896	214,390	11,712	17,674	12,428	76,895	2,291	(c)
1897	247,856	12,084	21,064	17,387	83,110	6,910	(c)
1898	302,317	15,803	22,724	23,231	89,067	7,248	(c)
1899	326,951	27,738	30,822	50,197	85,570	5,721	(c)
1900	263,421	53,426	40,344	65,019	86,273	7,321	(c)
1901	229,624	26,304	46,508	28,841	82,399	7,960	40,236
1902	241,806	42,592	30,137	45,517	81,634	5,333	53,467
1903	252,520	60,237	40,807	63,031	95,644	2,360	69,058
1904	295,017	66,442	60,252	64,698	101,753	2,151	53,781
1905	373,077	63,780	63,828	69,672	94,751	1,272	92,359

Year.	Lead. Ore.	Lead. Ash.	Lead. Litharge.	Lead. Metal and Alloys.	Lead. Red and Yellow	Lead. White.	Manganese Ore.	Millstones.	Mineral Paints.	Nickel Bars, Sheets, Wire, etc.
1895	3,758	118	782	208	24	233	425	1,977	2,244	54
1896	3,076	113	597	272	33	171	701	1,831	1,700	273
1897	2,138	114	355	241	24	47	622	1,773	1,621	170
1898	2,253	100	188	545	45	55	1,961	2,109	2,153	76
1899	2,502	99	188	258	45	41	1,127	1,904	2,061	38
1900	2,628	66	242	393	21	34	463	1,871	1,906	119
1901	4,143	112	179	68	32	23	398	1,971	1,947	97
1902	5,478	154	124	109	25	37	411	1,886	2,136	75
1903	8,961	147	145	152	19	25	724	2,311	1,873	124
1904	7,575	144	167	464	54	52	1,234	2,276	1,840	36
1905	7,944	342	141	957	60	39	995	2,232	2,091	89

Year.	Nickel and Cobalt Ores.	Nitric Acid.	Oxide. (Potash)	Ozocerite.	Peat and Peat Coke.	Petroleum Products. Petroleum. (d)	Petroleum Products. Benzine.	Petroleum Products. Paraffin.	Potassium Chloride.	Pyrites.
1895	139	418	5,665	5,054	3,753	5,317	....	....	1,074	383
1896	113	360	4,164	5,722	2,701	24,921	....	....	1,026	341
1897	117	310	5,997	5,153	1,655	14,682	....	....	1,005	255
1898	121	294	7,252	4,462	3,400	4,138	....	....	994	3,039
1899	75	420	10,113	5,441	4,010	11,756	20,646	10	974	5,201
1900	114	519	7,792	5,162	5,607	33,032	18,361	26	879	17,162
1901	120	632	4,234	2,717	4,558	19,804	17,021	14	909	16,491
1902	34	769	3,229	2,285	4,927	40,683	13,884	24	772	9,547
1903	12	908	3,409	2,258	3,638	74,454	14,000	1,153	802	10,857
1904	26	858	4,604	2,093	3,980	122,419	13,706	5,992	445	9,891
1905	16	1,377	5,511	1,614	3,746	200,736	8,187	8,996	1,048	9,168

Year.	Stone.					Sulphur.	Sulphuric Acid.
	Lithographic.	Limestone.	Marble.	Paving.	Not Elsewhere Specified.		
1895..........	4	33,479	2,737	29,392	241,040	989	6,466
1896..........	6	23,249	3,595	30,577	216,666	1,231	6,212
1897..........	22	11,373	2,994	38,256	223,321	947	7,903
1898..........	8	25,117	2,954	54,953	252,229	923	9,880
1899..........	11	11,053	4,096	67,212	239,819	885	12,422
1900..........	7	13,878	3,811	58,121	197,932	1,285	12,693
1901..........	2	23,909	4,394	51,613	200,461	1,225	10,373
1902..........	4	29,047	4,367	33.689	230,301	1,136	9,451
1903..........	2	26,951	5,628	39,388	229,060	1,123	8,369
1904..........	6	20,331	6,220	52,720	226,874	988	9,101
1905..........	4	32,610	3,883	44,359	248,115	859	12,823

Year.	Tin.			Whet-stones.	Zinc.				
	Ingot and Old.	Bars, Plates, Sheets, etc.	Ash.		Ores.	Crude and Old.	Sheets, etc.	White.	Ash.
1895......	53	90	248	2,169	7,491	504	1,158	1,688	179
1896......	130	78	281	2,035	9,453	1,256	1,139	1,825	277
1897......	87	75	306	2,323	12,914	770	993	1,673	197
1898......	96	72	324	2,316	14,065	1,184	757	1,240	298
1899......	167	77	273	2,215	20,461	1,614	1,313	1,096	73
1900......	153	102	208	2,270	20,379	1,088	502	1,719	149
1901......	162	109	257	2,359	23,150	1,374	813	2,720	167
1902......	193	128	188	2,852	24,519	2,002	1,127	3,113	237
1903......	292	111	158	2,569	15,108	4,420	729	3,446	267
1904......	126	102	123	2,159	17,314	4.606	532	3,666	158
1905......	197	94	78	2,355	19,602	5,023	498	3,861	113

(*a*) From *Statistik des Auswaertigen Handels des Oesterreichisch-Ungarischen Zollgebiets.* (*b*) Includes artificial barium sulphate. (*c*) Previous to 1901, magnesite was included with other minerals not elsewhere specified. (*d*) From 1895 to 1898 inclusive, includes crude and refined petroleum; from 1899 to 1905 inclusive, lubricating oil is also included. (*e*) Where gold or silver values are reported 1 crown=$0.203.

BELGIUM.

The production of mining and metallurgical products in Belgium, and the imports and exports, are reported as follows:

MINERAL, METALLURGICAL AND QUARRY PRODUCTION OF BELGIUM. (a)
(In metric tons except where otherwise noted.)

Year.	Barytes.	Chalk, Marl. *Cubic Meters.*	Clay.	Coal.		Coke.	Flint. *Cubic Meters.*		Iron Ore.
				Bituminous.	Briquets.		For Earthenware.	For Ballast. (c)	
1896....	25,000	191,100	83,020	21,252,370	1,213,760	2,004,430	23,450	244,050	307,031
1897....	23,000	204,600	270,715	21,492,446	1,245,114	2,207,840	23,050	235,495	240,774
1898....	21,700	287,805	287,805	22,088,335	1,351,884	2,161,162	22,150	360,960	217,370
1899..	25,900	351,800	291,125	22,072,068	1,276,050	2,304,607	25,185	258,835	201,445
1900....	38,800	377,550	313,205	23,462,817	1,395,910	2,434,678	25,700	263,850	247,890
1901....	22,800	449,000	298,340	22,213,410	1,587,800	1,847,780	17,700	7,860	218,780
1902....	33,000	390,700	299,820	22,877,470	1,616,520	2,048,070	17,430	7,705	166,480
1903....	21,000	501,920	292,855	23,796,680	1,686,415	2,203,020	16,250	8,935	184,400
1904....	60,000	450,400	347,135	22,761,430	1,735,480	2,211,820	18,070	12,500	206,730
1905....	26,000	372,000	274,550	21,775,280	1,711,920	2,238,920	12,800	26,895	176,940

Year.	Iron, Crude.					Iron, Manufactures of.			
	Forge Pig.	Foundry Pig.	Bessemer Pig.	Basic Pig.	Total Pig.	Merchant Bars.	Sheet and Plate.	Wrought	Other Mfres.
1896........	362,451	84,275	193,518	307,779	959,414	81,394	112,597	851	298,163
1897........	426,332	78,410	183,701	333,958	1,035,037	108,608	100,252	872	263,644
1898........	308,875	93,645	173,085	397,891	979,755	123,993	91,686	993	267,521
1899........	317,029	84,165	169,664	453,718	1,024,576	93,601	97,604	662	283,331
1900........	305,344	88,335	176,557	447,271	1,018,561	61,458	73,572	1,411	284,591
1901........	178,250	86,170	166,820	332,940	764,180	249,380	65,760	550	64,900
1902........	104,540	254,710	199,170	510,630	1,069,050	260,290	62,740	450	58,150
1903........	91,600	256,890	229,160	638,430	1,216,080	274,520	56,550	390	60,920
1904........	99,350	224,410	217,390	742,040	1,287,597	246,240	41,000	370	67,580
1905........	98,170	206,390	220,210	784,850	1,311,120	270,840	39,250	40	67,490

Year.	Steel.					Lead.		Manganese Ore.
	Ingots, Blooms and Billets.	Rails.	Tires.	Wrought.	Plates.	Ore.	Pig.	
1896....................	598,947	147,183	10,497	6,702	64,653	70	17,222	23,265
1897....................	616,541	136,911	10,870	23,104	64,366	108	17,023	28,372
1898....................	653,523	117,751	10,953	17,902	87,219	133	19,330	16,440
1899....................	731,249	123,119	11,212	32,180	68,051	137	15,727	12,120
1900....................	655,199	134,428	11,934	25,985	55,307	230	16,365	10,820
1901....................	529,840	132,260	12,380	3,310	83,810	220	18,760	8,510
1902....................	786,980	(e)268,220	12,790	2,910	94,360	164	73,357	14,440
1903....................	988,160	(e)351,540	17,810	2,920	118,200	90	68,700	6,100
1904....................	1,065,870	(e)266,900	23,540	4,300	149,270	91	23,470	485
1905....................	1,227,110	241,640	25,810	6,080	179,470	126	22,885	....

Year.	Mineral Paints. Ochers. *Cubic Meters.*	Phosphate of lime. *Cubic Meters.*	Pyrites.	Sand. *Cubic Meters.*	Slate. Pieces.	Silver. *Kg*	Stone. *Cubic Meters.* Dolomite.	Flagstones. *Sq. Meters.*	Freestone.
1896..	700	297,470	2,560	418,720	35,980,000	28,509	21,500	131,400	152,420
1897..	350	350,056	1,828	559,141	41,422,000	30,073	52,720	107,572	181,746
1898..	290	156,920	147	287,805	42,311,000	116,035	37,100	170,672	215,417
1899..	300	190,090	283	627,770	44,167,000	134,854	56,400	144,330	139,294
1900..	300	215,670	400	653,780	43,941,000	146,548	45,000	153,217	157,294
1901..	(*d*)2,100	(*d*) 222,520	560	626,020	39,030,000	169,450	31,500	106,470	167,310
1902..	(*d*) 200	(*d*) 135,850	710	722,775	37,120,000	212,249	39,140	101,945	185,319
1903..	(*d*) 200	(*d*) 184,120	720	724,495	38,953,000	232,740	43,600	117,165	245,184
1904..	(*d*) 450	(*d*) 202,480	1,075	807,715	41,240,000	252,920	48,600	71,630	216,717
1905..	(*d*) 300	(*d*) 193,305	976	775,385	41,435,000	201,935	78,860	83,455	325,000

Year.	Stone. *Cubic Meters.* (*Continued*). Limestone.	Limestone for Flux.	Marble.	Paving Stones. *Pieces.*	Whetstones and Hones. *Pieces.*	Zinc. Ore. (Blende)	Ore. (Calamine)	Spelter	Sheets.
1896..........	2,646,305	164,900	16,315	102,295,950	45,850	7,070	4,560	113,361	36,238
1897..........	3,010,877	225,300	17,797	95,542,700	43,150	6,804	4,150	116,067	37,011
1898..........	2,968,997	212,685	16,610	108,025,000	89,150	7,350	4,125	119,671	35,587
1899..........	3,238,875	195,505	17,740	114,103,900	82,100	5,736	3,730	122,843	34,289
1900..........	3,228,205	229,250	15,990	107,294,600	105,000	5,715	3,000	119,317	38,825
1901..........	3,751,880	193,370	15,390	110,920,000	160,150	4,445	2,200	127,170	37,380
1902..........	1,626,670	226,220	15,490	110,103,000	122,300	3,568	284	124,780	37,070
1903..........	1,580,330	210,250	16,735	111,318,000	134,620	3,565	65	131,740	42,280
1904..........	1,645,655	213,320	17,740	117,412,000	135,700	3,698	4	137,323	41,490
1905..........	1,493,745	250,500	17,254	115,440,000	154,820	3,929	....	142,555	45,320

(*a*) From *Statistique des Industries Extractives et Metallurgiques et des Appareils à vapeur en Belgique.* (*c*) Includes some gravel. (*d*) Metric tons (*e*) Includes beams.

MINERAL IMPORTS OF BELGIUM. (*a*)

(In metric tons or dollars; 5f.=$1.)

Year.	Ashes.	Cement.	Clay Products. Terra Cotta.	Common Pottery.	Coal.	Coal Briquets.	Coke.
1896.............	6,747	30,565	85,486	2,065	1,693,376	1,561	260,273
1897.............	10,870	17,681	86,493	2,115	2,017,344	632	269,606
1898.............	8,199	34,039	92,149	2,007	2,202,517	1,756	180,590
1899.............	15,818	18,649	99,156	2,856	2,844,274	10,722	296,508
1900.............	15,428	12,773	90,852	4,281	3,288,513	21,814	289,673
1901.............	14,802	13,558	81,359	2,223	2,930,874	17,160	154,247
1902.............	16,708	13,269	91,795	2,132	3,232,510	33,235	230,612
1903.............	37,178	19,698	107,457	2,095	3,554,807	43,835	308,877
1904.............	56,052	37,593	111,394	6,401	3,701,240	45,600	338,127
1905.............	39,210	34,610	105,619	5,605	4,230,313	72,643	356,136

Year.	Copper and Nickel. Crude.	Hammered, Drawn or Rolled.	Wrought	Diamonds Crude and Uncut.	Fertilizers (All).	Glass and Glassware. Common (Bottles, Broken Glass, Etc.)	Gold (Including Platinum). All Other Kinds.	Ore. *Kg.*	Unwr'ght. *Kg.*	Specie.	Jewelry.
1896....	15,506	1,109	$188,931	....	(*c*) 25,946	6,980	$541,225	93	4,923	$ 599,540	$ 757,507
1897....	14,821	1,418	193,242	....	(*c*) 5,162	4,699	664,176	......	3,824	1,726,700	701,535
1898....	14,947	1,821	205,705	....	(*c*) 10,657	4,247	635,021	8,390	1,282	372,000	840,593
1899....	8,327	2,174	226,853	....	(*c*) 15,072	3,757	651,208	51	1,136	744,000	965,170
1900....	13,768	2,087	231,800	$8,051,200	163,229	5,671	663,400	1,250	1,728	459,420	921,600
1901....	11,381	1,780	272,600	8,463,200	149,984	5,446	623,200	......	......	....	1,332,800
1902....	14,197	1,998	251,400	8,537,600	158,924	6,129	539,600	....	......	....	1,221,800
1903....	13,602	2,035	313,400	15,557,800	188,389	5,209	557,200	....	......	....	1,190,000
1904...	13,422	2,267	490,400	16,059,400	157,402	5,267	614,600	8,516	2,385	256,680	1,009,843
1905...	12,379	2,421	439,017	18,674,270	174,435	4,287	7,200	49	1,483	571,600	1,502,107

Year.	Iron.				Steel.		Lead.	
	Ore.	Pig and Scrap.	Manufactures.	Tin Plate.	Ingots, Blooms, and Billets.	Manufactures.	Pig.	Manufactures.
1896	2,069,676	378,191	28,930	3,203	28,435	16,199	35,221	$ 17,231
1897	2,544,377	354,178	32,390	3,875	25,370	27,016	43,840	91,580
1898	2,252,553	370,117	40,388	3,848	25,142	25,774	54,867	50,728
1899	2,621,152	423,968	41,894	3,900	11,666	34,622	60,649	191,508
1900	2,528,615	371,726	40,716	5,036	19,705	43,875	58,141	143,000
1901	1,768,441	222,230	58,020	(*d*)4,705	68,228	51,427	54,720	123,200
1902	2,550,347	348,337	43,625	(*d*)6,608	103,286	43,039	71,085	164,800
1903	3,054,808	387,884	52,305	(*d*)9,776	144,370	56,139	63,386	126,600
1904	3,359,430	388,732	42,662	(*d*)7,892	182,336	49,464	63,813	234,400
1905	3,382,832	558,414	46,393	(*d*)7,426	167,513	62,977	61,668	397,400

Year.	Lime.	Petroleum.		Resins and Bitumens, Not Specified.	Salt.	
		Crude.	Refined.		Crude.	Refined.
1896	11,522	95	158,979	216,278	92,408	38,785
1897	13,184	988	149,501	237,570	96,805	39,193
1898	12,674	382	161,281	269,914	92,300	50,136
1899	12,811	2,479	166,404	264,718	81,324	50,647
1900	11,448	1,751	158,064	253,788	97,812	59,375
1901	11,288	305	160,327	244,866	93,043	48,974
1902	13,820	247	183,592	305,597	102,110	50,704
1903	13,525	237	210,905	285,305	108,886	52,272
1904	16,896	1,893	192,805	305,136	110,516	54,615
1905	17,220	143	194,584	324,298	111,317	59,575

Year.	Silver.				Sodium Salts.		
	Ore.	Bullion–*Kg.*	Specie.	Jewelry.	Carbonate.	Nitrate.	Sulphate and Sulphite.
1896	1,477	8,980	$ 6,461,840	$415,967	(*e*)	194,202	(*e*)
1897	2,533	467,851	2,083,040	460,244	(*e*)	181,676	(*e*)
1898	461	299,369	7,655,200	449,244	(*e*)	152,164	(*e*)
1899	2,523	105,723	14,272,080	520,070	(*e*)	249,756	(*e*)
1900	922	11,366	7,324,560	513,800	20,603	153,318	49,020
1901	....	....	....	550,000	19,465	167,489	47,558
1902	....	....	....	546,200	13,584	135,937	60,721
1903	....	....	....	597,800	19,459	149,942	54,926
1904	....	....	....	635,800	28,006	157,005	44,122
1905	499	12,876	4,916,480	388,971	26,525	202,241	50.667

Year.	Stone.					Sulphur.	Tin.		Zinc.	
	Roofing Slate. *1000 Pieces.*	Building Stone, Including Marble and Alabaster.	Cut, Polished, Etc.	Paving.	All Other Kinds.		Block	Manufactures.	Spelter.	Manufactures.
1896	38,209	40,511	$ 61,769	6,163	81,360	14,399	4,617	$2,845	20,182	$11,230
1897	38,754	47,929	111,313	13,197	182,950	13,261	1,609	2,634	16,320	10,661
1898	38,216	45,544	63,665	8,926	239,281	13,322	1,208	2,545	17,441	11,575
1899	35,888	49,498	83,372	7,835	216,231	8,449	1,113	4,373	11,058	11,436
1900	34,331	60,057	145,800	3,761	135,245	17,516	1,653	5,540	11,478	12,000
1901	35,516	65,509	120,800	5,367	81,006	14,775	1,841	4,000	13,896	11,200
1902	36,950	69,745	109,000	7,124	72,384	12,367	1,416	4,800	17,830	14,600
1903	34,068	83,165	153,800	5,371	103,239	21,637	2,677	4,000	20,586	15,000
1904	34,139	86,013	143,800	4,545	115,275	24,788	3,416	4,200	17,424	13,800
1905	33,501	100,856	197,196	15,759	116,119	19,617	3,270	3,922	15,692	14,599

(*a*) From *Statistique de la Belgique; Tableau General du Commerce avec les Pays Etrangers, Brussels.* (*b*) Pieces. (*c* Guano only is reported. (*d*) Includes wrought tin-plate. (*e*) Included under nitrate.

MINERAL EXPORTS OF BELGIUM. (*a*)
(In metric tons or dollars; 5 fr.=$1.)

Year.	Ashes.	Cement.	Clay Products.		Coal.	Coal Briquets.	Coke.	Copper and Nickel.		
			Terra-cotta.	Common Pottery.				Crude.	Hammered Drawn or Rolled.	Wrought.
1896	1,084	277,615	302,526	2,628	4,649,799	459,974	863,067	11,700	2,073	$168,524
1897	2,675	322,024	294,815	3,197	4,448,544	615,074	909,486	9,994	1,996	198,665
1898	615	419,132	247,970	3,186	4,579,955	666,265	878,435	8,511	1,770	161,726
1899	400	445,602	328,733	3,216	4,568,938	525,625	1,008,470	4,665	2,111	261,060
1900	2,148	408,284	289,759	5,915	5,260,993	604,864	1,073,315	8,411	2,097	276,600
1901	3,401	492,882	220,326	6,661	4,820,300	714,455	829,421	6,309	1,988	198,200
1902	4,438	542,547	256,555	3,815	5,078,278	671,700	824,256	7,320	1,656	235,400
1903	3,202	599,092	342,149	5,826	4,923,368	623,691	841,142	7,638	1,702	256,400
1904	864	588,295	329,616	3,450	5,067,037	539,364	879,883	7,255	2,243	182,200
1905	3,713	679,426	249,707	3,047	4,704,063	480,247	977,095	6,805	2,056	315,648

Year.	Diamonds. Crude and Uncut.	Fertilizers. (All Kinds.)	Glass and Glassware.			Gold (Including Platinum).		
			Common (Bottles, Broken Glass, etc.)	Plate.	All Other Kinds.	Unwrought, *Kg.*	Specie.	Jewelry.
1896.......	....	(*c*)14,633	3,647	$3,468,668	178,611	3,713	$2,666,620	$ 75,570
1897.......	....	(*c*)14,044	3,546	3,761,219	174,232	2,547	605,120	126,385
1898.......	....	(*c*)21,626	2,747	3,007,865	25,226	1,231	578,120	118,897
1899.......	....	(*c*)18,213	3,850	4,942,298	196,842	504	998,200	118,916
1900.......	$ 8,601,000	258,366	3,993	4,413,000	159,557	549	613,180	161,200
1901.......	8,814,200	299,709	5,486	5,030,400	154,610	....	....	225,800
1902.......	8,855,400	359,607	6,338	4,799,800	198,647	....	....	193,800
1903.......	16,708,200	497,355	5,156	4,660,800	197,246	....	....	159,400
1904.......	17,028,400	555,956	6,781	4,707,200	146,637	671	726,020	137,400
1905.......	18,674,270	511,395	6,183	5,410,600	182,314	608	1,860,000	210,600

Year.	Iron.				Steel.		Lead.		Lime.
	Ore.	Pig and Scrap. (*d*)	Manufactures.	Tin Plate.	Ingots, Blooms and Billets.	Manufactures.	Pig.	Manufactures.	
1896..	389,235	63,906	$12,569,898	3,952	1,145	218,179	31,366	$36,821	477,213
1897..	410,817	60,678	13,599,204	1,191	1,201	220,739	35,988	33,286	520,588
1898..	384,047	40,522	9,139,480	973	1,018	208,901	40,302	16,450	546,199
1899..	318,415	76,047	15,834,350	1,436	1,259	187,009	41,618	37,022	537,357
1900..	420,180	79,172	15,543,000	940	975	174,998	46,566	9,000	617,666
1901..	327,499	70,027	14,083,400	(*e*) 642	290	199,387	47,971	6,600	579,123
1902..	368,560	96,302	15,192,400	(*e*)1,023	1,463	310,802	58,495	9,400	623,617
1903..	400,972	101,549	16,059,000	(*e*)3,577	3,047	403,588	7,765	8,200	659,125
1904..	441,059	85,489	17,302,800	(*e*)2,378	5,250	331,755	59,334	13,400	706,351
1905..	443,511	80,896	20,727,608	(*e*)1,322	19,075	356,505	53,378	53,669	720,123

Year.	Petroleum.		Resins and Bitumens, not specified.	Salt.		Silver.			
	Crude.	Refined.		Crude.	Refined.	Ore.	Bullion, *Kg.*	Specie.	Jewelry, etc.
1896............	2	29,321	86,906	1,434	129	19	40,118	$ 667,840	$137,137
1897............	1	18,088	92,591	493	231	423	57,933	20,851,320	189,959
1898............	782	19,556	107,806	298	386	......	107,385	13,083,640	127,194
1899............	2,146	25,970	112,392	506	885	......	54,358	13,483,160	214,175
1900............	1,759	21,812	97,970	2,345	799	......	38,331	1,304,840	103,200
1901............	*Nil.*	22,091	86,908	1,611	2,454	......	....	....	85,000
1902............	*Nil.*	23,344	101,950	1,378	1 077	......	....	....	88,000
1903............	*Nil.*	29,155	93,533	928	618	......	....	....	94,400
1904............	*Nil.*	30,209	97,462	1,899	1,955	......	....	6,870,000	108,800
1905............	*Nil.*	36,491	115,212	305	2,186	1,789	44 682	723,720	99,971

Year.	Sodium Salts.			Stone.		
	Carbonate.	Nitrate.	Sulphate and Sulphite.	Roofing Slate, 1000 *Pieces.*	Building Stone Including Marble and Alabaster.	Cut, Polished, etc.
1896	(*f*)	42,857	(*f*)	15,435	161,298	922,147
1897	(*f*)	9,054	(*f*)	17,304	187,180	934,286
1898	(*f*)	106,252	(*f*)	16,948	178,249	861,015
1899	(*f*)	109,253	(*f*)	15,316	164,952	948,090
1900	25,569	39,346	10,577	12,836	171,126	872,400
1901	22,520	44,303	15,833	12,947	168,824	735,800
1902	12,831	33,319	14,316	13,715	163,210	848,800
1903	9,454	36,057	17,434	11,984	160,874	887,800
1904	15,049	36,970	18,715	11,362	160,421	822,800
1905	16,571	58,815	15,367	11,934	147,934	867,905

Year.	Stone. (*Continued.*)		Sulphur.	Tin.		Zinc.	
	Paving.	All Other Kinds.		Block.	Manufactures.	Spelter.	Manufactures.
1896	154,737	796,231	5,335	1,055	$ 872	100,369	$ 56,349
1897	153,504	773,531	6,041	347	839	100,228	90,749
1898	159,455	917,654	6,355	508	2,394	108,507	102,583
1899	150,993	834,528	6,769	659	1,511	101,244	109,762
1900	178,057	1,022,781	7,363	495	3,000	99,233	98,400
1901	148,176	729,196	6,722	299	1,200	106,656	98,600
1902	145,019	685,567	7,349	234	800	118,118	95,800
1903	128,715	696,338	10,324	838	1,000	119,988	156,200
1904	119,557	964,255	9,020	815	800	116,289	165,800
1905	118,185	916,504	5,925	201	583	125,423	169,910

(*a*) From *Statistique de la Belgique; Tableau General du Commerce avec les Pays Etrangers.* (*c*) Guano only is reported. (*d*) Includes iron and steel filings. (*e*) Includes wrought tin plate. (*f*) Included under nitrate.

BOLIVIA.

Tin mining is the principal of the mineral industries of Bolivia and in spite of adverse transportation facilities, inadequate labor and an additional export duty of one boliviano ($0.352 U. S. currency) per quintal, the output has been increasing. Silver is produced almost exclusively from the mines at Huanchaca, San José, and Oruro and has been gradually falling off. Gold mining is practically at a standstill, the only washings reported being those at Chuquiaguillo near La Paz. Bismuth is mined at Chorolque, under the control of the European combination. The production of minerals and metals for the years 1903, 1904 and 1905 follow. Later returns are not yet available.

MINERAL AND METAL PRODUCTION OF BOLIVIA (*a*)
(In metric tons.)

Year.	Antimony. Ore.	Bismuth.	Borate of Lime.	Cobalt. Ore.	Copper. (*c*)	Gold. (*b*)	Silver. (*d*)	Tin. Ore.	Tungsten. Ore.
1903......	59	288	1,197	3.8	4,093	$33,810	39,063	18,425	68
1904......	7	406	1,080	1.5	3,228	17,130	21,172	20,692	700
1905......	17	592	2,146	....	6,708	15,044	8,266	26,428	68

(*a*) From a British Consular report.
(*b*) Reduced to U. S. currency.
(*c*) Includes ingots, precipitate, matte and ore.
(*d*) Includes ingots, ore and sulphide.

Ore is still largely transported on the backs of llamas or burros at a cost of about 10c. per 100 lb. per mile. In former years it cost from 4 to 6c., but insufficient pasturage caused by decreased rainfall has affected the maintenance of animals.

BRAZIL.

No statistics of mineral production in this Republic are reported officially, but its output is fairly represented by the exports, which are given in the accompanying table. Formerly, diamond mining was the most important feature of the mineral industry in Brazil, but since the discovery of diamonds in South Africa, the production of diamonds in Brazil has dwindled to practical insignificance, except for the black diamonds, or carbonado. However, Brazil is the largest producer of monazite sand, and is a considerable producer of manganese ore and of gold.

MINERAL EXPORTS OF BRAZIL. (*a*)
(In metric tons or dollars.)

Year	Agate.	Carbonado.	Copper Ore.	Diamonds.	Gold.	Manganese Ore.	Mica and Talc.	Monazite.	Platinum (grams).	Precious Stones (b)	Rock Crystal.
1902	81	$206,135	234	$328,540	$2,645,655	157,295	11	1,205	....	$18,000	....
1903	74	276,510	316	257,335	2,828,965	161,926	7	3,299	1,315	34,065	....
1904	54	132,935	610	145,005	2,534,025	208,260	14	4,860	2,122	51,845	35
1905	......	356,784	658	449,172	2 041,822	224,377	1	4,437	72,000	278,925	23
1906	......	992,164	1,484	1,055,444	2,394,308	121,331	6	4,351	....	438,750	37

(*a*) As reported by the *Brazilian Review*. (*b*) Other than carbonado and diamonds.

CANADA.

The mineral statistics of the Dominion of Canada as reported by the Geological Survey are summarized in the following tables:

MINERAL PRODUCTION OF THE DOMINION OF CANADA. (*a*)
(In metric tons or dollars.)

Year.	Arsenic.	Asbestos and Asbestic.	Barytes.	Cement—*Barrels.*		Chromite.	Clay Products.		
				Natural Rock.	Portland.		Fire Clay.	Pottery.	Terra-Cotta.
1896......	*Nil.*	11,113	132	70,705	78,385	2,124	764	$163,427	$83,855
1897	*Nil.*	27,617	518	85,450	119,763	2,392	1,921	129,629	155,595
1898......	*Nil.*	21,577	971	87,125	163,084	1,833	608	135,000	167,902
1899......	52	22,938	653	131,387	255,366	1,796	543	185,000	220,258
1900.. ...	275	27,797	1,213	125,428	292,124	2,335	1,129	200,000	259,450
1901......	630	36,477	592	133,328	317,066	1,274	3,609	200,000	278,671
1902......	726	36,657	994	127,931	594,594	900	2,486	200,000	276,241
1903......	725	37,902	1,055	92,252	627,741	3,509	2,394	200,000	405,796
1904	(*d*) 66	44,131	1,253	51,555	771,650	5,511	(*b*)	200,000	400,000
1905	*Nil.*	61,928	3,049	14,184	1,346,547	7,781	(*b*)	(*b*)	64,892
1906(*g*)...	(*b*)	72,025	3,628	8,610	2,139,164	7,936	(*b*)	(*b*)	(*b*)

Year.	Clay Products (*Continued*)	Coal.	Coke.	Copper. In Ore, etc.	Corundum.	Feldspar.	Gold. (*e*)	Graphite.	Grind-stones.
	Tiles and Sewer Pipe.								
1896...	$378,875	3,398,091	45,004	4,260	(*b*)	882	$2,754,774	126	3,368
1897...	389,250	3,434,756	55,042	6,032	(*b*)	1,270	6,027,016	395	4,147
1898...	406,717	3,784,532	79,453	8,048	(*b*)	2,268	13,775,420	(*b*)	4,476
1899...	386,546	4,467,021	91,444	6,838	(*b*)	2,721	21,261,584	1,188	4,091
1900...	456,525	5,087,060	142,521	8,588	3	288	27,908,153	1,743	5,024
1901...	498,115	5,648,208	331,537	17,155	403	4,852	24,128,503	2,004	4,155
1902...	551,965	6,524,180	455,353	17,598	697	6,871	21,336,667	993	5,835
1903...	592,970	6,933,107	509,115	19,357	880	12,633	18,843,590	660	5,023
1904...	653,894	6,812,834	493,107	19,497	834	10,057	16,400,000	410	4,091
1905...	(*f*)382,000	7,961,397	622,154	21,596	1,492	10,617	14,486,833	491	4,693
1906(*g*)	(*f*)446,790	9,033,973	(*b*)	25,863	2,063	14,397	12,023,932	405	5,029

Year.	Gypsum.	Iron Ore.	Iron, Pig. All kinds.	Iron and Steel, Rolled.	Lead (In ore, etc.)	Lime.	Limestone for Flux.	Mangan-ese Ore.	Mica.
1896.......	187,778	83,359	61,012	76,244	10,975	$650,000	33,978	(*d*)112	$60,000
1897.......	217,340	45,989	52,612	78,253	17,695	650,000	28,365	(*d*) 14	76,000
1898	198,864	52,917	69,853	101,748	14,469	650,000	30,759	45	118,375
1899.......	221,821	67,678	93,367	112,412	9,914	800,000	47,006	1,434	163,000
1900.......	228,656	110,654	87,594	102,301	28,648	800,000	48,040	27	166,000
1901.......	266,476	284,477	248,859	113,799	23,537	830,000	153,645	(*d*) 399	160,000
1902.......	301,165	366,431	324,617	164,069	10,411	892,000	266,290	(*d*) 156	135,904
1903.......	285,242	239,715	270,182	131,588	8,226	860,000	251,649	83	177,857
1904.......	309,133	317,387	275,367	(*b*)	17,241	(*h*)	182,023	(*d*)112	152,919
1905.......	395,341	263,113	476,453	(*b*)	25,391	(*h*)	309,907	(*d*) 20	168,170
1906(*g*) ...	378,904	269,842	452,769	(*b*)	24,580	(*h*)	331,976	(*d*) 84	(*d*) 581,043

Year.	Mineral Paints. (Ochers)	NaturalGas.	Nickel. (In ore, etc.)	Petroleum, Crude. *Barrels.* (*e*.)	Phosphate (Apatite).	Platinum.	Pyrites.	Salt.	Sand and Gravel Exports.
1896......	2,142	$276,301	1,541	726,822	517	$750	30,580	39,872	203,865
1897......	3,542	325,873	1,813	709,857	824	1,600	35,291	46,574	138,737
1898......	2,019	322,123	2,502	758,391	665	1,500	29,223	51,828	150,520
1899......	3,555	387,271	2,605	808,570	2,721	825	5,112	53,820	219,902
1900......	1,783	417,094	3,211	710,498	1,283	*Nil.*	6,308	56,284	179,185
1901......	2,025	339,476	4,167	623,392	937	457	1,982	53,901	178,953
1902......	4,494	195,992	4,849	530,624	776	190	2,304	58,462	144,932
1903......	5,683	202,210	5,671	486,637	1,205	(*b*)	30,822	56,644	322,703
1904......	3,562	247,370	4,786	552,575	832	(*b*)	29,980	62,411	362,701
1905......	4,632	314,249	8,565	634,095	1.180	(*b*)	29,713	41,159	332,879
1906(*g*)...	6,201	528,868	9,745	569,753	(*b*)	(*b*)	35,927	69,283	232,691

Year.	Sand (Molding).	Silver—*Kg.* (In ore, etc.)	Slate.	Soapstone and Talc.	Stone.			
					Building Bricks.	Building Stones.	Flagstones.	Granite.
1896.........	5,205	99,699	$53,370	372	$1,600,000	$1,000,000	$6,710	$106,709
1897.........	4,975	172,891	42,800	142	1,600,000	1,000,000	7,190	61,934
1898.........	9,589	138,486	40,791	367	1,900,000	1,300,000	4,250	81,073
1899.........	12,448	106,116	33,406	408	2,195,000	1,500,000	7,600	90,542
1900.........	5,606	138,980	12,100	1,288	2,275,000	1,520,000	5,250	80,000
1901.........	13,337	172,292	9,980	235	2,400,000	1,650,000	4,575	155,000
1902.........	12,110	133,478	19,200	625	2,593,000	1,900,000	7,760	210,000
1903.........	3,318	99,489	22,040	898	2,832,000	1,975,000	6,688	200,000
1904.........	3,105	115,666	23,247	762	(*h*)	5,667,000	6,720	100,000
1905.........	(*b*)	185,839	21,568	454	(*h*)	6,095,000	7,650	209,955
1906(*g*)......	(*b*)	266,521	24,446	1,119	(*h*)	7,200,000	(*b*)	(*b*)

(*a*) From *Reports Compiled by the Geological Survey of Canada.* (*b*) Not reported. (*c*) Gold values are calculated at the rate of $20.67 per oz. (*d*) Export. (*e*) One barrel contains 35 imp. gal. (*f*) Sewer pipe only reported. (*g*) From preliminary unrevised reports. (*h*) Included under building stone.

MINERAL IMPORTS OF THE DOMINION OF CANADA. (*a*.)

(In metric tons or dollars.)

Year. (*b*)	Abrasives.				Aluminum.		Antimony. (*c*)	Arsenic.	Asbestos. (*d*)
	Buhrstones Number.	Emery, (Wheels and Bulk)	Grindstones.	Pumicestone.	Manufactures.	Ingots, Sheets, Etc.			
1897............	1,499	$11,231	$25,547	$2,903	$5,717	....	61	68	$19,032
1898............	889	15,478	22,217	3,829	7,102	....	71	132	26,389
1899...........	1,116	22,343	27,476	5,973	9,275	....	131	264	32,607
1900...........	1,250	44,882	34,382	5,604	12,543	....	90	105	43,455
1901...........	3,641	39,116	39,068	5,516	16,202	....	159	72	50,829
1902...........	1,854	23,946	40,838	7,254	30,496	....	229	48	52,464
1903...........	35	40,235	53,388	6,152	14,201	$ 13,930	393	135	75,465
1904...........	2	50,899	46,039	6,537	16,065	101,427	190	188	83,827
1905...........	1,931	55,230	49,247	8,447	28,418	154,569	85	122	116,836
1906...........	1,746	63,876	59,627	9,053	23,565	168,405	183	202	138,000

Year.	Asphalt.	Brass. (*e*)	Cement.	Chalk and Whiting.	Clay Products.		
					Bricks and Tiles.	Clays.	Earthenware and China.
1897...........	342	$ 457,342	$ 260,842	$29,973	$ 44,622	$ 59,386	$ 595,822
1898...........	6,006	560,014	365,624	35,099	36,263	72,795	675,874
1899...........	8,196	747,557	477,617	44,771	55,204	88,517	916,727
1900...........	2,825	853,599	513,770	46,787	58,454	122,965	959,526
1901...........	2,849	985,776	666,350	72,507	76,760	141,251	1,114,677
1902...........	3,426	1,017,294	863,646	53,473	89,332	140,521	1,275,093
1903...........	3,037	1,197,012	890,745	56,364	157,783	176,416	1,406,610
1904...........	7,093	1,257,117	1,014,713	65,840	259,421	144,706	1,611,356
1905...........	5,096	1,375,907	1,263,828	77,387	369,561	197,609	1,636,214
1906...........	7,178	1,786,596	1,003,022	77,750	460,447	220,504	1,674,817

Year.	Coal. Anthracite (f)	Coal. Bituminous. (f)	Coal Tar. *Barrels.*	Coke.	Copper. Ingots, Pig and Scrap.	Copper. Manufactures.	Copper. Sulphate	Explosives.	Flint and Stones.
1897	1,321,767	1,604,517	23,661	75,580	22	$ 264,587	516	$131,562	475
1898	1,324,856	1,735,576	26,702	122 499	476	786,529	738	141,731	389
1899	1,583,132	2,220,250	39,296	128,145	751	551,586	726	212,968	243
1900	1,500,542	2,512,334	50,484	170,405	519	1,090,280	752	247,511	280
1901	1,753,488	2,658,257	54,928	280,069	432	951,045	673	306,067	222
1902	1,498,773	3,208,005	55,376	242,298	801	1,281,522	711	423,982	186
1903	1,320,239	3,684,502	29,325	232,848	924	1,291,635	1,010	347,020	403
1904	2,064,444	4,230,436	55,172	200,590	960	1,094,183	795	418,916	554
1905	2,361,952	4,377,667	77,856	337,035	882	1,666,955	934	369,311	844
1906	1,996,183	5,003,029	66,540	435,561	1,191	2,660,882	844	647,272	1,410

Year	Fuller's Earth.	Glass.	Gold and Silver. Coin and Bullion. (g)	Gold and Silver. Manufactures.	Gravel and Sand.	Graphite. Crude	Graphite. Manufactures. (h)
1897	$1,552	$1,139,764	$ 4,676,094	$296,143	19,330	$1,406	$38,537
1898	3,330	1,024,706	4,390,844	297,242	29,164	1,862	52,291
1899	3,418	1,343,058	4,705,134	342,320	27,477	4,979	57,824
1900	2,661	1,658,694	8,297,438	339,145	32,399	4,437	60,518
1901	3,147	1,584,922	3,537,294	367,857	35,744	2,357	75,536
1902	3,909	1,932,539	6,311,405	352,224	42,995	3,649	64,123
1903	4,169	2,084,451	8,976,797	434,273	83,047	2,870	69,676
1904	5,554	1,983,781	7,874,313	441,154	100,394	1,802	67,563
1905	4,967	1,948,969	10,308,435	502,305	77,402	2,499	75,288
1906	4,644	2,671,398	7,078,603	579,378	105,666	2,791	86,028

Year.	Gypsum. Crude and Ground.	Gypsum. Plaster of Paris.	Iron and Steel. Pig and Scrap.	Iron and Steel. Slabs, Blooms, Bars, Etc.	Iron and Steel. Alloys of Iron.	Iron and Steel. Manufactures.	Kainite
1897	482	440	33,442	2,566	387	10,133,379	206
1898	1,057	150	81,577	7,391	1,287	15,458,355	49
1899	310	225	69,819	5,640	1,053	18,874,092	30
1900	72	385	94,489	11,576	1,043	27,259,134	143
1901	289	228	59,033	10,659	1,372	25,686,154	88
1902	516	215	71,882	18,208	5,910	31,257,412	85
1903	1,007	286	129,641	17,896	5,762	39,350,068	259
1904	626	291	86,087	9,088	2,700	31,014,946	339
1905	2,972	3,595	90,698	14,420	11,738	29,847,298	306
1906	5,743	6,579	112,937	29,520	13,626	35,698,630	306

Year	Lead. Pig and Scrap.	Lead. Bars and Sheets.	Lead. Litharge.	Lead. Manufactures.	Lead. Pigments and Zinc White.	Lime. Burned. *Barrels.*	Lime. Chloride of.	Mineral Paints. (Ochers)
1897	2,962	477	546	$60,735	4,678	16,108	1,361	682
1898	4,012	1,008	519	63,179	5,754	12,850	1,765	965
1899	5,202	2,032	432	91,497	6,583	15,720	1,857	1,110
1900	2,829	703	415	104,736	6,661	12,865	1,967	1,122
1901	(i)3,871	739	505	107,260	4,647	19,657	1,605	1,031
1902	(i)5,548	844	590	120,020	7,071	24,602	1,806	1,148
1903	(i)4,471	523	632	134,151	8,715	31,108	2,104	1,459
1904	4,292	......	......	133,639	7,679	54,359	2,080	1,256
1905	2,589	800	811	147,177	9,695	98,676	2,507	1,417
1906	3,751	730	461	163,666	6,947	134,334	2,645	809

Year.	Mineral Waters.	Nickel.	Petroleum Products—*Gallons.* Illuminating oil, Etc. Crude or Refined.	Petroleum Products—*Gallons.* Paraffin wax and Candles.	Platinum.	Potassium Salts. Except Saltpeter.	Potassium Salts. Saltpeter.
1897	$ 22,142	$ 4,737	8,415,302	74	$9,031	265	456
1898	33,314	5,882	9,074,311	75	9,781	244	627
1899	38,046	9,446	10,394,208	70	9,671	472	930
1900	30,343	6,988	9,633,647	35	57,910	733	602
1901	40,802	12,029	11,082,822	74	20,263	476	581
1902	91,871	15,448	13,220,005	123	19,357	771	690
1903	108,130	26,177	18,799,312	307	21,251	1,060	916
1904	136,583	14,682	24,521,115	228	28,112	1,151	898
1905	161,790	19,076	13,229,855	98	61,719	945	1,048
1906	179,837	15,976	10,981,611	375	54,494	1,317	1,141

Year.	Precious Stones and Jewelry.	Quick-Silver.	Sal-Ammoniac	Salt.	Silex.	Sodium Salts except Chloride.	Stone, Building.
1897	$ 506,728	35	69	103,337	116	13,938	$ 38,714
1898	743,607	27	38	96,962	141	16,026	28,495
1899	923,837	47	53	88,397	179	20,742	48,040
1900	732,675	39	60	92,823	182	16,748	64,533
1901	1,279,617	64	76	103,402	162	18,631	46,078
1902	1,497,321	44	78	114,629	199	17,133	99,074
1903	1,977,359	75	114	112,188	159	18,887	87,866
1904	2,075,675	69	93	103,635	252	25,118	93,778
1905	2,315,889	47	143	97,723	405	26,219	102,817
1906............	2,489,364	68	209	99,788	338	30,401	189,261

Year.	Stone. (*Continued*).				Sulphur.	Tin and Tinware.	Zinc.	
	Lithographic Stones.	Marble.	Manu-factures.	Slate.			Spelter.	Manu-factures.
1897.......	$ 6,360	$ 77,150	$ 34,026	21,615	3,932	$1,274,108	542	$ 5,145
1898.......	7,791	95,894	41,240	24,907	17,248	1,550,851	1,595	10,503
1899.......	6,223	101,879	60,148	33,100	11,121	1,372,813	852	14,661
1900.......	6,294	94,017	57,039	53,707	9,584	2,418,455	1,304	1,475
1901.......	9,584	96,159	66,639	72,187	10,827	2,339,109	931	6,882
1902.......	12,272	130,424	72,397	72,601	11,180	2,293,958	1,582	6,683
1903.......	8,461	153,481	78,629	84,437	11,077	2,712,186	1,209	9,754
1904.......	17,981	181,511	102,829	86,057	8,786	2,389,557	1,540	14,092
1905.......	13,683	145,466	150,160	93,228	10,633	2,791,757	1,721	11,912
1906.......	6,772	190,044	121,302	113,151	19,512	3,105,876	3,383	12,921

EXPORTS OF DOMESTIC MINERAL PRODUCE FROM THE DOMINION OF CANADA. (*a*).

(In metric tons or dollars.)

Year (*b*).	Antimony Ore.	Asbestos.	Clay Products.		Cement.	Chromite.	Coal.	Coke.
			Bricks. *Thousands*	Clay. Mfres. of				
1897...	....	9,954	906	$ 796	$1,332	(*k*)1,911	1,000,061	1,692
1898...	1,118	16,718	276	343	609	(*k*)1,527	981,963	3,275
1899...	....	13,176	93	339	2,789	(*k*)1,369	1,035,245	4,024
1900...	6	16,483	342	215	2,274	(*k*) 334	1,489,139	12,558
1901...	219	24,242	728	761	3,554	(*k*)2,049	1,713,737	60,129
1902...	13	30,011	669	414	1,359	(*k*) 672	1,649,278	52,873
1903...	128	27,823	2,083	109	9,735	(*b*) 658	1,796,689	39,616
1904...	87	31,444	971	36	5,467	(*b*)2,103	1,494,106	61,750
1905...	340	37,320	670	2,755	5,430	(*b*)3,702	1,465,809	116,387
1906...	388	40,367	706	....	8,913	(*b*) 1,640	1,651,203	50,004

Year.	Coin and Bullion. (*m*)	Copper (*e*)	Explosives.	Fertilizer.	Glass and Glassware	Gold. Quartz, Dust, etc.	Graphite	Grindstones
1897...	$ 327,298	4,596	$ 76,578	$ 36,584	$ 7,208	$ 2,804,101	78	$15,760
1898...	1,045,923	6,319	74,305	46,864	7,494	3,387,953	348	18,785
1899..	1,101,245	3,843	115,065	51,224	11,788	3,272,702	662	18,619
1900...	1,670,068	6,274	155,764	51,410	11,016	14,148,543	1,742	22,196
1901...	1,978,489	11,954	240,535	37,706	13,574	24,445,156	1,246	38,304
1902...	1,669,420	13,789	248,434	61,831	11,587	19,668,015	783	21,878
1903...	619,963	13,445	254,605	116,474	14,065	16,437,528	530	14,169
1904...	2,465,557	20,279	212,124	177,193	21,452	18,715,539	269	12,676
1905...	1,844,811	17,431	184,531	229,212	16,163	15,208,380	201	27,985
1906 ..	9,928,828	20,082	205,856	236,114	10,558	12,991,916	180	15,793

Year.	Gypsum.		Iron and Steel.			Lead. (*p*)	Lime.	Manga-nese Ore.
	Crude.	Ground.	Iron Ore.	Pig and Scrap.	Manufactures.			
1897 ..	163,829	$ 18,710	(*n*) 3,056	....	....	13,636	$ 56,720	74
1898 ..	163,660	2,587	(*n*) 1,975	....	....	19,944	48,307	7
1899 ..	148,565	7,611	(*n*) 2,881	$ 90,505	$ 615,906	15,445	64,112	24
1900 ..	211,792	2,622	(*n*) 5,012	411,491	1,013,672	8,998	77,325	57
1901 ..	156,080	25,472	(*n*) 54,208	256,250	1,176,711	29,747	83,439	33
1902 ..	243,629	10,150	(*n*) 478,503	1,262,285	1,198,476	13,890	111,910	500
1903 ..	271,899	7,947	(*n*) 267,000	335,958	2,927,982	7,386	127,792	137
1904 ..	247,741	10,154	(*n*) 214,309	229,824	1,761,997	7,329	104,044	62
1905 ..	290,574	2,801	204,091	172,720	950,634	23,094	75,498	84
1906 ..	367,203	1,603	134,270	346,030	1,251,289	6 158	73,534	15

Year.	Mica.	Nickel in Ore,Matte, etc.	Petroleum, Crude and Refined.	Pyrites.	Salt. *Bushels.*	Silver—*Kg.* (In Ore, Matte, Etc.)	Stone. *All Kinds.*	Tin. Mfres.
1897.......	217	3,415	1,831	14,219	4,702	127,440	23,700	$ 2,764
1898.......	231	6,697	9,530	18,752	5,559	211,012	85,671	5,578
1899.......	538	6,546	4,268	11,707	5,209	137,400	110,290	3,159
1900.......	490	6,122	6,758	13,507	15,151	71,015	217,880	3,472
1901	444	4,327	19,942	22,146	56,461	125,110	182,342	14,481
1902	452	1,762	2,478	24,088	21,778	114,610	230,798	26,524
1903	632	4,098	413	16,762	7,959	100,861	212,097	90,953
1904	393	6,456	1,208	15,582	42,662	99,472	114,192	76,796
1905.......	461	5,431	6,441	20,473	5,663	112,076	78,791	37,535
1906.......	603	10,866	1,741	18,398	23,168	203,323	.....	22,106

(*a*) From Tables of the *Trade and Navigation of the Dominion of Canada.* (*b*) Fiscal year ending June 30. (*c*) Includes regulus and salts of antimony. (*d*) Asbestos in any form except crude, and all manufactures of. (*e*) Includes manufactures. (*f*) Includes coal dust. (*g*) Coin, gold and silver, except U. S. silver coin. (*h*) Includes black lead, and crucibles (clay or graphite). (*i*) Includes Canadian lead ore refined in the United States. (*k*) Calendar year. (*l*) Fine copper contained in ore, matte, regulus, etc. (*m*) Of foreign production. (*n*) Includes chromic iron ore. (*p*) Lead contained in ore, etc.

FRANCE.

In the following tables are given the statistics of mineral and metal production in France and the French colonies—Algeria, New Caledonia and Tunis—together with the foreign commerce of France in mineral and metal products:

MINERAL AND METALLURGICAL PRODUCTION OF FRANCE. (a)
(In metric tons.)

Year.	Aluminum.	Antimony.		Arsenic Ore.	Asphaltum.	Barytes.	Bauxite.	Bitumen. (c)
		Ore.	Metal.					
1896	370	5,675	969	....	17,717	2,791	33,820	225,784
1897	470	4,685	1,033	....	17,982	3,209	41,740	233,328
1898	565	4,433	1,226	....	18,832	2,763	36,723	229,108
1899	763	7,392	1,499	2,600	22,100	4,058	48,215	258,449
1900	1,026	7,843	1,573	4,705	25,228	3,635	58,530	266,474
1901	1,200	9,867	1,786	7,491	20,391	4,145	76,620	249,655
1902	1,355	9,715	1,725	5,372	....	4,323	96,900	258,295
1903	1,570	12,380	2,748	6,658	....	5,731	133,890	243,295
1904	1,650	9,065	2,116	3,117	22,000	6,944	75,640	227,177
1905	1,905	12,543	2,396	3,627	20,000	5,504	103,207	188,403

Year.	Cement.	Clay Products.		Coal.			Copper.	
		Potter's Clay.	Fire Clay.	Bituminous.	Lignitic.	Peat.	Ore.	Metal.
1896	934,624	246,677	291,690	28,750,452	439,448	130,207	106	6,544
1897	976,813	270,292	318,185	30,337,207	460,422	98,067	956	7,376
1898	1,072,025	260,362	295,913	31,826,127	529,977	104,265	382	7,834
1899	1,144,271	310,220	367,432	32,256,148	606,564	99,230	2,021	6,640
1900	1,147,670	331,396	329,561	32,721,562	682,736	95,630	3,031	6,446
1901	1,127,206	341,407	293,208	31,633,300	691,700	118,433	3,413	7,000
1902	962,930	(h)4,541,359	295,341	29,365,047	632,423	109,941	828	6,300
1903	898,393	(h)4,734,924	253,460	34,217,661	688,757	100,348	10,892	6,921
1904	903,632	(h)4,968,936	220,409	33,502,394	665,572	95,716	2,756	6,900
1905	922,531	(h)5,129,393	215,587	35,218,000	709,000	98,500	5,068	7,576

Year.	Gold.	Gypsum.		Iron.				Lead.	
		Crude.	Calcined.	Ore.	Pig.	Wrought Iron.	Wrought Steel.	Ore. (d)	Pig. (e)
1896	$217,308	264,187	1,429,550	4,069,390	2,339,537	828,758	916,817	19,042	8,232
1897	183,416	292,753	1,369,269	4,582,236	2,484,191	584,540	994,891	21,212	9,916
1898	177,435	303,531	1,449,384	4,731,394	2,525,100	766,000	1,174,000	23,342	10,920
1899	179,429	263,879	1,372,067	4,985,702	2,578,400	834,000	1,240,000	17,505	15,981
1900	134,904	192,916	1,405,845	4,676,740	2,714,298	672,172	1,226,537	24,276	15,210
1901	85,727	355,995	1,623,710	4,260,747	2,388,823	612,362	1,175,454	20,644	21,000
1902	(b)	219,487	1,572,687	5,003,782	2,405,000	639,600	1,245,800	22,634	19,000
1903	(b)	162,766	1,468,830	6,219,541	2,840,517	598,910	1,305,709	23,080	23,258
1904	(b)	106,173	1,481,303	7,022,841	2,999,787	554,632	1,482,708	14,173	18,800
1905	8,260	78,832	1,299,313	7,395,409	3,077,000	670,000	1,442,000	12,118	24,100

Year.	Lime	Manganese Ore.	Millstones.	Mineral Paints. (Ochers.)	Nickel.	Phosphate Rock.	Pyrites.	Salt.
1896	2,224,847	31,318	28,237	27,499	1,545	582,667	282,064	1,042,614
1897	2,201,428	37,212	32,175	32,299	1,245	535,390	303,488	948,003
1898	2,339,850	31,935	(f)38,929	33,780	1,540	568,558	310,972	999,283
1899	2,343,377	39,897	41,535	32,750	1,740	645,868	318,832	1,193,532
1900	2,377,110	28,992	41,103	33,080	1,700	587,919	305,073	1,088,634
1901	2,443,062	22,304	33,286	35,704	1,800	535,676	307,447	910,009
1902	4,796,807	12,536	34,504	34,770	1,600	543,900	318,235	863,927
1903	4,727,543	11,583	35,031	34,042	1,500	475,783	322,118	967,531
1904	4,583,522	11,254	37,409	34,945	1,500	423,521	271,544	1,153,754
1905	3,694,725	6,751	33,468	37,800	1,800	476,720	267,114	1,130,088

Year.	Slate.		Stone.				Sulphur Ore. (g)	Zinc.	
	Roofing.	Slabs.	Building.	Limestone. (Flux)	Marble.	Paving Blocks.		Ore.	Metal.
1896.	283,352	1,148	10,089,845	721,296	119,168	677,213	9,720	81,346	35,585
1897.	310 820	1,143	10,105,438	709,562	118,675	568,677	10,723	83,044	38,067
1898.	311,911	1,318	9,989,416	695,501	124,161	568,483	9,818	85,550	37,155
1899.	299,307	1,162	10,587,789	924,945	191,030	621,799	11,744	84,813	39,274
1900.	290,204	1,325	9,974,347	1,040,805	154,414	659,125	11,551	67,059	36,305
1901.	288,508	1,304	10,277,098	1,083,372	123,506	604,464	7,000	61,539	37,600
1902.	320,098	1,410	10,725,607	628,272	118,894	554,854	8,021	57,982	36,300
1903.	382,461	1,404	10,713,356	704,736	136,615	577,554	7,375	66,922	37,416
1904.	382,435	2,136	10,515,909	734,502	118,654	568,943	5,447	52,842	41,600
1905.	375,874	1,435	10,152,679	730,119	115,222	608,258	4,637	62,150	43,200

(*a*) From *Statistique de l'Industrie Minerale.* (*b*) Not reported. (*c*) Includes pure bitumen, bituminous schist and sand, and asphaltic limestone. (*d*) Argentiferous lead ore. (*e*) Lead produced from native ores only. (*f*) Finished product. (*g*) Sulphur and limestone impregnated with sulphur. (*h*) Includes potter's clay, white clay for stucco, kaolin and clay for bricks and tiles.

MINERAL PRODUCTION OF ALGERIA. (*a*)

(In metric tons.)

Year.	Antimony Ore.	Clays.	Copper Ore.	Gypsum.		Iron Ore.	Lead-silver Ore.
				Crude.	Plaster.		
1896	658	48,297	427	300	29,870	374,476	117
1897	781	67,180	289	350	29,120	441,467	145
1898	138	78,690	488	150	29,750	473,569	120
1899	200	88,600	472	200	31,800	550,921	389
1900	93	94,000	....	500	37,100	174,000	222
1901	....	119,195	7,267	600	34,740	161,303	1,614
1902	39	122,850	1,955	600	35,500	525,012	26
1903	490	125,800	100	300	33,000	588,893	499
1904	160	125,410	1,804	350	38,420	468,737	511
1905	....	133,100	1,784	....	34,743	568 609	7,470

Year.	Lime.		Marble.	Onyx.	Phosphate Rock.	Salt.	Sand and Gravel.	Zinc Ore.
	Hydraulic.	White.						
1896	20,000	9,450	900	900	165,738	19,658	41,400	17,587
1897	20,425	9,215	1,660	364	228,141	23,222	80,860	32,269
1898	13,000	12,975	985	219	269,500	21,300	72,185	29,800
1899	12,000	13,645	225	217	324,983	17,378	72,760	42,970
1900	12,000	13,700	....	228	319,422	18,325	71,860	30,281
1901	12,000	15,000	....	294	265,000	18,518	86,727	26,913
1902	....	....	375	150	305,174	27,263	72,180	33,139
1903	....	....	700	67	320,834	26,329	46,720	43,313
1904	....	....	530	121	343,317	18,563	51,020	47,192
1905	28,990	15,220	451	270	334,784	26,986	61,900	67,922

(*a*) From *Statistique de l'Industrie Minerale.*

MINERAL PRODUCTION OF NEW CALEDONIA. (*a*)
(In metric tons.)

Year.	Chrome Iron Ore.	Cobalt Ore.	Copper Ore.	Nickel Ore.
1897	3,949	3,200	2,200	26,464
1898	7,712	2,373	*Nil.*	74,614
1899	12,634	3,294	6,349	103,908
1900	10,474	2,438	2	100,319
1901	17,451	3,123	6,349	132,814
1902	10.281	7,512	3 720	129,653
1903	21,437	8,292	10	77,360
1904	42.197	8,964	*Nil.*	98,655
1905	51,374	7,920	*Nil.*	125,289
1906	57,367	2,487	207	130,689

(*a*) From *Statistique de l'Industrie Minerale.*

MINERAL PRODUCTION OF TUNIS. (*a*)
(In metric tons.)

Year.	Salt.	Lead Ore.	Phosphate of Lime.	Zinc Ore.
1896	5,500	(*b*)	1,000	12,100
1897	8,100	2,123	(*b*)	11,830
1898	7,300	2,375	(*b*)	21,477
1899	8,850	2,263	70,000	20,079
1900	9,160	6,864	178,000	16,596
1901	16,900	8,158	172,000	17,879
1902	21,600	12,892	264,930	18,400
1903	18,846	12,752	352,088	21,262
1904	23,600	16,800	455,197	27,200
1905	52,900	15,200	522,000	37,100

(*a*) From *Annual General Reports*, by C. Le Neve Foster, and *Statistique de l'Industrie Minerale.* (*b*) Not reported.

MINERAL IMPORTS OF FRANCE. (*a*)
(In metric tons or dollars. 5 f.=$1.)

Year.	Alum.	Bitumen. (*f*)	Borax.	Bromides.	Cement.	Coal and Coke.	Copper. Ore.	Copper. Ingot and Mfres.
1895	109	43,975	442	12	13,441	10,261,069	10,450	38,196
1896	41	30,954	255	13	14,395	10,180,449	8,584	46,830
1897	54	29,931	264	18	15,141	10,457,255	11,960	54,460
1898	27	20,385	139	30	11,290	10,445,090	8,779	52,976
1899	34	30,770	123	46	13,640	11,896,030	8,517	58,419
1900	23	39,598	111	10	13,612	14,601,981	9,766	61,638
1901	39	28,888	128	3	16,232	13,925,623	13,383	47,035
1902	36	26,053	141	3	15,720	13,137,720	17,862	54,484
1903	138	27,573	312	9	21,152	14,029,687	9,796	59,126
1904	370	17,178	3,113	17	21,702	13,936,475	9,942	69,183
1905	63	24,606	1.736	31	21.954	13,910.523	14.252	70,101

Year.	Copper. Sulphate.	Copper. Oxide.	Cobalt Oxide.	Gold. Bullion and Specie.	Iron. Ore.	Iron. Pig.	Iron. Iron and Steel, Mfres. of.	Iron. Sulphate.	Iron. Oxide.
1895	24,404	24	5	$50,775,039	1,651,369	36,247	66,240	3,882	855
1896	33,803	22	5	60,167,745	1,862,043	18,323	48,423	3,086	897
1897	30,132	29	9	58,143,077	2,137,860	35,633	60,804	1,353	1,125
1898	30,897	52	9	39,881,575	2,032,240	(*b*)	47,325	896	1,021
1899	21,733	36	9	63,697,020	1,950,665	(*b*)	64,178	1,698	1,037
1900	22,820	84	9	90,408,723	2,119,003	149,755	118,152	1,589	1,022
1901	15,313	162	8	85,485,000	1,662,875	61,085	77,742	45	1,001
1902	22,273	111	10	88,091,400	1,563,334	38,521	60,697	17	1,051
1903	25,428	129	11	62,154,022	1,832,820	121,726	119,799	36	1,207
1904	30,856	142	69	133,737,561	1,738,514	135,252	125,709	319	1,151
1905	23.805	57	35	158.388.076	2.151.954	122.102	150.480	709	1,330

Year.	Kaolin.	Lead.			Lime.		Manganese Ore.
		Ore.	Carbon-ate.	Pig, scrap and Mfres.	Common and Hydraulic.	Chloride of.	
1895	....	5,032	1,077	66,241	246,677	1,047	41,400
1896	38,703	5,569	892	79,752	283,707	2,033	61,600
1897	42,384	13,981	1,327	86,589	321,047	1,713	85,500
1898	40,352	14,377	1,376	74,902	346,000	1,288	100,243
1899	36,904	12,637	2,029	67,149	321,610	1,887	106,630
1900	39,842	19,772	1,739	70,857	399,092	1,215	120,790
1901	41,972	15,430	1,789	59,051	374,281	1,400	94,365
1902	41,165	13,121	2,223	58,694	359,210	2,130	85,629
1903	47,534	20,172	2,040	75,416	386,612	919	109,930
1904	50,465	25,731	2,221	76,198	403,679	1,679	105,652
1905	52,603	35,103	2,306	73,938	409,447	406	140,871

Year.	Nickel.		Petroleum.	Phosphate Rock.	Plaster.	Platinum. *Kg.*	Potassium.	
	Ore.	Metal.					Chloride.	Chromate (*h*)
1895	10,303	252	258,700	139,600	2,412	926	3,524	2,875
1896	15,756	425	272,693	256,888	1,774	2,117	11,499	2,838
1897	17,441	316	288,671	313,608	1,869	1,069	11,630	2,852
1898	24,935	330	291,961	336,842	2,040	505	10,929	2,890
1899	28,620	286	306,078	242,021	2,260	817	13,335	3,147
1900	17,687	299	302,482	283,921	3,648	2,398	13,524	3,293
1901	39,497	252	225,962	275,285	2,844	1,857	13,299	2,784
1902	58,374	301	148,170	302,898	2,440	2,940	10,802	2,861
1903	13,933	427	(*g*)476,230	343,012	2,664	3,764	12,275	2,760
1904	20,698	313	(*g*)435,730	419,720	2,674	5,650	14,734	2,618
1905	49,698	632	(*g*)512,727	447,738	1,983	4,023	21,819	2,619

Year.	Potassium. (*Cont'd*)		Pyrites.	Quicksilver.		Sal-Am-moniac.	Salt.	Sodium.	
	Nitrate.	Carbonate		Ore.	Metal.			Hydrate.	Nitrate.
1895	775	796	67,930	23	178	9,923	17,528	1,021	$8,624,200
1896	2,614	1,526	45,788	25	234	15,256	17,191	1,109	9,025,400
1897	1,309	1,769	69,470	24	248	27,454	32,917	1,378	8,105,400
1898	1,008	2,418	71,569	19	221	20,426	35,863	1,772	8,026,400
1899	1,015	2,779	109,696	21	276	12,210	37,970	1,494	9,341,600
1900	1,928	2,768	156,825	22	161	15,205	32,045	1,062	11,995,820
1901	757	2,520	205,617	23	205	9,268	32,347	869	10,526,400
1902	1,547	1,539	170,783	24	224	15,446	32,505	643	9,372,600
1903	1,530	3,019	205,322	20	220	12,462	48,556	781	10,810,775
1904	2,117	3,781	230,097	22	208	13,744	46,232	1,068	9,074,859
1905	1,022	3,542	271,684	..	228	11,639	45,241	860	11,336,752

Year.	Sulphur.	Sulphuric Acid.	Superphos-phate of Lime.	Tin.		Zinc.	
				Ore.	Metal.	Ore.	Metal.
1895	110,989	3,461	150,758	104	7,691	41,622	25,652
1896	111,515	3,995	185,602	7	8,400	50,899	33,459
1897	136,118	3,147	195,853	149	7,642	58,074	31,211
1898	130,289	4,666	178,569	357	9,247	60,481	32,342
1899	120,062	4,583	171,631	486	6,907	78,192	25,516
1900	133,531	4,254	143,437	512	7,324	66,178	33,144
1901	101,301	5,386	165,361	365	7,314	74,553	29,812
1902	85,839	7,793	116,093	748	8,575	69,451	36,564
1903	109,594	13,241	89,229	1,808	9,873	67,258	39,305
1904	148,547	11,212	72,921	1,344	9,352	88,083	35,737
1905	129,877	10,915	31,729	1,362	9,898	105,069	29,163

MINERAL AND METALLURGICAL EXPORTS OF FRANCE. (*a*)
(In metric tons.)

Year.	Alu-minum.	Antimony. Ore.	Antimony. Metal.	Cement.	Coal.	Copper. Ore (*c*)	Copper. Metal	Gold. *Kg.* (*d*)
1895	110	832	68	(*b*)	(*b*)	1,772	8,829	1,353
1896	793	736	74	242,247	1,044,820	1,261	10,494	2,193
1897	224	623	61	244,504	1,142,195	2,000	12,667	3,335
1898	192	616	101	241,150	1,320,616	1,783	14,350	1,812
1899	256	304	255	244,480	1,229,090	2,078	17,949	2,622
1900	324	154	336	232,577	1,201,210	9,197	16,791	883
1901	307	645	741	242,010	908,583	16,066	14,776	1,869
1902	748	595	666	210,590	910,760	20,489	14,423	1,517
1903	666	904	1,358	233,835	2,238,735	12,487	11,403	3,139
1904	664	1,191	720	260,686	2,384,928	14,258	12,663	1,537
1905	928	981	815	275,503	3,348,010	13,260	13,800	5,740

Year.	Iron. Ore.	Iron. Pig.	Iron. Bars.	Iron. Steel.	Lead. Ore.	Lead. Metal.	Man-ganese Ore.	Mill-stones. *Number.*
1895	236,923	150,540	....	29,074	8,670	8,037	16,193	....
1896	238,430	195,212	24,721	44,795	8,597	10,856	10,913	196,685
1897	299,589	108,645	39,894	45,809	12,007	10,364	19,464	158,979
1898	236,169	162,991	27,424	47,278	10,216	3,663	12,229	203,584
1899	291,346	153,792	29,112	33,584	3,909	1,163	12,289	112,620
1900	371,799	114,361	18,763	19,535	2,345	958	8,392	65,436
1901	258,925	96,463	25,220	56,347	3,490	718	5,289	52,383
1902	422,677	213,081	23,828	121,932	2,414	648	1,948	45,647
1903	714,173	196,444	40,533	215,737	2,313	13,048	717	11,557
1904	1,219,149	191,819	40,374	246,738	1,860	13,467	1,392	14,479
1905	1,355,932	218,227	67,240	343,612	3,064	12,903	662	13,078

Year.	Nickel Refined.	Phos-phate. Rock.	Plaster.	Pyrites.	Silver. *Kg.* (*e*)	Tin. (metal)	Zinc. Ore.	Zinc. Spelter, Sheets and Scrap.
1895	408	....	....	37,968	13,567	650	61,291	5,849
1896	490	48,719	89,952	44,232	9,849	744	62,415	10,485
1897	498	69,188	107,823	54,367	5,374	651	79,909	10,977
1898	526	93,742	106,790	60,406	1,886	587	60,664	16,995
1899	280	70,517	112,520	53,395	....	666	76,104	14,958
1900	599	89,135	108,387	64,530	15,470	716	54,663	12,712
1901	1,031	81,405	101,063	52,952	16,745	438	42,995	15,022
1902	397	62,375	110,270	63,920	17,184	654	47,724	16,158
1903	720	72,252	131,245	119,173	43,690	1,994	62,731	12,657
1904	906	78,612	139,551	40,833	23,105	2,300	57,780	19,063
1905	1,583	55,240	124,561	21,257	66,904	2,611	72,512	17,802

(*a*) From *L'Economiste Francais* (representing the *Commerce Special*) except for last three years, which were from *Tableau Général du Commerce et de la Navigation*. (*b*) Not reported. (*c*) Includes matte. (*d*) Gold and platinum in ore sheets, leaves or threads. (*e*) Silver in ore, sheets, leaves, wire, etc. (*f*) Includes bitumen, bituminous schist and sands and asphaltic limestone. (*g*) Crude and refined. Transposition from hectoliters to tons was performed by assuming specific gravity of petroleum to be .9. (*h*) Includes chromate of soda.

GERMANY.

The mineral production of the German Empire is given in the following table in metric tons unless otherwise specified, or in dollars, on the basis of four marks to the dollar.

PRODUCTION. (*a*)

Year.	Alum.	Aluminum Sulphate.	Arsenic.		Asphaltum.	Boracite.	Cadmium. *Kg.*	Coal.	
			Ore.	Salts.				Bituminous.	Lignitic.
1897.....	2,995	37,053	3,777	2,989	61,645	198	15,531	91,054,982	29,419,503
1898.....	4,069	35,366	3,527	2,679	67,649	230	14,943	96,309,652	31,648,898
1899.....	3,358	37,693	3,834	2,423	74,770	183	13,608	101,639,753	34,204,666
1900.....	4,355	44,372	4,379	2,415	89,685	232	13,553	109,290,237	40,498,019
1901.....	4,145	46,807	4,035	2,549	90,193	184	13,144	108,539,444	44,479,970
1902.....	4,108	47,905	3,959	2,828	88,374	196	......	107,473,933	43,126,281
1903.....	3,934	49,727	4,369	2,768	87,454	159	16,565	116,637,765	45,819,488
1904.....	3,850	55,881	4,390	2,829	91,736	135	25,245	120,815,503	48,635,080
1905.....	4,127	52,892	4,913	2,535	103,006	183	24,568	121,298,607	52,512,062
1906.....	4,494	55.969	6,259	3,052	117,413	161	......	137,117,926	56,415,333

Year.	Cobalt, Nickel and Bismuth Ores.	Copper.				Gold.	Graphite.
		Ore.	Matte. (*b*)	Ingots.	Sulphate.		
1897................	3,355	700,619	315	29,408	5,549	$1,848,114	3,861
1898................	3,157	702,781	62	30,695	4,352	1,891,974	4,593
1899................	1,270	733,619	95	34,634	5,142	1,731,153	5,196
1900................	4,495	747,749	4,207	30,929	5,076	2,030,200	9,248
1901................	10,479	777,339	365	31,317	5,192	1,830,835	4,435
1902................	12,433	761,921	447	30,578	4,997	1,770,361	5,023
1903................	14,607	772,695	583	31,214	5,200	1,709,223	3,720
1904................	14,016	798,214	641	30,264	6,584	1,819,538	3,784
1905................	10,848	793,488	1,635	31,713	6,988	2,611,812	4.921
1906................	......	768,523	757	32,275	6,757	2,791,026	4,005

Year.	Iron and Steel.					Lead.		
	Iron Ore.	Pig Iron. (*c*)	Castings.	Steel.	Sulphate. (*d*)	Ore.	Pig.	Litharge
1897..............	15,465,980	6,881,466	1,473,211	6,248,141	10,351	150,178	118,881	3,441
1898..............	15,901,263	7,312,766	1,597,434	6,941.278	10,422	149,311	132,742	3 857
1899..............	17,989,635	8,153,133	1,776,878	7,532,524	10,931	144,370	129,225	3,562
1900..............	18,964,294	8,520,540	1,812,603	7,377,275	10,913	148,257	121,513	3 088
1901..............	16,570,182	7,880,087	1,520,617	7,033,438	11,148	153,341	123,098	4,101
1902..............	17,963,591	8,529,900	1,575,525	8,317,231	......	167,855	140,331	4,197
1903..............	21,230,650	10,017,901	1,721,781	9,226,898	12,243	165,991	145,319	4,428
1904..............	22,047,393	10,058,273	1,879,879	9.239,302	13,585	164,440	137,580	4,332
1905..............	23,444,073	10,875,061	2,045,477	10 309,690	12,949	152,725	152,590	3,786
1906..............	26,734,560	12,293,825	....	11,135,085	13.376	140,914	150,741	4,137

Year.	Magnesium Salts.		Manganese Ore.	Nickel. (*e*).	Petroleum.	Potassium Salts.				
	Chloride.	Sulphate.				Chloride.	Kainite. (*f*)	Sulphate.	Potassium and Magnesium Sulphate.	Unspecified.
1897........	18,014	35,072	46,427	1,464	23,303	168,001	992,389	13,774	7,812	953,798
1898........	19,819	30,295	43,354	1,691	25,989	191,347	1,103 643	18,853	13,982	1,105,212
1899........	21,370	39,540	61,329	1,747	27,027	207,506	1,108,159	26,103	9,765	1,384.972
1900........	19,397	48,591	59,204	1,989	50,375	271,512	1,227,873	30,853	15,368	1,822,758
1901........	21,018	46 714	56,691	2,207	44,095	294,666	1,498,569	37,394	15,612	2,036,325
1902........	19,658	39,262	49,812	2,196	49,725	267,512	1,322,623	28,278	18,147	1,962,384
1903........	22,990	37,844	47,994	2,637	62,680	280,248	1,557,243	36,674	23,631	2,073,720
1904........	25,730	39,412	52,886	3,063	89,620	297,238	1,905,893	43,959	29,285	2,179,471
1905........	29,017	58,568	51 463	3,317	78,869	373,177	2,387,643	47,994	34,222	2,655,845
1906......	38,468	43,013	52,485	.....	81,419	403,387	2,679,264	54,490	34,239	2,803,732

Year.	Pyrites.	Salt.		Silver and Gold Ore.	Silver. *Kg.*	Sodium Sulphate.	Sulphur.	Sulphuric Acid.
		Rock.	Evaporated.					
1897............	133,302	763,412	543,272	9,708	448,068	68,822	2,317	702,445
1898............	136,849	807,792	565,683	14,702	480,578	69,111	1,954	754,151
1899............	144,623	861,123	571,058	13,506	467,590	79,062	1,663	813,141
1900............	169,447	926,563	587,464	12,593	415,735	90,468	1,445	829,376
1901............	157,433	985,050	578,751	11,577	403,796	76,066	963	835,000
1902............	165,225	1,010,412	572,846	11,724	430,610	90,742	.....	894,409
1903............	170,867	1,095,541	598,394	11,467	396,253	83,087	219	928,190
1904............	174,782	1,079,868	621,064	10,405	389,827	75,171	209	963,384
1905............	185,368	1,165,495	612,062	10,286	399,775	68,454	205	1,228,211
1906............	196,971	1,235.030	632,922	8,066	393,442	81,175	178	1,305,065

Year.	Tin.			Uranium and Tungsten Ores.	Zinc.		
	Ore.	Block.	Chloride.		Ore.	Spelter.	Sulphate.
1897........	55	929	.....	38	663,850	150,739	5,488
1898........	51	993	.. ...	50	641,706	154,867	6,104
1899........	72	1,481	. ...	50	664,536	153,155	7,117
1900	80	2,031	(*g*) 143	43	639,215	155,790	6,027
1901........	82	1,464	(*g*) 135	43	647,496	166,283	5,552
1902........	104	2,779	.....	31	702,504	174,927	.....
1903........	110	3,065	1,064	35	682,853	182,548	5,994
1904........	99	4,216	816	23	715,732	193.058	6,185
1905........	123	5,233	811	26	731,271	198,208	5,896
1906	...	6,147	987	..	704,596	205,691	6,092

(*a*) From the *Vierteljahrshefte zur Statistik des Deutschen Reichs.* (*b*) Includes black copper. (*c*) Includes ferromanganese and spiegeleisen. (*d*) Contains a small quantity of copper and iron sulphate mixed. (*e*) Includes nickeliferous by-products, metallic bismuth, and uranium compounds. (*f*) Compound of potassium chloride and magnesium sulphate. (*g*) Includes nickel sulphate.

INDIA.

The official statistics of mineral production in British India are summarized in the subjoined table:

MINERAL PRODUCTION OF INDIA. (*a*)
(In metric tons or dollars; £1 = $5.)

Year.	Amber.	Coal.	Corundum.	Gold. (*c*)	Graphite.	Iron Ore.	Jade. (*e*)	Magnesite.
1896.....	(*b*)	3,909,764	(*b*)	$7,085,432	(*b*)	50,559	215	(*b*)
1897.....	(*b*)	4,128,330	(*b*)	8,041,055	(*b*)	61,697	219	(*b*)
1898.....	$5,080	4,681,927	133	7,798,709	(*b*)	(*d*)42,524	196	(*b*)
1899.....	755	5,174,752	40	8,357,087	1,548	(*d*)52,832	228	(*b*)
1900.....	515	6,222,591	63	9,205,518	1,859	(*d*)57,912	142	(*b*)
1901.....	55	6,741,899	74	9,394,723	2,530	(*d*)58,725	206	(*b*)
1902.....	2,160	7,543,272	26	9,611,985	4,648	(*d*)77,273	137	3,597
1903.....	2,070	7,557,400	(*b*)	11,203,926	3,448	(*d*)62,337	191	838
1904.....	4,190	8,348,561	(*b*)	11,513,340	2,955	72,754	171	1,193
1905.....	4,725	8,552,422	(*b*)	11,760,957	2,361	103,814	122	2,096

Year.	Manganese Ore.	Mica. (*e*)	Petroleum. *Gallons.*	Rubies	Salt.	Saltpeter (Potassium nitrate.)	Tin Ore.
1896..........	57,782	452	15,057,094	$171,884	1,043,171	21,425	82
1897..........	74,862	652	19,128,828	200,613	937,932	26,845	62
1898..........	61,419	527	22,234,438	289,750	1,043,862	21,224	40
1899..........	88,524	497	32,934,007	454,240	977,269	18,555	64
1900..........	129,865	1,025	37,729,211	486,630	1,071,877	20,189	94
1901..........	122,831	1,505	50,075,117	522,380	1,208,933	17,711	63
1902..........	160,311	739	56,607,688	434,475	1,116,797	18,005	91
1903..........	174,563	926	87,859,069	444,095	1,021,581	20,861	100
1904..........	152,602	828	118,491,382	453,060	1,319,535	7,469	63
1905..........	257,958	6,039	144,798,444	(*f*) 441,700	1,425,010	18,473	67

(*a*) Records of the Geological Survey of India. (*b*) Not reported. (*c*) £1 = $4.866. (*d*) Production of iron ore in Bengal only. (*e*) Production and exports the same as for fiscal year ending March 31. (*f*) For year ending Feb. 28, 1906.

ITALY

The following tables itemize the statistics of the production and the foreign commerce of mineral and metallurgical products in Italy:

MINERAL PRODUCTION AND REFINED PRODUCTS OF ITALY. (*a*)
(In metric tons or dollars; 5 lire=$1.)

Year.	Alum.	Aluminum Sulphate.	Alunite.	Antimony.	Antimony Ore.	Asphalt, Mastic and Bitumen.	Asphaltic Rock.	Barytes.
1895.......	995	2,950	7,000	423	2,241	14,491	46,713	(*b*)
1896.......	850	2,390	6,000	538	5,086	12,490	45,456	(*b*)
1897.......	1,030	2,310	6,500	404	2,150	18,644	55,339	(*b*)
1898.......	1,165	2,915	7,000	380	1,931	17,813	93,750	12,400
1899.......	945	2,330	5,800	581	3,791	41,732	81,987	12,545
1900.......	1,097	2,403	5,200	1,174	7,609	33,127	101,738	14,003
1901.......	1,075	2,260	4,900	1,721	8,818	31,814	104,111	13,245
1902.......	....	....	8,200	....	6,116	....	64,245	....
1903.......	....	....	8,100	905	6,927	35,757	89,078	....
1904.......	2,490	2,210	8,000	836	5,712	34,227	111,390	250
1905.....	2,975	2,740	8,500	327	5,083	29,716	106,586	375

Year.	Borax. Refined.	Boric Acid.		Coal. (c)	Coal. (Briquettes).	Coke	Copper.		Sulphate.
		Crude.	Refined.				Ore.	Ingot, etc.	
1895.....	944	2,633	253	305,321	451,470	394,043	83,670	2,375	
1896.....	943	2,616	253	276,197	422,409	426,906	90,408	2,842	4,756
1897.....	990	2,704	260	314,222	549,050	430,617	93,377	2,980	5,337
1898.....	702	2,650	166	341,327	594,500	469,228	95,128	3,230	6,364
1899.....	709	2,674	129	388,534	566,000	485,951	94,764	3,032	7,795
1900.....	858	2,491	283	479,896	703,740	487,831	95,644	2,797	13,191
1901.....	544	2,558	347	425,614	754,800	490,803	107,750	3,097	15,374
1902.....	....	2,763	....	414,569	....	....	101,142	....	14,601
1903.....	....	2,583	....	346,887	724,993	554,559	114,823	....	18,164
1904.....	569	2,624	314	362,151	903,610	607,297	157,503	2,313	17,237
1905...	1,007	2,700	749	412,916	842,250	627,984	149,045	1,175	26,212

Year.	Gold.		Graphite.	Iron and Steel.				
	Ore.	Bullion.		Ore.	Pig.	Bar, Sheet, Pipe, Wire, etc.	Tin Plate.	Steel.
1895.....	7,099	$186,074	2,657	183,371	9,213	163,824	5,860	50,314
1896.....	7,659	172,552	3,148	203,966	6,987	139,991	2,918	65,955
1897.....	10,723	209,998	5,650	200,709	8,393	149,944	6,500	63,940
1898.....	9,549	124,869	6,435	190,110	12,387	167,499	7,200	87,467
1899.....	11,859	75,294	9,990	236,549	19,218	197,730	8,000	108,501
1900.....	5,840	38,212	9,720	247,278	23,990	190,518	10,000	115,887
1901.....	890	2,725	10,313	232,299	15,819	180,729	7,550	123,310
1902.....	1,215	....	9,210	240,705	....	....	....	....
1903.....	5,734	41,933	7,920	374,790	90,744	177,392	11,275	154,134
1904.....	6,746	43,063	9,765	409,460	112,598	181,385	16,655	177,086
1905...	1,200	9,968	10,572	366,616	181,248	205,915	18,560	244,793

Year.	Lead. Ore.	Lead. Pig.	Manganese Ore.	Manganiferous Iron Ore.	Marble.	Petroleum, Crude.	Petroleum, Benzine, etc.	Pumice-Stone.
1895	30,632	20,353	1,569	5,860	186,900	3,594	4,191	(b)
1896	33,545	20,786	1,890	10,000	209,428	2,524	2,734	(b)
1897	36,200	22,407	1,634	21,262	236,958	1,932	3,392	(b)
1898	33,930	24,543	3,002	11,150	271,725	2,015	5,040	2,766
1899	31,046	20,543	4,356	29,874	313,744	2,242	5,384	7,300
1900	35,103	23,763	6,014	26,800	310,336	1,683	6,077	7,000
1901	43,449	25,796	2,181	24,290	334,146	2,246	4,211	8,300
1902	42,330	....	2,477	23,113	....	2,633	....	....
1903	42,443	22,126	1,930	4,735	....	2,486	4,577	....
1904	42,846	23,475	2,836	*Nil.*	390,118	3,543	6,388	11,600
1905	39,030	19,077	5,384	*Nil.*	389,869	6,122	9,924	11,300

Year.	Pyrites. (Cupriferous in part).	Quicksilver. Ore.	Quicksilver. Metal.	Salt. Brine.	Salt. Rock.	Salt. Sea.	Silver. Ore.	Silver. Bullion. *Kg.*
1895	38,586	10,504	199	10,605	18,710	448,335	870	44,189
1896	45,728	14,305	186	11,974	17,300	422,555	640	38,075
1897	58,320	20,659	192	11,725	19,801	429,253	405	45,313
1898	67,191	19,201	173	11,546	18,199	451,426	435	43,437
1899	76,538	29,322	205	11,021	18,721	363,826	540	33,645
1900	71,616	33,930	260	10,890	18,331	338,034	584	31,169
1901	89,376	38,614	278	10,690	23,054	401,443	511	32,464
1902	93,177	44,261	....	10,581	23,677	424,239	421	....
1903	101,455	55,528	312	10,962	25,911	451,633	405	24,388
1904	112,004	60,403	352	11,878	18,638	433,810	143	24,943
1905	117,667	63,378	369	12,756	19,669	405,274	170	20,215

Year.	Sulphur. Crude (Fused).	Sulphur. Ground.	Sulphur. Refined.	Talc. Ground.	Zinc. Ore.	Zinc. Spelter.
1895	370,766	91,517	75,329	(b)	121,197	*Nil.*
1896	426,353	89,292	71,072	(b)	118,171	*Nil.*
1897	496,658	69,178	85,872	(b)	122,214	250
1898	502,351	146,001	99,494	12,760	132,099	250
1899	563,697	161,509	110,213	11,000	150,629	251
1900	544,119	167,466	157,957	14,415	139,679	547
1901	563,096	171,252	141,431	11,770	135,784	511
1902	....	....	....	....	131,965	....
1903	553,751	139,376	139,464	6,300	157,521	126
1904	527,563	189,266	163,695	6,740	148,365	189
1905	568,927	180,676	180,774	6,490	147,834	5

(a) From *Rivista del Servizio Minerario.* (b) Not reported. (c) Includes anthracite, lignite, fossil wood and bituminous schist.

MINERAL IMPORTS OF ITALY. (a)

(In metric tons or dollars; 5 lire = $1.)

Year.	Antimony	Arsenic. *Kg.*	Asbestos.	Asphaltum.	Barytes.	Borax and Boric acid.	Cement and Hydraulic Lime.	Chalk.
1896	38	(b)	851	11,892	549	166	12,810	15,716
1897	66	2,604	619	1,632	578	253	16,680	28,937
1898	58	700	1,186	1,150	860	147	12,029	18,252
1899	64	600	1,675	1,473	936	123	14,391	13,738
1900	37	900	1,645	1,933	859	122	15,494	18,436
1901	49	1,800	2,019	1,450	825	232	14,872	20,731
1902	80	1,200	1,536	1,020	1,170	516	13,732	15,216
1903	98	4,400	1,691	1,567	1,099	504	15,547	10,063
1904	131	3,700	2,174	2,604	1,875	271	15,260	6,891
1905	117	3,400	1,806	3,252	1,444	112	15,797	5,556
1906	50	5,300	2,171	2,854	1,400	163	18,937	7,714

Year.	Clay Products.			Coal.	Copper Ore.	Copper Cement.
	Brick, Tile, etc.	Kaolin	Terra cotta.			
1896	18,504	3,775	2,675	4,081,218	484	1,150
1897	19,086	5,719	2,167	4,259,643	1,611	1,049
1898	21,681	9,079	2,122	4,431,524	5,471	2,040
1899	22,410	12,105	2,200	4,859,556	2,777	1,328
1900	25,702	9,595	2,031	4,947,180	5,290	1,298
1901	35,534	12,809	2,482	4,838,994	11,047	1,987
1902	29,156	14,165	2,537	5,406,069	9,422	2,299
1903	24,586	11,033	2,883	5,546,823	9,459	649
1904	33,655	18,610	3,236	5,904,578	8,104	309
1905	37,762	15,315	4,080	6,437,539	6,879	486
1906	47,667	19,218	4,657	7,673,435	9,363	802

Year.	Copper, Brass and Bronze.	Copper and Iron Sulphates.	Glass and Manufactures.	Gold.			Graphite.	Iron.		
				Specie. *Kg.*	Unrefined. *Kg.*	Manufactures: *Kg.*		Ore.	Pig.	Wrought.
1896	6,955	24,255	9,322	1,004	2,517	1,515	204	594	119,491	4,820
1897	7,999	28,878	11,182	444	807	1,375	315	5,831	156,019	3,801
1898	7,433	25,560	10,399	154	507	1,844	382	8,723	169,059	4,076
1899	7,334	27,408	10,303	181	326	1,390	608	20,799	191,613	4,158
1900	9,249	32,127	11,356	188	309	1,348	982	19,205	160,686	7,405
1901	8,659	32,058	10,448	1,115	494	1,547	102	4,054	159,972	5,695
1902	10,865	25,107	11,813	8,967	479	1,269	60	4,314	155,143	6,603
1903	9,588	24,566	12,830	44,218	1,396	1,220	63	5,937	126,756	6,380
1904	15,198	37,298	13,957	8,364	1,961	1,640	52	4,390	149,130	6,740
1905	18,188	30,684	14,227	46,509	5,768	1,799	107	4,745	136,077	7,616
1906	21,472	25,060	20,987	30,940	4,571	2,633	361	6,452	168,985	13,441

Year.	Iron and Steel.		Lead.			Lead. Oxide and Carbonate	Mineral Paints.	Nickel Alloys and Manufactures.	Petroleum.
	Plates, Rods and Manufactures.	Scrap.	Ore. (*c*)	Metal and Alloys in Pigs.	Mnfres.				
1896	52,202	162,035	9,730	1,166	192	523	852	411	70,217
1897	88,895	130,938	14,854	1,178	247	580	888	432	68,973
1898	98,564	138,426	10,947	1,431	435	647	692	258	70,654
1899	142,645	245,616	7,476	3,990	249	662	958	250	71,391
1900	149,263	197,415	9,134	3,248	233	557	958	232	73,089
1901	146,970	148,305	9,063	2,926	268	815	865	476	69,298
1902	161,222	198,914	1,680	7,563	288	846	670	561	68,781
1903	167,085	206,036	689	5,398	273	768	859	525	68,220
1904	167,128	246,359	2,187	4,541	247	871	940	652	69,233
1905	180,757	276,311	465	6,764	295	686	974	574	66,493
1906	285,256	345,342	4,526	10,959	319	984	1,954	717	64,541

Year.	Phosphate Rock.	Potash, Ammonia and Caustic Soda.	Potassium Sulphate.	Precious Stones. Manufactures.	Quick-Silver.	Silver. *Kg.*			Slag.
						Specie.	Unrefined in Bars.	Manufactures.	
1896	(*b*)	9,841	431	$1,505,461	30	22,751	2,291	6,533	30,275
1897	(*b*)	11,012	562	2,033,714	30	26,008	2,434	5,286	37,201
1898	65,126	11,047	928	1,629,008	39	8,241	991	5,673	51,199
1899	116,283	12,370	1,297	1,323,317	62	20,605	1,782	4,881	56,549
1900	140,281	14,077	1,670	1,801,264	49	29,291	2,678	4,358	32,254
1901	142,108	14,693	1,411	2,967,041	36	35,089	4,391	4,213	7,312
1902	159,341	17,617	1,566	3,885,484	57	28,662	8,768	3,455	5,634
1903	172,328	17,528	1,353	4,866,327	28	81,373	12,541	5,893	8,849
1904	217,162	14,846	1,663	4,896,867	25	67,520	15,885	4,409	3,821
1905	240,144	17,752	1,804	6,333,376	57	51,997	20,697	5,437	5,326
1906	307,762	16,718	1,534	7,157,829	12	122,737	20,410	5,994	6,227

Year.	Sodium Salts.			Tin.		Zinc.			
	Carbonate.	Nitrate (Crude).	Sod. and Pot. Nitrates, Refined.	Block.	Mnfres.	Ore.	Oxide.	Spelter and Old.	Mnfres.
1896	18,927	11,685	541	1,763	91	(b)	540	2,596	3,482
1897	20,721	16,400	917	1,520	81	(b)	570	3,278	3,556
1898	20,845	19,961	702	1,722	109	216	573	2,813	3,200
1899	22,654	22,385	671	1,240	96	(b)	804	3,498	3,221
1900	23,215	27,706	511	1,643	56	85	1,034	3,627	3,543
1901	21,956	40,498	315	1,858	91	23	813	3,991	4,079
1902	26,133	24,483	314	2,114	110	131	904	3,805	4,167
1903	24,753	43,480	638	2,288	130	46	1,416	4,551	4,461
1904	27,747	32,283	613	2,170	150	362	1,124	5,202	4,168
1905	29,066	46,517	689	2,304	103	14	1,246	5,997	4,701
1906	31,170	32,508	395	3,361	167	2,042	1,920	6,835	4,421

MINERAL EXPORTS OF ITALY. (*a*)
(In metric tons or dollars; 5 Lire=$1.)

Year.	Antimony.	Asbestos.	Asphaltum.	Barytes.	Borax and Boric Acid.	Cement and Hydraulic Lime.	Chalk.
1896	361	130	13,729	66	2,719	3,871	5,593
1897	271	170	15,310	143	1,618	5,330	7,556
1898	338	208	19,465	70	2,167	5,192	6,744
1899	240	245	26,402	45	2,872	5,462	5,386
1900	467	261	24,287	40	2,114	6,860	2,980
1901	765	302	21,856	32	2,190	8,463	3,428
1902	359	144	20,884	91	1,847	7,930	4,215
1903	314	222	24,303	35	901	6,325	3,802
1904	107	163	14,880	70	1,122	7,810	4,089
1905	132	236	23,740	162	2,255	8,445	5,007
1906	208	205	27,180	147	2,777	6,774	4,194

Year.	Clay Products.				Coal.	Copper Ore.	Brass and Bronze.	Copper, and Iron Sulphate.
	Brick, Tile, etc.	Kaolin.	Porcelain.	Terra-Cotta.				
1896	143,648	49	99	2,852	18,924	3,603	643	71
1897	125,925	93	138	2,472	23,191	2,408	641	18
1898	125,614	94	86	2,751	17,749	2,356	857	25
1899	136,402	94	124	2,796	20,803	1,148	1,734	20
1900	122,388	179	96	3,051	23,926	1,179	1,209	60
1901	108,057	368	266	3,325	25,594	9	659	20
1902	122,482	*Nil*	154	3,378	33,374	11	874	39
1903	148,407	133	192	4,482	29,219	15	952	44
1904	157,999	18	242	4,581	35,149	43	939	29
1905	151,243	86	229	5,194	38,555	77	812	249
1906	145,064	1,173	207	5,731	31,666	189	1,244	102

Year.	Glass and Manufactures.	Gold.		Graphite.	Iron.			Iron and Steel Plates, Rods and Manufactures.
		Specie.	Unrefined. *Kg.*		Ore.	Pig.	Wrought.	
1896	5,867	$1,539,460	2,517	3,727	187,059	1,378	427	1,819
1897	6,022	996,960	1,381	4,164	207,619	498	1,434	7,148
1898	5,209	1,590,920	1,739	5,145	217,556	840	699	4,606
1899	5,930	1,281,540	1,162	8,114	234,515	378	611	4,817
1900	6,158	1,453,900	2,763	7,820	170,286	329	440	5,610
1901	8,182	1,159,400	2,955	7,169	121,592	311	499	5,120
1902	5,131	1,176,140	733	7,098	209,070	395	1,054	5,540
1903	5,344	507,160	1,291	7,068	98,319	810	1,670	6,594
1904	5,490	1,011,840	1,494	7,433	2,577	229	847	9,945
1905	5,019	655,340	1,731	6,811	11,358	1,395	951	12,215
1906	4,856	722,746	1,476	4,904	1,833	254	472	12,017

Year.	Lead. Ore.	and Lead Alloys in Pigs.	Manufactures.	Oxide and Carbonate.	Mineral Paints.	Phosphate Rock.	Potash, Ammonia and Caustic Soda.	Quicksilver.	Salt.
1896	4,731	1,419	1,441	489	2,412	(b)	88	155	171,740
1897	4,747	2,790	1,410	461	2,318	(b)	66	236	176,520
1898	4,492	5,870	1,764	414	2,884	(b)	85	244	126,860
1899	3,129	2,497	910	389	2,784	(b)	120	223	114,050
1900	3,741	5,018	1,408	367	2,977	1,726	142	259	112,900
1901	3,977	4,463	2,128	410	2,913	1,290	198	301	114,210
1902	3,354	5,650	2,258	404	2,953	894	136	215	145,190
1903	5,041	2,911	1,934	426	3,305	2,942	233	222	144,910
1904	5,524	1,954	1,887	347	3,231	2,812	162	266	130,940
1905	4,311	976	2,043	310	3,632	3,519	238	243	116,040
1906	8,356	2,005	2,201	315	4,502	1,652	304	278	126,199

Year.	Silver. *Kg.* Specie.	Unrefined.	Slag.	Sodium Salts. Carbonate.	Nitrate. (Crude.)	Sod. and Pot. Nitrates, Refined.	Stone. Alabaster. (Crude.)	Building Stone.	Marble. (Crude.)	Marble and Alabaster.
1896....	28,377	26,854	4,753	279	51	306	289	23,580	80,750	68,639
1897....	72,605	50,503	8,847	275	151	344	269	36,229	83,081	62,750
1898....	8,241	68,607	6,861	391	79	256	457	35,945	88,404	68,150
1899....	32,085	32,432	4,898	438	136	124	714	53,904	98,485	84,104
1900....	10,501	25,310	4,222	486	58	129	489	54,051	61,650	72,619
1901....	14,446	42,325	3,261	377	116	59	474	53,668	96,631	73,599
1902....	10,978	20,427	3,615	446	346	259	727	74,036	112,967	83,172
1903....	4,377	9,486	4,929	482	781	492	605	69,473	130,316	87,079
1904....	3,834	24,165	4,458	376	363	230	800	80,337	131,087	82,911
1905....	2,371	25,947	9,725	214	424	159	935	116,110	132,765	94,295
1906....	3,465	18,262	8,961	253	80	133	942	83,001	148,579	99,776

Year.	Sulphur.	Tin. Block.	Manufactures.	Zinc. Ore.	Oxide.	Spelter and Scrap.	Manufactures.
1896........................	356,370	10	89	115,454	48	33	8
1897........................	358,932	29	109	133,125	189	309	63
1898........................	405,823	34	177	130,064	110	156	14
1899........................	424,018	69	176	140,107	123	227	21
1900........................	479,139	147	153	111,870	102	359	24
1901........................	414,018	202	187	103,020	140	349	18
1902........................	439,242	236	174	114,894	122	338	66
1903........................	461,289	173	180	116,449	116	591	51
1904........................	437,067	171	151	126,393	483	263	46
1905........................	381,128	285	107	117,810	173	434	48
1906........................	336,339	306	81	144,244	687	639	50

(a) From *Statistica del Commercio speciale di Importazione e di Esportazione*. (b) Not reported. (c) Includes argentiferous lead ore.

MEXICO.

Owing to the incompleteness of the Mexican statistics of production, we are unable to give any satisfactory table. Exports may, however, be taken as indicating the condition of the mining industry.

MINERAL EXPORTS OF MEXICO. (a)
(In metric tons or Mexican dollars.)

Year.	Antimony Ore.	Coal.	Copper.		Gold.				
			Ore.	Ingot.	Ore.	Bullion.	Specie.	Cyanide.	Sulphide.
1895.........	600	61,686	3,006	20,429	$103,773	$4,920,504	$175,098	$31,231	$3,026
1896.........	3,231	75,541	144	20,659	206,874	5,533,789	261,078	161,784	44,890
1897.........	5,873	105,298	1,094	16,858	365,226	6,220,765	202,223	226,986	33,916
1898.........	5,932	118,553	13,146	10,362	1,037,202	6,493,735	(b)	294,730	64,061
1899.........	10,382	113,192	223	25,293	335,849	7,017,286	183,474	115,961	266,782
1900.........	2,313	38,676	408	27,970	306,392	7,435,864	192,456	128,675	177,193
1901.........	5,103	17,281	5,576	33,818	284,722	8,324,681	210,431	178,803	81,744
1902.........	1,280	3,406	6,101	63,609	303,979	9,079,371	129,899	78,295	40,658
1903.........	7,302	1,840	10,912	51,716	264,503	9,693,692	54,636	85,465	124,020
1904 (c).....	1,775	(b)	48,365	57,338	537,290	10,867,272	(b)	79,129	176,090

Year.	Graphite.	Gypsum.	Lead.		Silver.					
			Ore.	Base Bullion.	Ore.	Bullion.	Specie.	Sulphide.	Cyanide.	Slag.
1895.	794	1,340	568	50,122	$10,977,079	$22,178,294	$18,300,553	$555,475	$14,649	$72,590
1896.	795	2,050	167	48,663	9,971,053	28,565,843	18,737,331	1,495,306	38,049	64,121
1897.	759	2,095	2	60,029	11,401,176	35,775,125	21,925,347	1,663,581	123,246	39,800
1898.	(b)	1,650	(b)	60,918	11,048,358	37,137,599	(b)	1,663,501	257,342	46,488
1899.	2,305	1,050	1	67,441	10,766,099	37,585,911	5,580,834	1,929,085	76,942	4,810
1900.	2,561	1,600	468	74,944	12,495,524	41,468,745	22,679,655	1,893,646	67,607	87,883
1901.	762	800	(b)	79,097	9,615,939	36,348,374	12,038,158	2,141,685	259,282	93,549
1902.	1,434	(b)	118	107,366	4,108,088	45,796,576	17,753,526	1,978,919	108,344	132,093
1903.	1,404	(b)	11	100,532	11,781,048	48,276,797	16,167,673	1,642,627	135,561	289,900
1904.	126	(b)	1	95,010	11,000,869	45,430.020	....	1,392,356	171,452	202,594

(a) From the *Estadistica Fiscal*. The figures for the calendar years were arrived at by combining those of the successive semesters of the different fiscal years. (b) Not reported. (c) Figures for 1904 were from *Anuario Estadistico de la Republica Mexicana* for 1904.

NORWAY.

The official statistics of mineral production, imports and exports, are summarized in the following tables:

MINERAL PRODUCTION OF NORWAY. (*a*)
(In metric tons or dollars; 1 Krone=27 cents.)

Year.	Apatite (*b*)	Chrome Ore.	Copper.		Feldspar.	Gold.	Iron.		
			Ore.	Ingot.			Ore.	Pig and Cast.	Bars and Steel.
1896......	1,106	....	29,910	1,067	12,223	$9,450	2,000	335	400
1897......	872	....	27,606	1,064	17,392	675	3,627	417	452
1898......	3,593	....	37,047	941	11,355	1,539	4,425	231	379
1899	1,500	41	43,358	1,209	19,260	2,700	4,576	406	666
1900......	300	165	46,858	1,280	17,609	2,430	17,925	444	614
1901......	738	85	40,726	1,073	18,323	2,700	42,252	261	376
1902......	2,295	22	40,499	1,347	19,591	36,990	53,675	527	461
1903......	1,795	*Nil*	35,417	1,382	18,590	83,700	53,475	509	442
1904......	1,456	154	36,891	1,342	20,835	*Nil*	45,328	350	395
1905......	2,522	*Nil*	37,045	1,153	22,508	*Nil*	46,582	474	253

Year.	Nickel.		Pyrites, Iron and Copper.	Rutile.	Silver.		Zinc Ore (*d*)
	Ore.	Metal.			Ore and NativeSilver.	Metal. *Kg.*	
1896..................	*Nil*	16	60,507	30	527	4,664	450
1897..................	*Nil*	*Nil*	94,484	32	642	5,372	908
1898..................	*Nil*	*Nil*	89,763	35	497	4,802	320
1899..................	220	5	95,636	30	429	4,600	379
1900..................	1,888	13	98,945	40	475	4,600	204
1901..................	2,018	40	101,894	55	519	5,680	90
1902..................	4,040	60	121,247	*Nil*	471	6,220	30
1903..................	5,670	75	129,939	25	481	7,269	335
1904..................	5,352	73	133,603	25	1,297	8,064	42
1905..................	5,477	77	162,102	35	1,570	7,100	4,241

(*a*) *Tabeller vedkommendo Norges Bergværkdsdrift, Statistik Aarbog for Knogeriget Norge.* (*b*) Exports which represent production. (*d*) Includes lead ore.

MINERAL IMPORTS OF NORWAY.(*a*)
(In metric tons.)

Year.	Borax. *Kg.*	Cement and Hydraulic Lime.	Coal.	Copper and Brass.		Glass and Glassware.
				Plates and Bars.	Wares.	
1896............................	38,305	16,028	1,136,087	1,074	479	3,729
1897............................	44,495	18,734	1,229,966	1,140	591	4,262
1898............................	71,590	25,403	1,232,792	1,064	807	3,905
1899............................	62,060	33,652	1,478,080	1,000	1,120	3,229
1900............................	71,124	24,511	1,520,162	696	1,164	2,874
1901............................	68,000	20,993	1,413,228	1,018	761	1,793
1902............................	(c)	18,984	1,547,089	1,118	(c)	(c)
1903............................	(c)	17,906	1,528,612	899	309	690
1904............................	54,953	12,845	1,394,343	688	866	1,158
1905............................	(c)	13,797	....	882	1,146	1,106

Year.	Iron and Steel.							Other Mnfres.
	Pig and Cast.	Bars, Hoops, etc. Wrought Iron.	Anchors, Cables and Chains.	Rails.	Nails and Spikes.	Steel.	Bars, Rods, etc.	
1896.......	29,764	26,552	1,090	4,315	1,760	2,754	17,930	6,831
1897.......	35,055	29,038	1,367	7,637	2,097	4,350	23,350	10,695
1898.......	41,227	26,203	1,485	10,327	2,087	2,428	26,894	17,182
1899.......	46,680	25,379	1,394	8,137	1,529	2,652	32,192	21,400
1900.......	32,113	23,010	1,203	11,952	1,219	2,085	29,318	17,493
1901.......	31,660	20,672	1,708	22,959	1,808	1,905	31,184	18,372
1902.......	30,590	26,685	2,103	15,316	2,205	1,754	36,288	22,069
1903.......	31,869	21,977	1,807	4,631	1,261	1,958	42,098	18,855
1904.......	30.975	24,094	2,109	5,814	1,071	1,610	42,013	5,462
1905.......	......	27,740	2,224	6,566	714	1,436	......	......

Year.	Lead in Pigs and Sheets.	Lead, White and *Zinc* Oxide.	Petroleum and Paraffin.	Potash.	Salt.	Saltpeter.	Soda.	Sulphur. (*b*)	Tin in Blocks, etc.	Zinc in Bars, Plates, etc.
1896...	653	1,192	35,823	945	117,920	308	5,156	9,347	142	1,101
1897...	848	1,119	39,810	919	164,572	277	5,492	10,701	236	1,102
1898...	732	1,491	36,504	754	127,341	477	4,823	9,589	257	1,370
1899...	869	1,296	42,182	802	134,583	278	4,555	10,734	546	1,509
1900...	670	1,216	39,657	638	143,365	356	4,576	14,827	149	1,254
1901...	590	1,321	47,011	518	127,607	208	5,220	11,149	141	1,027
1902...	(*c*)	(*c*)	(*c*)	(*c*)	141,415	315	(*c*)	(*c*)	(*c*)	1,104
1903...	311	(*c*)	58,822	457	143,110	245	4,200	8,829	106	1,015
1904...	498	1,898	50,543	477	153,699	321	3,197	12,181	176	940
1905...	448	1,309	43,860	393	137,800	1,048	3,704	10,240	134	967

MINERAL EXPORTS OF NORWAY. (*a*)

(In metric tons.)

Year.	Apatite.	Clay Products.		Copper.			Feldspar.
		Bricks. *Thousands.*	Earthenware.	Ore.	Ingot.	Scrap.	
1896	1,160	10,008	365	30,367	1,276	712	12,223
1897.....................	872	11,711	260	15,111	1,222	670	17,392
1898	3,593	15,534	2	13,587	1,650	1,206	11,355
1899	1,500	11,949	2	7,198	1,785	1,038	19,260
1900	300	5,266	2	5,756	1,891	1,168	17,609
1901.....................	738	12,103	7	6,041	1,465	774	(*d*)18,423
1902	2,295	(*c*)	....	4,848	1,913	(*c*)	(*d*)19,611
1903	1,795	37,972	5	3,448	1,930	888	(*d*)18,640
1904.....................	1,456	29,706	11	2,673	1,124	785	20,835
1905.....................	2,522	29,861	....	3,393	958	968	20,696

Year.	Glass and Glassware.	Iodine-*Kg.*	Iron.				
			Ore.	Pig and Scrap.	Bars and Hoops.	Nails and Spikes.	Steel.
1896	1,231	1,959	2,051	5,493	12	10.664	132
1897..................	1,432	2,395	4,242	4,631	56	9,097	167
1898	841	5,474	4,601	3,844	25	7,270	158
1899	840	16,180	12,517	6,085	337	6,089	377
1900..................	1,531	11,210	27,158	8,141	135	5,643	220
1901..................	2,142	10,000	39,173	3,250	370	6,001	179
1902..................	(*c*)	....	48,775	7,359	166	6,431	240
1903..................	200	11,417	41,575	6,350	10	6,504	200
1904..................	219	9,414	45,434	10,152	13	7,477	167
1905..................	381	9,000	60,558	9,920	34	8,725	88

Year.	Nickel Ore.	Pyrites.	Silver Ore.	Stone.			
				Ashlar.	Marble and Dolomite.	Quartz.	Whetstones.
1896.....................	*Nil.*	41,562	174	66,233	5,421	3,178	205
1897.....................	*Nil.*	70,552	119	74,492	3,111	5,608	112
1898.....................	30	67,502	79	98,692	4,267	2,244	137
1899.....................	63	83,912	14	105,591	2,814	3,291	170
1900.....................	272	84,604	90	101,959	3,134	3,523	136
1901.....................	55	104,151	6	121,362	2,052	3,512	181
1902.....................	1	105,980	*Nil.*	144,691	2,341	3,428	138
1903.....................	*Nil.*	118,148	*Nil.*	165,874	4,491	4,485	170
1904.....................	30	116,550	*Nil.*	189,237	3,132	6,679	169
1905.....................	220	147,155	*Nil.*	173,558	....	....	137

(*a*) From *Tabeller vedkommende Norges Bergvaerksdrift und Tabeller vedkommende Norges Handel.* (*b*) Includes flowers of sulphur. (*c*) Returns not available. (*d*) Includes a small quantity of fluorspar.

PERU.

In this Republic there were, at the end of 1905, metallurgical works to the number of 89, of which 19 were amalgamation, 32 leaching, 23 smelting, and 12 of other kinds, besides three refineries of petroleum and sulphur. In 1906 the erection of a large smelting plant at Rio Blanco was begun by an American company. The statistics of mineral and metal production in Peru in 1903–1905, as reported by the Cuerpo de Ingenieros de Minas del Peru, in its *Boletin*, No. 41, Lima, 1906, are given in the following table:

MINERAL AND METAL PRODUCTION OF PERU.
(In metric tons.)

Year.	Bismuth	Borate	Coal (*a*)	Copper	Gold *Kg.*	Nickel *Kg.*	Petro leum	Quick-silver *Kg.*	Silver. *Kg.*	Salt
1903......	....	2,466	36,920	9,497	1,078.3	....	37,079	....	170,800	17,637
1904......	....	2,675	59,920	9,504	601.4	....	38,683	....	145,165	18,545
1905......	12	1,954	75,338	12,213	776.6	1,778	49,700	1,554	191,476	21,039

(*a*) Includes asphaltum and bituminous schist.

Details of the various branches of the mineral industry in Peru are given under the respective captions in the previous part of this volume. Mining in the Republic is greatly retarded by the instability and indolence of the labor. The total number of men engaged in the industry in 1905 was 12,861, of whom 8807 were employed in the mines, 3210 in salt works, and 844 in the production of petroleum.

RUMANIA AND SERVIA.

The mineral statistics of these countries have been reported only for a few years back. Such as are available are given in the following tables:

MINERAL PRODUCTION OF RUMANIA. (a)
(In metric tons.)

Year.	Coal.			Petroleum.	Salt.	Stone. (Building)
	Anthracite.	Bituminous.	Lignitic.			
1902–1903	327	4,547	117,607	324,735	105,056	138,627
1903–1904	535	4,520	119,562	412,390	109,175	142,730
1904–1905	997	7,625	121,604	530,526	117,449	103,726

(a) From *Current Report and Statistics.*

MINERAL PRODUCTION OF SERVIA. (a)
(In metric tons.)

Year.	Antimony.	Coal.		Copper.		Gold. (b)	Lead.	Silver. *Kg.*
		Bituminous.	Lignitic.	Ore.	Metal.			
1902	337	35,888	117,866	2,520	140	$12,626	6	21
1903	344	40,962	118,866	....	192	7,310	82	....
1904	472	43,529	139,675	50	164	56,487	25	48

(a) From British blue book, "Mines and Quarries," 1904, Part IV, Colonial and Foreign Statistics (b) 1 Kg. gold=$664.55.

SPAIN.

The following tables record the mineral and metal production of Spain, as reported by official authorities:

MINERAL PRODUCTION OF SPAIN. (*a*)
(In metric tons.)

Year.	Aluminous Earths.	Antimony ore.	Arsenic Sulphide.	Asphaltum.	Asphalt Rock.	Barytes.	Cement, Hydraulic.	Coal. Anthracite.
1896	320	54	271	1,285	1,117	345	130,738	14,895
1897	409	354	244	1,878	1,656	429	159,439	8,758
1898	505	130	111	2,354	2,383	364	164,862	20,105
1899	685	50	101	2,646	2,542	887	165,645	34,842
1900	420	30	150	2,331	4,193	833	185,811	68,427
1901	305	10	120	4,182	3,956	1,067	189,909	85,266
1902	337	67	*Nil.*	6,034	6,301	642	201,856	109,298
1903	381	42	1,088	4,675	6,277	507	245,294	108,959
1904	925	245	400	3,463	3,761	453	286,737	163,275
1905	221	77	1,140	5,805	5,725	290	296,605	159,519

Year.	Coal. (*Continued.*)			Coke.	Copper Ore.		Copper.		
	Bituminous.	Lignitic.	Briquettes.		Argentiferous.	Pyritic.	Fine.	Matte.	Precipitate.
1896	1,852,947	55,413	343,432	288,523	(*c*) 157,365	2,200,919	6	16,378	29,873
1897	2,010,960	54,232	332,272	755,394	(*c*) 18,488	2,161,182	7	16,120	29,652
1898	2,414,127	66,422	369,418	768,151	203	2,299,444	593	16,024	29,703
1899	2,565,437	70,901	348,838	341,443	1,103	2,443,044	4	15,755	41,927
1900	2,514,545	91,133	341,156	381,000	2,006	2,714,714	5	18,159	29,652
1901	2,566,591	95,867	338,684	455,586	(*b*)	2,672,365	79	15,634	28,433
1902	2,614,010	84,242	331,957	404,503	878	2,617,776	(*b*)	.....	36,045
1903	2,587,652	104,232	339,120	433,780	3,056	2,796,733	(*b*)	.....	27,448
1904	2,903,771	100,773	307,630	432,726	(*b*)	2,624,512	(*b*)	8,117	29,494
1905	2,912,466	168,994	290,830	448,073	(*b*)	2,621,054	(*b*)	8,243	17,988

Year.	Fluorspar.	Gold and Silver Ore.	Iron Ore.		Iron and Steel.			Kaolin. (China Clay.)
			Argentiferous.	Non-Argentiferous.	Pig.	Forged Iron.	Steel.	
1896	3	854	3,581	6,762,582	100,786	53,793	68,126	1,240
1897	2	2,456	5,559	7,419,768	146,940	80,894	66 007	6,294
1898	5	555	24,190	7,197,047	113,492	65,900	50,362	5,445
1899	310	(*d*) 1,110	17,139	9,397,733	113,071	40,332	112,982	2,790
1900	4	(*d*) 1,300	26,348	8,675,749	91,126	54,307	144,355	3,794
1901	(*b*)	(*d*) 1,595	27,726	7,906,517	135,600	47,085	121,023	2,220
1902	93	(*d*) 1,764	24,361	7,904,555	330,747	......	163,564	3,412
1903	4,000	(*d*) 2,681	90,996	8,304,153	380,284	......	199,642	2,578
1904	(*b*)	(*b*)	122,109	7,964,748	283,819	50,858	186,705	1,700
1905	(*b*)	(*b*)	152,027	9,007,245	305,462	11,366	223,545	720

Year.	Lead. (Argentiferous.)		Lead. (Non-Argentiferous.)		Manganese Ore.	Mineral Paints. (Ocher.)	Phosphate Rock.	Pyrites. (Iron)
	Ore.	Metal.	Ore.	Metal.				
1896	182,565	84,802	104,160	82,215	38,265	212	770	100,000
1897	186,692	91,258	110,496	75,112	100,566	200	2,084	100,000
1898	244,068	88,981	150,472	78,370	102,228	200	4,500	70,265
1899	184,906	70,874	123,753	91,739	104,974	100	3,510	107,386
1900	182,016	74,341	131,437	98,189	112,897	58	4,170	34,638
1901	207,188	73,895	174,376	⅜5,399	60,325	164	4,220	33,953
1902	227,645	74,370	100,403	103,190	46,069	(b)	1,150	145,173
1903	179,858	56,687	108,660	118,422	26,194	(b)	1,124	155,739
1904	177,104	57,956	93,230	127,906	18,732	(b)	3,305	161,841
1905	160,381	56,361	105,113	129,332	26,020	(b)	1,370	179,079

Year.	Pyrites (arsenical.)	Quicksilver.		Salt.	Silver.		Soapstone.
		Ore.	Metal.		Ore.	Metal. *Kg.*	
1896	(b)	34,959	1,524	521,751	1,230	64,554	756
1897	(b)	32,378	1,728	508,606	982	71,168	3,601
1898	230	31,361	1,691	479,358	967	76,295	2,613
1899	(b)	32,144	1,361	598,108	764	88,409	4,844
1900	515	30,216	1,095	450,041	742	140,457	8,109
1901	1,328	23,367	754	345,063	391	94,977	4,880
1902	5,648	26,037	1,425	426,434	175	96,975	542
1903	7,996	30,370	968	427,394	231	112,978	3,725
1904	3,510	27,185	1,130	543,658	303	117,418	5,165
1905	4,790	26,485	853	493,451	540	123,607	4,364

Year.	Sulphur.		Tin Ore (*Dressed.*)	Topaz. *Kg.*	Tungsten Ore.	Zinc.		
	Crude rock.	Refined.				Ore.	Spelter.	Sheets.
1896	26,204	1,800	(e)2,348	80	31	64,828	3,485	2,648
1897	18,805	3,500	(e)2,378	44	10	73,848	3,907	2,337
1898	34,943	3,100	4	90	37	99,836	4,300	1,731
1899	58,922	1,100	57	44	151	119,710	4,100	2,084
1900	64,364	750	47	95	1,958	86,158	2,855	2,756
1901	49,856	610	115	310	6	119,708	2,573	2,781
1902	15,442	450	12,762	*Nil.*	11	127,618	5,569	(b)
1903	38,573	1,680	330	90	*Nil.*	154,126	5,134	(b)
1904	40,389	605	229	....	60	156,329	5,887	2,913
1905	38,153	610	209	....	375	160,561	6,184	2,936

(a) Figures are from *Estadistica Minera de España* except for 1896 and 1898 which were from the official *Reports of the Junta Superior Facultativa de Minas*, Madrid. (b) Not reported. (c) Represents non-argentiferous copper ore. (d) Gold ore only. (e) Undressed tin ore.

SWEDEN.

The official statistics of mineral production, imports and exports are summarized in the following tables, 1905 being the latest year for which the reports have been published:

MINERAL PRODUCTION OF SWEDEN. (a)
(In metric tons.)

Year.	Alum.	Fire Clay.	Coal.	Copper.			Feldspar.	Gold-*Kg.*
				Ore.	Ingot.	Sulphate.		
1896	334	120,426	225,848	24,351	249	1,506	12,789	114.5
1897	131	112,283	224,343	25,207	289	1,315	19,298	113.3
1898	153	131,391	236,277	23,335	235	1,165	20,737	125.9
1899	164	129,875	239,344	22,334	179	1,287	16,017	106.2
1900	167	160,585	252,320	22,725	136	1,265	15,228	88.5
1901	121	175,876	271,509	23,660	137	1,224	13,502	62.7
1902	132	(b)	304,733	30,095	178	1,257	17,960	94.3
1903	140	172,718	320,390	36,687	776	1,171	19,392	50.6
1904	125	166,888	320,984	36,834	533	1,248	18,021	60.9
1905	139	119,947	322,384	39,255	1,385	1,029	19,224	55.0

Year.	Iron and Steel.					Steel.		
	Ore.	Pig.	Blooms.	Bars, Rods, Sheets, etc.	Iron Sulphate	Bessemer.	Basic.	Crucible.
1896	2,039,019	494,416	188,396	308,132	191	114,120	42,301	604
1897	2,086,119	538,197	189,633	304,537	232	107,679	165.836	691
1898	2,302,546	531,766	198,923	299,846	124	102,254	160,706	1,013
1899	2,434.606	497,727	195,331	328,999	105	91,898	179,357	1,225
1900	2,607,925	526,868	188,455	324,604	183	91,065	207,418	1,121
1901	2,793,566	528,375	164,850	269,507	140	77,231	190,877	1,088
1902	2,896,208	538,113	186,076	(b.)	127	84,014	201,311	1,091
1903	3,677,520	506,825	192,342	325,200	62	84,229	232,878	1,105
1904	4,083,945	528,525	189,246	324,676	148	78,577	252,832	1,162
1905	4,364,833	539,437	182,640	356,898	144	78,204	288,675	1,319

Year.	Lead.		Manganese Ore.	Pyrites.	Silver. lead Ore.	Silver—*Kg.*	Sulphur.	Zinc Ore.
	Ore.	Pig.						
1896	14	1,530	2,056	1,009	15,381	2,082	77	44,041
1897	99	1,480	2,749	517	10,068	2,218	(b)	56,636
1898	50	1,559	2,358	386	6,743	2,033	50	61,627
1899	35	1,606	2,622	150	5,730	2,290	(b)	65,159
1900	85	1,424	2,651	179	5,300	1,927	70	61,044
1901	56	988	2,271	*Nil.*	11,366	1,557	(b)	48,630
1902	63	843	2,850	*Nil.*	9,378	1,365	74	48,783
1903	25	678	2,244	7,793	9,792	1,005	(b)	62,927
1904	55	589	2,297	15,957	8,187	651	35	57,634
1905	40	576	1,992	20,762	8,397	606	....	56,885

(a) From *Bidrag till Sveriges Officiela Statistik Bergshandteringen.* (b) Not reported.

MINERAL IMPORTS OF SWEDEN. (a)
(In metric tons or dollars; 1 Krone=27 cents.)

Year.	Alum.	Aluminum Sulphate.	Ammonium.					Antimony (crude).	Arsenious Acid.
			Carbonate.	Chloride.	Hydrate.	Nitrate.	Sulphate.		
1895	93	348	74	84	76	11	39	81	36
1896	75	629	79	88	81	11	88	63	33
1897	103	733	109	110	59	42	67	58	33
1898	136	968	99	101	105	12	81	53	33
1899	158	866	89	112	110	12	181	59	12
1900	133	1,197	141	99	100	5	227	85	22
1901	346	1,192	131	113	92	1	285	50	12
1902	250	1,430	127	145	130	13	241	59	14
1903	302	1,082	113	133	150	47	197	54	21
1904	87	1,245	154	208	115	41	254	67	17
1905	106	1,127	142	170	70	26	189	67	16

Year.	Asbestos. (c)	Asphalt.	Barytes	Borax.	Boric Acid.	Bromine and Bromides. *Kg.*	Cement	Chalk. White. Unground. *Hectoliters.*
1895	68	3,256	240	111	54	3,999	4,270	4,466
1896	116	4,092	298	128	73	4,334	2,901	6,148
1897	119	5,458	270	175	56	5,549	1,826	14,368
1898	112	5,409	299	196	75	5,401	1,656	7,016
1899	567	6,286	292	190	65	4,914	1,363	16,079
1900	763	5,676	411	194	66	6,084	1,941	12,059
1901	178	4,524	295	253	68	6,602	2,868	13,569
1902	213	5,779	....	242	71	7,278	9,822	11,583
1903	217	5,957	....	240	71	7,419	11,145	41,868
1904	356	6,243	....	299	77	10,128	10,526	10,115
1905	140	4,760	264	294	82	18,788	10,999	13,305

Year.	Coal.	Copper, also Alloys.	Emery.	Gold.		Graphite.	Gypsum.	Iron (crude.)
				Bars and Mfres. *Kg.*	Specie.			
1895	1,911,629	2,930	125	89	$11,600	134	4,112	40,582
1896	1,991,760	4,037	104	1,161	608	135	4,940	34,549
1897	2,240,247	4,944	128	4,267	948	158	7,260	89,606
1898	2,392,451	5,227	131	3,998	2,396	167	7,979	76,832
1899	3,047,618	4,740	125	362	9,774	162	6,457	68,909
1900	3,033,885	4,745	136	3,365	98,905	213	6,794	82,957
1901	2,793,309	5,153	169	1,454	736,852	180	6,589	66,131
1902	2,911,286	6,890	147	945	Nil.	(b)	6,754	43,828
1903	3,192,990	6,109	132	89	965,346	(b)	8,795	49,411
1904	3,367,826	7,367	221	1,400	1,207,187	(b)	8,868	90.102
1905	3,297,485	6,481	271	1,765	750,009	(b)	11,270	87,843

Year.	Lead.	Lime *Hectoliters.*	Litharge.	Phosphorus. *Kg.*	Platinum. *Kg.*	Porcelain	Potassium.			
							Chloride.	Cyanide. *Kg.*	Hydrate.	Carbonate.
1895	1,624	4,436	117	71,407	42	277	561	1,457	91	1,979
1896	1,911	7,768	150	52,482	34	327	241	2,122	285	1,933
1897	2,098	20,050	199	57,972	63	362	363	2,922	1,381	1,432
1898	2,139	23,079	160	66,466	49	298	259	2,604	1,451	1,112
1899	2,125	34,343	177	59,989	59	346	225	2,313	1,266	1,231
1900	2,067	25,047	148	67,557	99	382	364	2,221	1,915	1,257
1901	1,991	12,204	165	70,672	172	386	260	2,658	1,435	1,266
1902	2,509	6,579	172	68,441	130	....	222	2,950	1,720	1,238
1903	2,644	6,449	237	112,659	116	....	245	3,294	2,034	1,150
1904	2,849	13,388	213	47,421	84	396	214	3,237	2,234	1,184
1905	2,823	16,099	205	69,526	105	416	1,296	3,437	2,251	1,133

Year.	Quick-silver. *Kg.*	Salt.		Silver.		Sodium.			
		Crude.	Refined.	Scrap and Mfres. *Kg.*	Specie.	Carbonate.	Hydrate.	Nitrate. (*d*)	Sulphate. (*e*)
1895........	6,318	106,820	2,535	3,050	$114,340	12,711	1,043	9,388	7,420
1896........	5,194	84,629	3,673	7,375	204,691	11,425	908	12,518	8,486
1897........	3,125	87,050	3,055	20,557	136,823	14,625	625	12,531	11,384
1898........	2,631	85,246	2,188	21,696	191,766	11,917	575	15,419	11,544
1899........	4,210	98,417	3,166	11,565	156,707	13,323	929	15,006	15,140
1900........	3,629	70,302	3,098	11,559	62,315	12,680	1,038	14,245	15,590
1901........	5,958	79,038	3,072	7,476	78,416	13,669	800	17,614	15,494
1902........	4,866	82,439	3,037	4,853	74,826	....	1,623	15,553	18,924
1903........	5,043	88,139	3,419	11,259	90,366	....	1,426	20,616	16,120
1904........	5,768	84,237	4,615	19,034	86,891	11,898	2,112	19,776	17,596
1905........	4,609	87,677	3,889	11.067	82,620	13,592	1,489	23,183	17,115

Year.	Sulphur.	Sulphuric Acid.	Tin.		Zinc.
			Salts-*Kg.*	Block.	
1895..	7,140	772	4,091	462	2,216
1896..	11,369	615	4,437	551	2,275
1897..	9,723	1,418	3,823	541	2,551
1898..	10,837	1,742	3,874	595	3,030
1899..	13,505	2,558	5,404	486	2,829
1900..	20,152	2,472	3,243	630	2,912
1901..	20,715	1,950	2,334	541	2,900
1902..	23,002	1,887	1,652	644	3,255
1903..	24,577	2,620	1,467	655	3,312
1904..	18,248	2,001	1,460	719	3,705
1905..	18,631	3,424	1.727	597	3.780

MINERAL EXPORTS OF SWEDEN. (*a*)
(In metric tons or dollars; 1 Krone=27 cents.)

Year.	Alum.	Ammonium Sulphate	Antimony, Crude-*Kg*	Asbestos. *Kg.*	Cement.	Coal.	Copper.		Graphite.
							Ore.	Copper & Alloys.	
1895................	38	(*b*)	2,300	28	31,029	52	(*b*)	2,283	9,749
1896................	40	100	800	2,040	22,991	141	1,094	1,911	3,500
1897................	54	180	800	1,348	27,112	74	(*b*)	933	7,215
1898................	32	36	4,700	1,055	28,676	496	1,102	1,346	9,108
1899................	26	2	2,600	2,812	31,101	762	315	1,230	16,664
1900................	24	2	4,600	2,436	42,564	1,108	448	2,012	17,719
1901................	56	156	1,800	2,179	17,794	716	602	1,243	16,761
1902................	20	174	4,090	1,864	19,499	866	845	1.516	5,420
1903................	22	*Nil*	3,473	15,357	21,319	509	1,555	1,858	8,744
1904................	9	219	3,810	16,339	27,509	605	749	1,396	(*b*)
1905................	12	445	3,147	2.386	38,504	425	2,137	2,6 4	(*b*)

Year.	Gypsum and Mfres.	Iron and Steel.		Lead and Mfres.	Lime. *Hectoliters.*	Peat.	Phosphorus. *Kg.*	Potassium Chloride.
		Ore.	Unwrought.					
1895.................	20	800,452	311,866	1,380	40,287	1,143	885	436
1896.................	9	1,150,695	304,138	1,182	102,787	1,452	1.510	254
1897.................	9	1,400,801	279,525	1,473	106,053	1,816	1,627	463
1898.................	27	1,439,860	301,192	570	123,939	1,616	4,085	506
1899.................	8	1,628,011	320,742	818	80,153	1,979	1,890	335
1900.................	10	1,619,902	304,175	1,209	84,242	3,843	879	931
1901.................	55	1,761,257	268,143	1,028	69,000	3,064	1,254	708
1902.................	117	1,729,000	(*f*) 73,403	546	81,796	3,620	1,290	1,114
1903.................	119	2,828,000	(*f*) 70,788	333	63,870	3,217	300	790
1904.................	162	3,065,522	(*f*) 88,124	275	56,417	4,212	1,994	1,266
1905.................	156	3,316,626	(*f*)120.987	512	113.702	5,1 7	34,388	14,99

Year.	Salt, Refined Kg.	Silver. Kg.		Soda.	Sulphur.	Tin.		Zinc.	
		Bullion.	Mfres.			Block.	Mfres-Kg.	Ore.	Crude and Mfrs
1895	1,661	751	10	959	11	13.3	786	33,075	26
1896	830	819	14	772	9	18.9	2,996	41,401	184
1897	1,424	329	119	686	11	25.6	7,113	44,425	135
1898	216	130	238	509	11	20.8	1,263	49,597	184
1899	110	367	258	227	68	8.8	1,033	45,634	157
1900	407	296	103	238	20	21.5	1,521	40,879	156
1901	1,556	179	9	237	12	20.4	8,110	41,248	101
1902	1,945	110	506	621	147	25.5	1,603	43,813	63
1903	*Nil*	484	195	10	217	43.3	3,893	45,389	351
1904	1,883	115	69	45	4	45.6	3,479	44,259	332
1905	*Nil*	10	77	403	4	33.9	654	51,765	295

(*a*) From *Bidrag till Sveriges Officiella Statistik.* (*b*) Not reported. (*c*) Includes crude and manufactures. (*d*) Includes a small quantity of potassium nitrate. (*e*) Includes sodium bisulphate. (*f*) Includes only crude or ballast iron.

UNITED KINGDOM.

The statistics of the mineral production, imports and exports, are given in the subjoined tables.

MINERAL AND METALLURGICAL PRODUCTION OF THE UNITED KINGDOM. (a)
(In metric tons.)

Year.	Alum Shale.	Arsenious Acid.	Arsenical Pyrites.	Barytes.	Bauxite.	Chalk.	Clay. (e)	Coal.
1897	621	4,232	13,347	23,087	13,540	3,920,183	12,908,479	205,364,010
1898	13,835	4,241	11,272	22,581	12,600	4,366,782	14,974,290	205,287,388
1899	5,913	3,890	13,735	25,059	8,137	4,752,982	15,305,895	223,616,279
1900	1,329	4,146	9,727	29,937	5,871	4,444,765	14,279,181	228,772,886
1901	4,019	3,416	2,620	26,844	10,357	4,399,043	14,393,196	222,614,981
1902	5,755	2,165	842	23,986	9,192	4,466,004	15,549,002	230,728,562
1903	3,337	916	58	24,659	6,226	4,541,494	16,460,526	234,019,821
1904	6,636	992	44	26,748	8,839	4,509,768	16,210,734	236,130,373
1905	7,245	1,552	651	29,528	7,417	4,608,153	15,376,910	239,906,999
1906	9,605	1,625	620	35,586	6,760	4,825,299	12,459,213	255,067,622

Year.	Copper.		Fluorspar.	Gold.		Gravel and Sand.	Gypsum.	Bog Ore (c)
	Ore and Precipitate.	Fine.		Ore.	Bullion—*Kg.*			
1897	7,470	526	302	4,589	63.2	1,378,496	184,287	7,238
1898	9,277	650	57	715	12.3	1,652,701	199,174	5,505
1899	8,452	647	796	3,096	103.5	1,800,208	215,974	4,390
1900	9,643	777	1,472	21,135	437.6	1,867,211	211,436	4,221
1901	6,903	541	4,232	16,641	194.5	1,990,926	204,045	2,649
1902	6,210	490	6,388	30,432	130.0	2,100,829	228,264	4,983
1903	6,977	545	12,102	29,057	171.0	2,281,689	223,426	4,156
1904	5,552	501	18,450	23,574	610.7	2,275,426	237,749	4,616
1905	7,267	(b)	38,606	16,237	(b)	2,277,486	259,596	3,256
1906	7,882	(b)	36,860	17,662	(b)	2,392,012	228,627	5,512

Year.	Iron.		Lead.		Manganese Ore.	Mineral Paints.	Oil Shale.	Phosphate of Lime.
	Ore.	Pig.	Ore.	Pig.				
1897	14,008,484	4,942,679	35,903	26,988	609	14,653	2,259,325	2,032
1898	14,403,769	4,928,347	33,513	25,761	235	20,144	2,172,201	1,575
1899	14,692,711	4,992,468	31,494	23,929	422	16,575	2,246,197	1,469
1900	14,257,344	4,743,172	32,487	24,762	1,384	15,448	2,318,736	630
1901	12,475,700	4,158,745	33,084	20,361	1,673	14,780	2,392,812	71
1902	13,641,459	4,470,420	25,000	17,988	1,299	17,235	2,141,355	87
1903	13,935,748	4,573,202	26,993	20,278	831	14,377	2,041,851	71
1904	13,994,670	4,596,803	26,796	20,155	8,896	16,307	2,370,391	59
1905	14,824,183	(f) 9,746,221	28,091	(b)	14,582	16,468	2,536,734	*Nil*
1906	15,748,412	(f) 10,210,178	30,210	(b)	23,126	14,437	2,587,266	*Nil*

Year.	Pyrites.	Salt.	Silica. (chert and flint.)	Silver. *Kg.*	Stone. Granite.	Limestone.(*d*)	Sandstone.	Slate.
1897	10,752	1,933,949	95,209	7,750	1,876,880	11,179,580	5,043,535	618,941
1898	12,302	1,908,723	83,370	6,575	1,905,830	12,172,267	5,325,988	679,461
1899	12,426	1,945,531	69,955	5,969	4,785,284	12,499,736	5,296,026	650,077
1900	12,484	1,873,601	78,971	5,964	4,709,997	12,099,940	5,101,868	595,428
1901	10,405	1,812,180	132,700	5,452	5,131,787	11,363,202	5,199,234	496,756
1902	9,315	1,893,881	100,938	4,560	5,554,696	12,368,196	5,571,121	525,665
1903	9,794	1,917,274	74,355	5,440	5.512 605	12,419,120	5,496,312	540,143
1904	10,452	1,921,899	66,300	4,967	6,084,642	12,235,825	5,391,265	572,181
1905	12,381	1,920,149	71,808	(*b*)	6 052,210	12,701,808	5,729,799	523,892
1906	11,318	1,996,593	....	.....	....	12,962,725	5,345,328	500,546

Year.	Strontium Sulphate	Tin. Ore, Dressed.	Tin. Block.	Tungsten Ore.	Uranium Ore.	Zinc. Ore.	Zinc. Spelter.
1897	15,227	7,234	4,524	127	30	18,586	7,162
1898	13,148	7,498	4,722	331	26	23,929	8,711
1899	12,831	6,494	4,077	96	7	23,505	8,837
1900	9,270	6,911	4,337	9	42	25,070	9,214
1901	16,923	7,407	4,634	21	80	23,967	8,555
1902	32,799	7,681	4,462	9	53	25,462	9,275
1903	23,209	7,500	4,351	276	6	25,287	9,430
1904	18,460	6,849	4,198	164	Nil.	28,097	10,427
1905	14 523	7,316	(*b*)	174	105	24,025	(*b*)
1906	14,338	7,267	.....	255	11	23,189	(*b*)

(*a*) From *Mineral Statistics of the United Kingdom*. (*b*) Not reported. (*c*) Bog ore, which is raised in Ireland, is an ore of iron, used principally for purifying gas. (*d*) Does not include chalk. (*e*) Includes China clay, potters' clay, and fuller's earth. (*f*) Includes production from imported ore.

MINERAL IMPORTS OF THE UNITED KINGDOM. (*a*)

(In metric tons or dollars; £1=$5.)

Year.	Alkali.	Asphaltum.	Borax.	Brass and Bronze Mfres.	Clay Products, Porcelain and Earthenware.	Coal, Coke and Pat. Fuel.	Copper. Ore.	Copper. Regulus and Precipitate	Copper. Wrought, Unwrought and Old.
1897	11,557	44,541	(*b*)	2,129	17,847	9,605	83,916	90,008	62,055
1898	12,179	46,398	1,255	2,357	16,405	11,191	91,141	76,201	70,018
1899	12,078	59,073	3,076	1,988	18,341	1,777	130,611	84,015	60,502
1900	16,360	53,061	15,667	2,335	18,329	10,112	102,365	89,123	72,223
1901	(*c*)13,429	74,694	15,710	2,469	20,754	7,685	102,503	93,338	68,809
1902	(*c*)26,292	65,896	13,390	3,403	19,029	3,331	90,007	74,684	92,349
1903	(*c*)14,321	(*b*)	11,959	(*b*)	18,576	3,535	85,644	77,884	64,591
1904	(*c*) 14,325	(*b*)	16,012	(*b*)	18,980	2,812	80,771	67,739	90,717
1905	(*c*)16,593	(*b*)	11,552	(*b*)	20,076	49,277	94,198	70,235	71,294
1906	(*c*)14,070	(*b*)	16,955	(*b*)	22,400	49,269	97,789	76,073	75.487

Year.	Diamonds.	Glass. All Kinds.	Gold. Ore.	Gold. Bullion and Specie.	Gold. Leaves. *Number.*	Iron and Steel. Iron Ore.	Iron and Steel. Pig Iron.	Iron and Steel. Scrap.
1897	$22,176,300	$15,034,555	$736,415	$154,044,290	68,173,400	6,064,179	(*f*)160,531	20,735
1898	22,619,075	16,423,915	1,393,175	218,614,800	63,632,700	5,555,889	(*f*)162,075	24,619
1899	20,597,910	16,044,400	761,755	162,667,485	49,108,570	7,168,061	(*f*)174,159	32,427
1900	17,168,180	15,997,555	1,432,970	130,954,365	54,346,025	6,398,639	178,199	31,687
1901	24,385,210	17,647,635	2,747,195	103,578,140	59,048,856	5,637,670	198,536	44,721
1902	26,901,950	18,484,960	2,688,170	108,145,245	59,334,023	6,542,793	226,708	39,584
1903	(*b*)	18,636,475	3,590,095	143,286,965	(*b*)	6,417,188	132,364	17,051
1904	(*b*)	16,901,500	4,259,360	169,382,940	(*b*)	6,198,368	132,494	19,326
1905	(*b*)	16,992,835	(*b*)	192,839,475	(*b*)	7,172,171	128,183	23,569
1906	(*b*)	16.346,330	(*b*)	230,212,950	(*b*)	7,634,839	90.674	36,559

Year.	Iron and Steel. (*Continued.*)							
	Puddled and Wrought.	Sheets and Plates.	Rails.	Strips and Wire Rods.	Nails, Screws. Rivets, Bolts.	Steel Ingots, Blooms, Billets, etc.	Steel Bars, Shapes, Beams and Pillars	Mnfrs. Unenumerated (*k*)
1897	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	40,628	(*h*)	$27,894,295
1898	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	40,875	(*h*)	33,379,160
1899	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	78,257	(*h*)	39,527,075
1900	189,891	(*g*)	38,636	(*g*)	(*g*)	182,210	94,667	17,861,660
1901	102,811	(*g*)	55,809	(*g*)	(*g*)	185,810	124,648	18,252,435
1902	178,425	(*g*)	48,942	(*g*)	45,095	285,494	129,743	14,424,835
1903	196,084	73,079	74,939	35,574	51,888	278,441	343,259	9,007,990
1904	109,289	69,552	40,438	38,214	50,649	531,069	219,510	8,630,380
1905	110,576	69,831	34,439	60,318	55,331	613,612	148,995	7,783,895
1906	111,062	83,747	11,900	61,288	57,071	493,805	149,363	3,824,405

Year.	Lead.		Manganese Ore.	Mica, Sheet.	Mica and Talc.	Paraffin.	Petroleum. *Liters.*	Phosphate Rock.
	Ore.	Pig and Sheet.						
1897	32,818	170,121	158,825	412	1,683	39,284	842,920,307	330,335
1898	44,457	197,591	156,390	517	1,398	48,104	829,995,751	334,884
1899	30,263	201,551	261,740	519	6,025	54,712	908,107,248	426,830
1900	21,566	198,416	270,098	469	7,952	50,033	965,167,850	361,309
1901	29,944	221,549	195,736	(*b*)	7,117	42,643	960,650,967	360,568
1902	25,838	235,522	237,066	1,078	6,127	52,023	1,078,095,152	370,697
1903	18,923	232,939	235,574	(*b*)	(*b*)	49,163	1,299,570,625	398,997
1904	8,748	250,452	208,458	(*b*)	(*b*)	42,882	1,373,488,176	425,978
1905	(*b*)	233,214	289,827	(*b*)	(*b*)	41,247	1,364,301,583	427,762
1906	(*b*)	211,577	314,016	(*b*)	(*b*)	44,673	1,130,667,737	450,058

Year.	Paints, Mineral.	Platinum Wrought and Unwrought. *Kg.*	Potassium Nitrate.	Pyrites of Iron and Copper.	Quicksilver.	Silver Ore. (*e*)	Sodium Nitrate.
1897	$5,049,225	2,257	16,744	633,009	1,862	$7,149,210	107,525
1898	5,619,835	3,389	13,323	665,544	1,856	5,729,525	132,412
1899	5,842,063	5,404	12,635	712,393	1,759	5,162,750	163,387
1900	6,747,800	5,027	12,798	752,605	1,113	5,154,430	143,461
1901	6,489,710	4,917	12,115	664,041	1,202	5,309,920	108,822
1902	6,367,320	3,027	11,526	620,948	1,129	5,383,515	116,791
1903	7,025,050	(*b*)	9,425	747,714	1,187	6,596,045	118,582
1904	6,718,095	(*b*)	12,277	754,722	1,130	8,271,480	122,454
1905	7,168,725	(*b*)	8,260	709,926	1,158	10,426,570	106,107
1906	7,534,495	(*b*)	10,125	771,473	1,320	10,532,020	110,222

Year.	Stone and Marble, Hewn or Mfr'd,	Sulphur.	Tin.		Zinc.		
			Ore.	Block, Ingot, Bars or Slabs.	Ore.	Spelter.	Mnfres.
1897	752,345	22,811	5,345	27,214	25,238	70,929	21,395
1898	883,699	19,642	5,710	20,665	53,945	78,761	21,613
1899	905,432	21,906	6,324	27,608	38,143	71,068	21,521
1900	961,492	22,993	7,449	33,648	42,755	61,504	21,751
1901	1,159,276	22,440	10,690	35,397	38,660	68,633	21,343
1902	1,192,023	23,863	12,255	35,713	45,312	89,688	21,717
1903	1,219,782	21,313	12,473	36,076	41,009	86,539	23,118
1904	1,329,502	17,629	15,734	39,932	54,438	90,088	22,788
1905	1,217,652	18,163	(*b*)	40,391	(*b*)	92,261	20,013
1906	1,245,174	22,704	(*b*)	44,306	(*b*)	95,203	19,664

(*a*) From *Accounts Relating to Trade and Navigation of the United Kingdom.* (*b*) Not reported. (*c*) Classified as soda compounds since 1901. (*d*) Includes machinery and mill work. (*e*) Includes the value of silver in argentiferous ore and metal. (*f*) Includes puddled iron. (*g*) Not separately enumerated. (*h*) Former returns not available. (*k*) Prior to 1900 many manufactures were not reported separately.

MINERAL EXPORTS OF THE UNITED KINGDOM—DOMESTIC PRODUCTS. (*a*)
(In metric tons or dollars; £1=$5.)

Year.	Ammonium Sulphate.	Bleaching Materials.	Brass and Mfres.	Cement.	Clay. Unmanufactured.	Clay products. (*c*)	Coal.
1897	155,449	(*b*)	5,713	398,023	(*b*)	$8,629,425	35,919,965
1898	139,806	(*b*)	5,418	331,648	(*b*)	8,253,640	35,619,365
1899	142,577	(*b*)	5,797	359,273	(*b*)	9,314,255	41,839,217
1900	147,528	57,478	6,131	365,742	(*b*)	10,193,970	46,845,739
1901	154,282	46,912	5,905	318,216	(*b*)	9,963,985	42,547,114
1902	165,213	40,939	6,378	308,104	463,309	9,497,995	43,849,591
1903	....	49,415	7,646	406,388	490,157	10,879,940	45,669,258
1904	....	35,289	8,860	390,736	527,013	10,531,620	46,995,636
1905	....	42,526	12,526	463,863	573,706	10,487,585	48,236,334
1906	....	45,510	13,569	668,461	611,919	11,914,095	56,489,367

Year.	Coke.	Patent Fuel.	Supplied to Steamers.	Coal Products (*d*)	Copper. Ingot.	Copper. Mixed or Yellow Metal	Copper. Mfres.	Copper. Sulphate.
1897	993,980	(*b*)	10,623,050	$8,340,420	21,252	11,192	15,275	60,326
1898	782,053	(*b*)	11,444,431	7,624,740	27,102	10,452	13,765	52,573
1899	881,172	(*b*)	12,422,429	7,712,965	32,449	7,038	11,231	40,822
1900	1,001,131	(*b*)	11,940,353	9,058,220	18,300	8,940	10,765	43,601
1901	820,594	1,098,459	13,804,222	5,756,265	26,935	9,252	11,156	36,601
1902	669,664	1,067,060	15,390,485	5,991,025	21,658	13,314	14,075	43,995
1903	728,957	970,449	17,068,646	7,290,825	23,723	14,425	16,975	54,307
1904	779,060	1,257,589	17,465,954	6,879,400	14,791	16,704	18,467	71,367
1905	786,496	1,126,190	17,674,484	6,742,455	21,232	9,959	22,128	55,219
1906	828,268	1,399,244	18,887,656	7.237,120	19,778	7,149	16,195	43,670

Year.	Fertilizers.	Glass. All kinds.	Iron. Ore.	Iron. Pig.	Iron. Scrap.	Iron. Cast Iron and Mfres.	Iron. Wrought Iron, Shapes and Mfres.	Iron. Rails.
1897	(*e*)	4,359,535	(*e*)	(*f*)1,219,958	99,259	(*b*)	170,285	(*g*)795,983
1898	(*e*)	4,410,205	(*e*)	(*f*)1,058,973	86,602	(*b*)	152,911	(*g*)619,976
1899	(*e*)	4,578,415	(*e*)	(*f*)1,401,365	118,262	(*b*)	161,679	(*g*)601,266
1900	$12,041,450	5,172,910	(*e*)	(*f*)1,450,365	96,567	(*b*)	159,677	379,939
1901	11,987,555	5,285,775	(*e*)	(*f*) 852,609	86,559	(*b*)	119,962	474,073
1902	13,862,875	5,489,650	4,062	1,120,207	104,890	(*b*)	(*b*)	(*b*)
1903	14,254,340	5,512,470	4,534	1,082,426	143,929	62,249	217,139	613,741
1904	14,869,250	5,049,495	6,706	823,909	166,010	49,004	173,233	533,895
1905	14,425,085	5,558,820	14,664	997,601	151,619	49,193	186,340	555,390
1906	18,164,135	6,393,930	13,415	1,670,753	180,547	54,876	203,521	470,652

Year.	Iron. (*Continued.*) Wire and Mfres. of.	Plates and Sheets.	Galvanized Sheets.	Black Plates for Tinning.	Tinned Plates.	Steel Ingots, Billets, Blooms, etc	Steel Shapes Beams and Pillars.	Total Iron and Steel and Mfres. of.	Lead. Pig and Mfres.
1897	52,471	120,868	231,319	59,663	276,260	304,249	(*b*)	3,750,122	40,911
1898	44,954	102,638	230,219	59,289	255,797	290,182	(*b*)	3,299,326	38,684
1899	50,041	111,773	242,167	86,936	260,735	333,837	(*b*)	3,777,098	40,923
1900	39,104	39,157	251,203	66,810	278,338	313,383	(*b*)	3,602,083	36,576
1901	48,107	36,418	254,290	52,217	275,661	217,236	(*b*)	2,944,083	38,166
1902	(*b*)	(*b*)	336,572	58,245	317,201	306,152	(*b*)	3,529,223	33,537
1903	60,800	165'672	357,665	66,279	297,485	13,427	159,330	3,621,635	36,152
1904	61,894	154,774	391,608	63,467	365,262	4,324	176,232	3,315,047	35,600
1905	82,519	207,866	413,533	69,937	360,630	8,735	219,491	3,781,059	42,265
1906	96,641	279,459	450,221	66,749	381,421	11,924	311,231	4,763,868	45,612

Year.	Salt.	Sodium. Soda Ash.	Sodium. Carbonate and Bicarbonate.	Sodium. Hydrate.	Sodium. Sulphate.	Tin. Block.	Zinc. Ore.	Zinc. Spelter.	Zinc. Mfres.
1897	680,477	(*h*)252,736	(*i*)	(*i*)	(*i*)	5,050	6,072	6,951	1,047
1898	698,882	(*h*)191,578	(*i*)	(*i*)	(*i*)	5,557	6,483	7,577	1,227
1899	638,213	(*h*)193,492	(*i*)	(*i*)	(*i*)	4,785	8,171	5,492	1,249
1900	556,704	(*h*)185,783	(*i*)	(*i*)	(*i*)	5,713	13,913	7,136	1,159
1901	627,078	58,412	22,161	50,624	26,057	5,584	13,981	7,512	1,256
1902	624,752	59,894	24,654	61,658	35,672	6,210	16,717	6,756	1,345
1903	594,300	58,605	23,574	59,725	45,630	6,349	15,659	8,102	(*k*)
1904	632,605	61,327	25,252	61,985	40,324	5,953	14,606	7,993	(*k*)
1905	588,389	67,678	28,425	68,675	33,681	7,741	(*b*)	7 451	(*k*)
1906	629,658	86,232	26,970	72,218	44,448	8,631	(*b*)	7,962	(*k*)

(*a*) From *Accounts Relating to Trade and Navigation of the United Kingdom.* (*b*) Not reported. (*c*) Comprises porcelain and earthenware. (*d*) Including naphtha, paraffin, paraffin oil and petroleum. (*e*) Previous reports not available (*f*) Includes puddled iron. (*g*) Includes railroad material of all kinds. (*h*) Includes all soda compounds; not separate; enumerated previous to 1901. (*i*) Included under soda ash. (*k*) Included under spelter.

UNITED STATES.

Of the following three tables, the first records the imports of foreign mineral and metal products into the United States, whether dutiable or duty free; the second shows the exports of materials produced in the United States; and the third reports the re-exports of products of foreign origin. The statement of production in the United States is given on an early page in this volume. These statistics are as reported by the Bureau of Statistics of the Department of Commerce and Finance, and special acknowledgment is due to Hon. O. P. Austin, chief of the bureau, for furnishing the figures for many substances which are not reported in the "Monthly Summary."

IMPORTS (a)

Year.	Aluminum. Crude. Lb.	Aluminum. Crude. Kg.	Aluminum. Crude. Value.	Aluminum. Crude. Value per Lb.	Aluminum. Mfrd. Value.	Ammonium Sulphate. Lb.	Ammonium Sulphate. Metric Tons.	Ammonium Sulphate. Value.	Ammonium Sulphate. Value per Lb
1900........	256,559	116,374	$44,455	$0.172	$3,111	24,024,188	10,897	$591,937	$0.025
1901........	564,803	251,657	104,168	.186	5,580	31,711,085	14,384	728,085	.023
1902........	745,217	338,028	215,032	.290	3,645	35,535,558	16,119	858,036	.024
1903........	498,655	226,190	139,298	.279	4,273	29,104,817	13,199	765,230	.026
1904........	515,416	234,293	128,350	.249	478	39,859,690	18,077	1,058,981	.027
1905........	530,429	240,284	106,108	.200	33	15,512,399	7,038	416,048	.027
1906........	770,713	349,195	154,292	.200	1,865	46,834,866	21,231	1,303,987	.028

Year.	Antimony. Lb.	Antimony. Metric Tons.	Antimony. Value.	Antimony. Value per Lb.	Antimony Ore. Lb.	Antimony Ore. Metric Tons.	Antimony Ore. Value.	Antimony Ore. Value per Lb.
1900..................	3,632,843	1,648	$285,749	$0.079	6,035,734	2,738	$78,581	$0.013
1901..................	3,674,923	1,667	255,346	.069	1,731,756	786	24,256	.014
1902..................	5,742,703	2,605	347,899	.061	1,639,043	743	29,476	.018
1903..................	5,125,515	2,325	279,957	.054	2,673,142	1,213	51,489	.019
1904..................	4,056,299	1,840	235,401	.058	2,487,602	1,129	50,362	.020
1905..................	5,737,891	2,603	431,774	.075	1,976,694	897	52,868	.027
1906..................	7,900,194	3,583	1,417,816	.179	2,247,131	1,019	128,347	.057

Year.	Asbestos. Crude Value.	Asbestos. Mfd. Value.	Asbestos. Total Value.	Asphaltum. Long Tons.	Asphaltum. Metric Tons.	Asphaltum. Value.	Asphaltum. Value per L. T.
1900...........................	$331,796	$24,155	$355,951	113,557	115,374	$404,921	$3.57
1901...........................	667 087	24,741	691,828	132,079	134,192	516,515	3.85
1902...........................	729,421	33,013	762,434	139,944	142,183	439,570	3.09
1903...........................	657,269	32,058	689,327	167,554	170,235	514,051	3.06
1904...........................	700,572	51,290	751,862	119,575	121,489	510,524	4.27
1905...........................	776,362	70,117	846,479	86,748	88,136	382,667	4.41
1906...........................	536,500	....	....	97,274	98,830	388,010	3.93

Year.	Arsenic. (b)				Barytes.				Bauxite.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900....	(f)	.....	.....	.....	(f)	.....	.....	.....	8,656	8,795	$32,967	$3.81
1901....	(f)	.....	.....	.....	(f)	.....	.....	.....	17,866	18,153	66,107	3.70
1902....	(f)	.....	.....	.....	(f)	.....	.....	.....	15,790	16,043	54,410	3.45
1903....	7,391,566	3,241	$256,097	$0.036	6,344	6,446	$22,777	$3.59	14,889	15,127	49,684	3.34
1904....	6,391,566	2,900	226,481	0.036	6,689	6,796	27,463	4.11	15,475	15,723	49,577	3.20
1905....	6,444,083	2,924	219,198	0.034	7,879	8,005	36,796	4.67	11,726	11,914	46,517	3.96
1906....	7,639,507	3,464	336,609	.044	4,293	4,362	37,296	8.69	17,809	18,094	63,221	3.55

Year.	Brass and Mfres. of, Value.	Chloride of Lime.				Cement.			
		Lb.	Metric Tons.	Value.	Value per Lb.	Barrels. (c)	Metric Tons.	Value.	Value per Bbl.
1900......	$20,113	132,520,478	60,111	$1,524,205	$0.012	2,386,684	433,037	$3,330,453	$1.40
1901......	35,976	120,611,346	54,709	1,673,190	.014	944,892	170,431	1,305,692	1.38
1902......	51,626	112,374,478	50,973	1,456,435	.013	1,994,790	361,932	2,581,883	1.29
1903......	206,905	113,285,240	51,586	912,843	.008	2,317,951	420,569	3,027,111	1.30
1904......	310,311	87,909,168	39,876	707,174	.008	1,046,404	189,910	1,382,913	1.32
1905......	1,169,320	104,919,462	47,604	843,285	.008	846,577	153,644	1,102,041	1.30
1906......	1,891,126	105,221,371	47,718	863,490	.008	2,205,712	400,115	2,950,268	1.33

Year.	Chrome Ore.				Bismuth.				Clays and Earths.			
	Long Tons.	Metric Tons.	Value.	Value per L.T	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900..........	17,572	17,823	$305,001	$17.38	....	....	....	......	146,524	148,868	$966,579	$6.59
1901..........	20,112	20,434	363,108	18.04	....	....	....	......	181,013	183 589	1,176,633	6.51
1902..........	39,570	40,203	582,597	14.72	....	....	....	......	191,764	194,832	1,228,945	6.41
1903..........	22,932	23,299	302,025	13.17	147,324	66,826	235,199	$1.59	203,173	206,424	1,264,544	6.22
1904..........	24,227	24,615	348,527	14.38	147,712	67,002	268,837	1.82	191,853	194,923	1,168,552	6.09
1905..........	54,434	55,305	725,301	13.32	148,589	67,459	318,007	2.14	321,641	326,787	1,363,760	4.24
1906..........	43,441	44,136	557,594	12.84	254,733	115,000	318,452	1.25	260,481	236,256	1,586,938	6.09

Year.	Coal, Anthracite.				Coal, Bituminous.				Total Coal.	
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Value.
1900.....	118	120	$549	$4.65	1,909,258	1,939,806	$5,019,553	$2.63	1,909,366	5,020,102
1901.....	286	291	1,844	6.45	1,919,962	1,950,681	5,291,429	2.75	1,920,248	5,293,273
1902.....	73,006	74,174	323,517	4.43	2,478,375	2,518,029	7,012,674	2.84	2,551,381	7,339,791
1903.....	151,023	153,439	675,623	4.47	3,295,379	3,348,105	9,329,221	2.83	3,446,402	10,004,844
1904.....	72,526	73,686	220,665	3.04	1,556,149	1,581,047	3,915,613	2.52	1,628,675	4,136,278
1905.....	34,262	34,810	107,394	3.13	1,618,581	1,644,478	3,908,877	2.42	1,652,843	4,016,271
1906.....	32,357	32,875	105,190	3.25	1,712,150	1,739,544	4,129,555	2.41	1,744,507	4,234,745

Year.	Coke.				Cobalt Oxide.				Copper, Ore and Matte.			
	Long Tons.	Metric Tons.	Value.	Value pr L.T.	Lb.	Kg.	Value.	Value pr Lb.	Long Tons.	Metric Tons.	Value	Value per L.T.
1900..........	103,175	104,826	$371,341	$3.60	54,073	24,527	$88,651	$1.64	54,329	55,201	$5,195,010	$92.23
1901..........	172,729	173,893	266,078	3.67	71,969	32,645	134,208	1.86	96,047	97,584	14,692,645	152.99
1902..........	107,437	109,156	423,774	4.05	79,984	36,281	151,115	1.89	181,566	184,470	8,695,780	47.89
1903..........	127,479	129,519	437,625	3.43	73,350	33,272	145,264	1.98	284,912	289,471	3,177,582	11.15
1904..........	161,476	164,060	648,520	4.01	42,352	19,211	86,925	2.05	268,234	272,527	4,308,410	16.06
1905..........	181,376	184,278	796,544	4.39	70,048	31,802	139,377	1.99	296,251	300,991	5,765,238	19.46
1906..........	114,703	116,538	558,419	4.87	41,084	18,652	83,167	2.02	209,696	213,051	6,845,870	32.65

Year.	Copper, Ingots, Old, etc.				Copper Mfres.	Total Copper.	Cryolite.			
	Lb.	Metric Tons.	Value.	Value per Lb.			Long Tons.	Metric Tons.	Value.	Value per L. T.
1900..........	68,796,808	31,206	$10,557 870	$0.153	$23,390	$15,776,270	5,437	5,524	$72,763	$13.38
1901..........	73,826,406	33,488	11,812,216	.160	24,775	26,529,636	5,383	5,469	70,886	13.17
1902..........	103,129,568	46,778	13,051,159	.126	52,464	21,799,403	6,188	6,287	85,640	13.83
1903..........	136,707,995	62,011	17,262,148	.126	31,624	20,471,354	7,708	7,831	102,879	13.35
1904..........	142,344,433	64,567	18,374,959	.129	37,913	22,721,282	959	974	13,708	14.30
1905..........	160,619,385	72,876	22,103,741	.137	117,404	27,986,383	1,600	1,623	22,482	14.05
1906..........	176,558,390	80,069	30,416,578	.172	62,191	31,324,639	1,505	1,529	29,683	19.72

Year.	Ceramics.	Emery Grains.				Emery Rock.				Emery Mfres.	Total Emery.
		Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value. per L. T.		
1900....	$9,143,536	661,482	300	$26,520	$0.040	11,392	11,574	$202,980	$17.82	10,006	$239,506
1901....	9,816,074	1,116,729	506	43,207	.039	12,441	12,640	240,856	19.35	10,927	294,990
1902....	9,838,426	1,665,737	756	60,079	.036	7,166	7,281	151,959	21.21	13,776	225,814
1903....	11,582,013	3,595,239	1,630	109,272	.030	10,885	11,059	188,985	17.36	23,317	321,574
1904 ..	11,656,686	2,281,193	1,035	109,772	.048	7,054	7,167	131,493	18.64	19,059	260,324
1905....	12,199,605	3,209,915	1,456	143,729	.045	11,073	11,250	185,689	16.77	17,996	347,414
1906...	....	4,655,168	2,113	215,357	.043	13,840	14,061	286,386	20.69	19,105	521,848

Year.	Fertilizers.									Fuller's Earth.	
	Guano.				Crude Phosphates.				All Other.		
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric. Tons.	Value.	Value. per L. T.		Long Tons.	Value.
1900....	6,620	6,726	$67,413	$10.18	137,086	139,272	$791,189	$5.77	$1,400,336	....	....
1901....	4,949	5,028	71,140	14.37	175,765	178,577	872,503	4.97	1,506,965	....	....
1902....	8,407	8,542	164,783	19.60	137,386	139,584	646,264	4.70	1,725,333	....	....
1903....	21,007	21,343	251,966	12.00	132,965	134,092	679,112	5.11	2,353,496	15,267	$120,671
1904....	35,876	36,430	478,388	13.33	130,214	132,297	745,744	5.73	2,856,141	9,126	78,006
1905....	25,651	26,061	365,823	14.26	56,021	56,917	273,289	4.88	4,051,003	13,001	105,997
1906....	22,947	22,314	320,565	13.97	23,281	23,653	147,547	6.34	4,231,723	13,238	108,696

Year.	Gold.		Iron Ore.				Pig Iron.			
	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900...	$45,703,256	$21,045,828	879,831	893,908	$1,303,196	$1.48	52,565	53,406	$1,907,361	$36.28
1901...	33,237,629	21,524,251	966,950	982,421	1,659,273	1.72	62,930	63,937	1,792,014	28.48
1902...	22,710,957	21,482,360	1,165,470	1,184,118	2,583,077	2.22	619,354	629,264	10,935,831	17.66
1903..	44,054,902	21,212,794	980,440	996,127	2,261,008	2.31	599,574	609,167	11,173,302	18.64
1904..	75,646,128	9,157,106	487,613	495,415	1,101,384	2.26	79,590	80,772	1,765,107	22.20
1905...	38,564,328	11,729,077	845,651	859,181	2,062,161	2.44	212,465	215,864	5,185,784	24.41
1906..	139,705,887	15,873,493	1,060,390	1,077,356	2,967,434	2.80	379,828	385,905	11,851,210	31.20

Year.	Scrap, Iron and Steel.				Bar Iron.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900..................	34,431	34,982	$663,231	$19.26	19,685	19,094	$1,058,761	$56.34
1901..................	20,130	20,452	339,827	16.88	20,792	21,126	1,093,736	52.60
1902..................	109,510	111,262	1,606,720	14.67	28,844	29,307	1,286,238	44.58
1903..................	82,921	84,248	1,273,941	15.36	43,392	44,090	1,904,469	43.89
1904..................	13,461	13,676	189,506	14.08	20,905	21,247	917,254	43.88
1905..................	23,731	24,111	370,328	15.61	37,294	37,891	1,522,434	40.82
1906..................	19,091	19,397	248,106	13.00	35,793	36,366	1,590,592	44.44

Year.	Rails.				Hoop, Band or Scroll.				Ingots, Blooms, Slabs, Billets, etc.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900	1,448	1,471	$56,129	$38.77	165	167	$12,409	$75.26	12,709	12,913	$1,332,896	$104.84
1901	1,905	1,935	67,052	35.19	2,974	3,021	116,841	39.29	8,164	8,295	2,340,112	164.15
1902	63,522	64,538	1,576,679	24.82	3,362	3,416	131,052	38.97	289,318	293,965	7,943,818	27.76
1903	95,555	97,083	2,159,273	22.59	1,525	1,550	74,898	49.11	261,570	265,932	7,331,299	28.03
1904	37,776	38,380	808,775	21.41	2,135	2,169	60,934	28.54	(m) 10,807	10,980	1,537,531	142.27
1905	17,278	17,554	409,807	23.72	4,772	4,848	137,612	28.84	(m) 14,641	14,875	2 072,606	141.56
1906	4,943	5,022	137.104	27.74	10.231	10,395	256,836	25.10	(m) 21,337	21,678	3,010,589	141.10

Year.	Sheet, Plate and Taggers Iron or Steel.				Tin Plates, Terne Plates and Taggers Tin.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	5,143	5,226	$426,541	$82.93	60,386	61,356	$4,617,813	$76.46
1901	5,626	5,716	443,880	79.10	77,395	78,638	5,294,789	68.41
1902	7,156	7,270	545,739	76.26	60,115	61,080	4,023,421	66.93
1903	11,557	11,741	540,272	46.75	47,360	48,118	2,999,252	63.33
1904	4,165	4,232	302,500	72.63	70,652	71,782	4,354,761	61.63
1905	2,236	2,272	242,955	108.66	65,740	66,792	4.090,523	62.22
1906	3,231	3,283	325,276	67.35	56,982	57,894	3,883,225	68.15

Year.	Wire-Rods.				Wire and Articles Made from.				Total Iron Imports. (e)
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons	Value.	Value per L. T.	
1900	21,092	21,430	$1,212,594	$57.49	1,848	1,877	$409,087	$221.37	$20,443,911
1901	16,804	17,073	964,744	57.40	4,129	4,192	585,354	141.77	20,404,122
1902	21,382	21,725	1,033,074	48.31	3,469	3,525	606,724	174.90	41,468,826
1903	20,836	21,169	1,028,977	49.39	5,018	5,098	728,430	145.16	41,258,864
1904	15,313	15,558	707,779	46.22	3,956	4,019	624,892	157.96	21,621,970
1905	17,616	17,898	800,027	45.41	3,978	4,042	705,465	177.34	26,401,283
1906	17.799	18,084	876.270	49.23	6,610	6,716	1,079.868	163.37	34,827,132

Year.	Lead in Ore and Base Bullion.				Lead in Pigs and Old.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	Short Tons.	Metric Tons.	Value.	Value per Sh. T.
1900	(g)114,397	103,780	$3,975,695	$34.60	....	....	....	....
1901	111,867	101,486	4,807,762	47.37	604	548	$33,882	$56.10
1902	105,186	95,425	4,424,511	46.37	2,529	2,294	132,500	52.40
1903	103,384	93,790	3,596,635	38.35	3,023	2,742	164,528	54.42
1904	104,127	94,464	3,517,691	37.24	8,724	7,914	461,316	52.88
1905	92,657	84,081	3,565,282	38.48	5,720	5,812	367.106	64.18
1906	72,371	65,640	3,490,776	48.23	11,763	10,669	910,391	77.39

Year.	Lead, Sheet, Pipe, Shot, Etc.				Other Lead Mfrs.	Total Lead.	White Lead.			
	Lb.	Metric Tons.	Value.	Value per Lb.			Lb.	Metric Tons.	Value.	Value per Lb.
1900	27,945	13	$1,393	$0.050	$5,854	$3,964,942	456,872	207	$28,366	$0.062
1901	56,735	26	2,773	.048	4,654	4,849,071	384,673	174	21.226	.056
1902	224,208	102	7,765	.034	18,918	4,533,694	506,423	230	25,320	.050
1903	17,008	8	810	.048	8,071	3,770,044	453,284	206	24,595	.054
1904	69,581	32	2,441	.035	7,755	3,989,203	587,338	266	33,788	.058
1905	54,779	25	2,638	.048	4,580	3,939,606	597,510	271	34,722	.058
1906	346,177	157	17,250	.050	20,681	4,439.098	647,636	294	41.233	.064

Yr.	Litharge.				Red Lead.				Orange Mineral.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	77,314	35	$2,852	$0.032	549,551	249	$25,532	$0.046	1,068,793	485	$61,885	$0.059
1901	49,306	22	1,873	.038	485,467	220	19,370	.040	977,644	443	52,409	.053
1902	88,115	40	2,908	.033	1,075,839	488	37,383	.035	997,494	452	49,060	.049
1903	42,756	19	1,464	.934	1,152,715	523	40,846	.035	756,742	343	36,407	.048
1904	44,541	20	1,500	.034	836,077	379	30,115	.036	766,469	348	37,178	.049
1905	117,757	53	4,139	.035	704,402	320	26,553	.038	628,003	285	31,106	.049
1906	87,230	40	3.737	.043	1,093,619	497	50,741	.046	770,342	350	42,519	.055

Year.	Magnesite.				Manganese Ore.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value	Value per L. T.
1900	....	....	....	....	256,252	260,352	$2,042,361	$7.97
1901	30,350	30,835	....	....	165,720	168,372	1,486,573	8.97
1902	45,157	45,880	$373,928	8.28	235,576	239,345	1,931,282	8.20
1903	49,684	50,479	461,399	9.29	146,056	148,393	1,278,108	8.75
1904	35,106	35,668	286,828	8.17	108,519	110,255	901,592	8.31
1905	66,405	67,566	638,619	9.46	257,033	261,146	1,952,407	7.60
1906	80,711	82,002	863,492	10.70	221,260	224,800	1,696,043	7.67

Year.	Oil, Mineral.				Paints and Colors.	Platinum, Unmanufactured.				Platinum Mfres.
	Gal.	Liters.	Value.	Value per Gal.		Lb. Troy.	Kg.	Value.	Value per Lb. Troy.	
1900...	3,039,094	11,503,913	$274,766	$0.091	$1,491,902	9,246	3,450	$1,728,777	$187.00	$36,714
1901...	2,294,684	8,686,389	151,913	.066	1,524,125	7,496	2,797	1,673,713	223.30	24,482
1902...	3,578,393	13,545,646	207,310	.058	1,745,989	8,670	3,235	1,950,362	224.96	37,618
1903...	4,266,974	16,152,203	261,199	.061	1,811,902	9,540	3,561	1,921,772	201.44	1,727,830
1904...	4,846,681	18,344,650	277,399	.057	1,526,072	8,648	3,230	1,812,242	209.55	105,636
1905...	13,725,720	52,020,479	672,127	.049	1,570,839	8,681	3,240	1,985,107	228.67	188,156
1906...	21,145,216	80,034,643	1,061,076	.050	1,833,938	13,928	5,198	3,601,021	258.54	187,639

Year.	Marble and Stone Mfres.		Metal Composition:		Mica.	Nickel. (*h*)	Nickel Ore and Matte.			
	Marble.	Stone. (*d*)	Bronze.	All Other.			Long Tons.	Metric Tons.	Value.	Value per L. T.
1900..	$945,705	$256,624	$791,306	$5,420,483	$ 319,560	(*f*)	(*f*)	......	....	....
1901..	1,226,524	237,191	945,702	5,162,392	335,054	....	(*f*)	......	....	....
1902..	1,435,457	222,435	816,668	5,942,453	466,332	....	(*f*)	......	....	....
1903..	1,502,111	258,968	790,639	6,402,215	317,969	$207,954	11,936	16,191	$1,285,935	$80.70
1904..	1,270,443	294,035	746,073	5,965,247	269,808	206,021	8,549	8,685	915,470	107.10
1905..	1,348,624	293,055	839,133	7,314,809	403,755	335,211	13,451	13,666	1,626,920	120.95
1906..	1,370,413	355,908	991,182	9,283,836	1,042,608	86,336	15,156	15,398	1,816,631	119.86

Year	Potassium Salts.											
	Chlorate.			Chloride.			Chromate and Bichromate.			Nitrate.		
	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value. per Lb.	Lb.	Value.	Value per Lb.
		$	$		$	$		$	$		$	$
1900	1,243,612	68,772	0.055	130,175,481	1,976,604	0.015	111,761	7,758	0.069	10,545,392	276,664	0.026
1901	811,127	61,348	.076	148,189,337	2,316,577	.015	430,996	29,224	.068	9 656,393	253,286	.026
1902	1,209,148	60,429	.050	140,980,460	2,141,553	.015	231,009	15,161	.066	10,505,474	299,416	.028
1903	468,042	19,308	.041	169,337,673	2,550,478	.015	41,229	2,784	.067	13,835,668	367,721	.026
1904	95,889	4,209	.044	174,865,872	2,832,554	.016	26,053	1,817	.069	14,184,287	376,931	.027
1905	42,510	2,876	.067	214,207,064	3,326,748	.016	59,650	4,225	.070	9,911,534	304,596	.027
1906	45,873	3,103	.068	223,203,387	3,858,895	.017	30,098	2,102	.080	11,326,256	371,595	.033

Year.	Potassium Salts. All Other.			Precious Stones.			Pyrites (*i*).			
	Lb.	Value.	Value per Lb.	Uncut.	Cut, not Set.	Jewelry.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900...	54,904,088	$1,407,303	$0.025	$3,751,219	$9,612,127	(*f*)	332,517	337,837	$1,095,598	$3.30
1901...	72,489,913	1,636,856	.022	6,637,860	17,166,049	(*f*)	398,969	405,353	1,407,244	3.53
1902...	91,857,009	1,820,585	.020	8,282,760	18,494,288	(*f*)	437,319	444.316	1,623,430	3.71
1903...	70,205,850	1,593,380	.023	10,374,877	15,428,819	$954,456	427.319	434,156	1,636,450	3.83
1904...	74,720,241	1,678,699	.023	10,316,615	16,934,090	803,952	413,585	420,202	1,533,564	3.73
1905...	82,935,632	1,891,081	.023	10,206,350	26,699,670	801,566	515,722	520,926	1,780,800	3.47
1906...	30,302,735	763,513	.025	11,937,572	22,201,949	988,766	598,078	607,647	2,148,559	3.59

Year.	Salt.				Silver.		Sodium Nitrate.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	207,933	188,636	$633,192	$3.05	$14,695,965	$25,404,378	182,108	185,022	$4,935,520	$27.12
1901	194,967	176,872	670,648	3.44	12,957,987	18,188,795	208,654	211,992	5,997,595	28.82
1902	188,775	167,481	654,990	3.47	8,502,614	17,900,321	205,245	208,529	5,996,205	29.21
1903	157,201	142,494	489,179	3.11	7,935,844	16,038,664	272,947	277,314	8,700,806	31.88
1904	167,295	151,810	515,822	3.08	11,865,805	14,221,237	228,012	231,660	9,333,613	32.41
1905	158,449	143,783	491,079	3.10	16,472,911	19,466,224	321,231	326,371	11,206,548	34.89
1906	170,505	154,648	503,583	2.95	20,402,738	23,825,103	372,222	378,178	14,115,206	37.92

Year.	Sodium Hydroxide (Caustic).			Soda Ash and Carbonate.			All Other Sodium Salts.		
	Lb.	Value	Value. per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.
1900	8,403,749	$150,530	$0.018	73,815,425	$613,379	$0.008	20,484,938	$259,802	$.013
1901	3,812,847	94,303	.025	31,415,788	276,261	.009	14,491,559	189,543	.013
1902	3,334,697	77,482	.020	31,889,252	284,634	.009	17,151,682	283,745	.016
1903	2,970,426	73,647	.025	25,313,370	228,041	.009	14,272,646	268,738	.019
1904	2,570,984	64,405	.025	23,631,832	205,496	.009	10,399,711	281,527	.027
1905	2,245,789	56,515	.025	15,754,979	146,812	.009	11,257,629	247,413	.0
1906	1,209,053	35,262	.022	6,800,055	70,979	.010	....	....	....

Year.	Sulphur.											
	Crude.				Flowers.				Refined.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900	166,457	169,120	$2,918,610	$17.53	628	638	$17,437	$27.77	243	247	$ 6,279	$25.84
1901	174,162	176,949	3,256,951	18.70	748	761	20,201	26.98	268	272	6,308	23.57
1902	176,951	179,782	3,360,562	19.00	738	750	19,954	27.04	14	15	369	24.99
1903	188,888	191,910	1,649,756	19.32	1,854	1,883	52,680	28.42	189	192	7,254	38.44
1904	128,885	130,947	2,463,779	19.12	1,332	1,353	39,133	29.38	204	207	9,776	47.92
1905	83,201	84,532	1,522,005	18.29	572	581	16,037	28.04	778	790	19,960	25.66
1906	72,559	73,720	1,282,873	17.68	1,100	1,118	29,565	26.87	709	720	17,028	25.29

Year.	Talc.				Tin.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	79	72	$1,070	$13.54	69,989,502	31,747	$19,458,586	$0.278
1901	2,386	2,164	27,015	11.74	74,560,487	33,820	19,024,761	.255
1902	2,859	2,594	35,336	12.35	85,043,353	38,575	21,263,337	.250
1903	1,790	1,623	19,635	11.00	83,133,847	37,702	22,265,367	.268
1904	3,268	2,964	36,370	11.13	83,168,657	37,718	22,356,896	.270
1905	4,000	3,630	48,225	12.06	89,227,698	40,507	26,316,023	.294
1906	5,643	5,118	67,818	12.02	101,027,188	45,816	37,446,508	.371

Year.	Zinc.								
	Blocks, Pigs and Old.				Oxide (j).		Sulphide.		Mfres.
	Lb.	Metric Tons.	Value	Value. per Lb.	Lb.	Value.	Lb.	Value.	Value.
1900......	2,013,196	913	$97,772	$0.048	2,657,514	....	....	....	$36,836
1901......	775,881	352	30,920	.040	3,327,976	....	....	....	42,643
1902......	1,238,091	561	46,713	.038	3,434,466	....	1,247,936	$32,879	37,191
1903......	728,614	330	30,900	.042	3,653,076	$188,495	1,229,806	33,077	18,938
1904......	933,474	423	44,455	.048	2,809,905	165,110	1,228,875	31,382	11,918
1905......	1,042,081	473	51,052	.048	3,779,311	196,220	1,235,360	33,308	12,390
1906......	4,042,486	1,835	240,632	.060	4,494,014	288,065	1,286,469	40,112	17,385

(*a*) From Summary of Commerce and Finance of the United States. (*b*) Includes arsenic sulphide. (*c*) Barrels of 400 lb. (*d*) Including slate. (*e*) Not including iron ore. (*f*) Not reported. (*g*) Includes pig and old. (*h*) Includes nickel oxide, alloys in which nickel is the principal constituent and manufactures of nickel. (*i*) Containing more than 25% sulphur. (*j*) Includes white pigments containing zinc but not lead, dry and in oil. (*m*) Includes bar of steel and steel forms not elsewhere specified

EXPORTS OF DOMESTIC PRODUCTS. (*a*)

Year.	Aluminum and Mfres. of.	Asbestos and Mfres. of.	Brass and Mfres. of.	Brick.	Cement.				Chemicals and Drugs.
					Bbl. (*i*)	Metric Tons.	Value.	Value per Bbl.	
1900.....	$281,821	$124,971	$2,068,072	....	100,400	18,216	225,306	$2.24	$13,771,682
1901.....	183,579	113,316	2,078,178	$541,589	373,934	67,393	679,296	1.82	14,267,110
1902.....	116,052	130,437	1,809,312	501,434	340,821	61,838	526,471	1.54	13,437,367
1903.....	157,187	158,360	2,063,569	439,277	285,463	51,748	433,984	1.52	14,276,465
1904.....	166,876	223,096	3,093,803	587,385	774,940	140,898	1,104,086	1.42	14,929,583
1905.....	290,777	....	3,055,189	799,878	1,016,236	185,345	1,387,706	1.37	17,126,967
1906.....	364,251	259,760	4,180,403	885,066	583,299	105,811	944,886	1.62	18,509,077

Year.	Coal.									
	Anthracite.				Bituminous.				Coke.	
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Value.
1900.	1,654,610	1,681,084	$7,092,489	$4.29	6,262,909	2,363,631	$14,431,590	$2.31	376,999	$1,358,968
1901.	1,993,307	2,025,200	8,937,147	$4.48	5,390,086	3,476,327	13,085,763	2.53	384,330	1,516,898
1902.	907,977	922,505	4,301,946	4.73	5,218,969	5'302,472	13,927,063	2.66	392,491	1,785,188
1903.	2,008,857	2,040,999	9,780,044	4.86	6,303,241	6,404,093	17,410,385	2.76	416,385	2,091,875
1904.	2,228,392	2,264,046	11,077,570	4.97	6,345,126	6,446,648	17,160,538	2.74	523,090	2,311,401
905.	2,229,983	2,265,663	11,104,654	4.98	6,959,265	7,070,613	17,867,964	2.56	599,054	2,243,010
1906.	2.216,969	2,252.441	10,896.200	4.91	(*k*)7,704.850	7.828.128	(*k*)19.787.459	2.57	765.190	2.753.551

Year.	Copper.									
	In Ore and Matte (b).				Ingots, Bars, Plates and Old.				Mfres.	Total Except Ore.
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Lb.	Metric Tons.	Value.	Value per Lb.	Value.	
1900...	10,007	10,168	$1,332,829	$133.18	337,973,751	153,304	$55,285,047	$0.164	$2,257,563	$57,542,610
1901...	19,613	19,924	2,536,549	129.40	194,249,828	88,111	31,692,563	.164	1,842,336	33,384,899
1902...	18,035	18,321	1,326,131	73.53	354,668,849	160,876	43,392,800	.122	2,092,798	45,485,598
1903...	12,291	12,488	855,367	69.59	310,729,524	140,920	41,170,059	.132	2,339,729	43,509 788
1904...	18,927	19,230	1,202,537	63.54	554,550,030	251,497	71,488,116	.129	3,328,818	74,816,934
1905...	37,688	38,291	1,531,429	40.63	534,907,619	242,699	80,693,232	.151	4,184,070	84,890,302
1906...	47,619	48,380	1,760,140	36.96	454,752,018	206,239	84,728,400	.186	4,284,611	89,013,011

Year	Earthen and China Ware.	Fertilizers.								Glassware.	Gold.	
		Crude Phosphates.				All Other.						
		Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.		In Coin and Bullion (c)	In Ore. (d)
1900	$558,794	619,995	629,915	$5,217,560	$8.38	25,976	26,392	$537,908	$20.71	$2,042,633	$54,064,697	$69,926
1901	526,820	729,539	741,212	5,839,245	8.01	14,153	14,379	332,964	23.54	2,087,043	56,717,350	1,012,589
1902	604,646	802,086	814,919	6,193,372	7.73	16,451	16,714	383,438	23.31	2,094,701	35,722,835	307,756
1903	589,001	785,259	797,823	6,109,230	7.78	20,343	20,668	557,059	27.38	2,053,516	43,765,360	581,474
1904	791,739	842,484	855,964	6,521,555	7.74	25,549	25,958	714,367	27.96	2,130,297	120,226,424	985,403
1905	983,554	934,940	949,899	7,465,592	7.91	22 865	23,231	723,744	32.05	2,252 399	46,099,580	694,887
1906	1,118,450	904,214	918,681	7,373,945	8.16	31,999	32,511	1,088,004	34.00	2,534,271	46,068,451	640,707

Year	Gunpowder and Other Explosives.	Iron.							
		Ore.				Pig.			
		Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value Per L. T.
1900........	....	51,460	52,283	$154,756	$3.01	286,687	291,404	$4,654,582	$16.23
1901........	$1,965,875	64,703	65,748	163,465	2.54	81,211	82,510	1,257,699	15.65
1902........	2,393,480	88,445	89,860	294,168	3.32	27,487	27,927	502,947	18.30
1903........	2,367,148	80,611	81,901	255,728	3.17	20,379	20,705	384,334	18.86
1904........	2,466,278	213,865	217,287	458,823	2.14	49,025	49,809	764,543	15.60
1905........	2,848,155	208,017	211,345	530,457	2.55	49,221	50,009	762,899	15.50
1906........	3,793,043	265,240	269,484	771,831	2.91	83,317	84,650	1,506,774	18.08

Year.	Iron, Bar.				Iron, Band, Hoop and Scroll.				Billets, Ingots and Blooms.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900...	13,298	13,512	$558,576	$42.04	2,976	3,024	$137,437	$46.20	107,385	109,103	$2,915,371	$27.15
1901...	17,708	17,993	674,671	38.16	1,561	1,586	74,056	47.44	28,614	29,072	708,887	24.78
1902...	22,249	22,605	869,519	39.08	1,674	1,701	82,322	49.18	2,409	2,447	74,938	31.11
1903...	19,380	19,690	796,631	41.11	1,241	2,175	101,839	47.56	5,445	5 532	141,924	26.07
1904...	29,582	30,055	1,133,128	34.93	3,435	3,489	162,039	47.18	314,324	319,353	6,150,035	19.56
1905...	32,025	32,537	1,255,418	39.20	4,426	4,497	182,431	41.22	237,738	241,542	4,701,909	19.79
1906...	56,024	56,920	2,575,905	45.98	5,405	5,491	242,776	44.92	192,616	195,698	4,094,659	21.26

Year.	Iron, Nails and Spikes, Cut.				Iron, Nails and Spikes, All Other.				Iron, Plates and Sheets.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Met. Tons.	Value.	Value per L. T.
1900	25,005,308	11,342	$626,497	$0.025	65,444,387	29,681	$1,816,813	$0.028	9,331	9,481	$600,600	$64.35
1901	20,835,944	9,452	450,331	.021	46,298,262	21,001	1,152 368	.025	6,909	7,020	452,695	65.52
1902	16,122,775	7,312	339,227	.021	64,565,650	29,287	1,456,768	.022	3,434	3,489	229,887	66.94
1903	19,912,563	9,031	424,985	.021	75,654,532	34,310	1,698,500	.024	4,782	4,858	273,618	57.22
1904	20,772,049	9,422	416,389	.020	80,279,746	36 403	1,949,908	.024	4,728	4,804	247,694	52.39
1905	17,674,099	8,019	352,405	.020	89,976,088	40,506	2,118,836	.024	8,004	8,132	460,995	57.60
1906	16,951,893	7,688	340,526	.020	116,310,428	52,747	2,731,021	.024	17,054	17,327	1,139,526	66.81

Year.	Steel, Sheets and Plates.				Iron Rails.				Steel Rails.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900.	45,534	46,264	$1,638,478	$35.98	5,374	5,460	$119,206	$22.18	356,445	361,945	$10,895,416	$30.58
1901.	23,923	24,303	959,471	40.11	901	915	32,357	35.93	318,055	323,044	8,628,781	27.14
1902.	14,866	15,104	725,547	48.80	211	214	4,639	22.02	67,455	68,534	1,902,396	28.09
1903.	13,312	13,525	657,713	49.47	181	184	8,808	48.67	30,656	31,146	937,779	30.59
1904.	50,477	51,278	2,064,241	40.89	1,405	1,427	23,870	17.00	414,845	421,482	10,661,222	25.72
1905.	67,093	68,166	2,889,084	43.06	*Nil.*	....	....	....	295,023	299,473	7,310,029	24.78
1906.	93,601	95,099	4,081,915	43.61	*Nil.*	....	....	....	328,030	333,285	8,903,411	27.14

Year	Structural Iron and Steel.				Wire.				Steel Wire Rods.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900.	67,714	68,797	$3,570,769	$52.73	78,014	79,262	$4,604,047	$59.77	10,652	10,822	$505,529	$47.37
1901.	54,005	54,869	3,031,861	56.10	88,238	89,650	4,805,608	54.36	8,165	8,296	271,552	33.26
1902.	53,859	54,721	2,828,460	52.52	97,843	99,414	5,140,702	52.54	24,613	25,007	831,067	33.76
1903.	30,641	31,131	1,788,556	58.37	108,521	110,258	5,528,726	50.94	22,360	22,718	713,718	31.92
1904.	55,514	56 402	2,777,768	50.04	118,581	120,478	5,935,093	50.05	20,073	20,394	695,448	34.64
1905.	84,234	85,582	4,357,186	51.73	142,601	144,883	7,061,442	49.52	6,514	6,618	277,651	42.62
1906.	112,555	114,356	6,140,861	54.56	174,014	176,798	8,770,042	50.40	5,896	5,990	221,679	37.60

Year.	Lead and Mfres. of.	Marble, Stone.	Mica.	Nickel. (*e*)	Petroleum products. (In Thousands of Units.)*							
					Crude.				Naphtha.			
					M Gals.	M Liters.	M Value.	Value per Gal.	M Gals.	M Liters.	M Value.	Value per Gal.
1900	$459,574	$1,556,981	$165	$1,382,727	138,161	523,000	$7,341	$0.053	18,570	70,295	$1,681	$0.081
1901	625,234	1,785,515	3,584	1,521,291	127,008	480,781	6,038	.050	21,685	82,087	1,742	.079
1902	696,010	1,587,957	....	924,579	145 234	549,775	6,331	.042	19,683	74,509	1,393	.071
1903	491,362	1,688,316	760	703,550	126,512	478,847	6,782	.054	12,973	49,103	1,519	.118
1904	616,126	1,337,754	1,770	2,130,933	111,176	420,801	6,351	.057	24,989	94,583	2,322	.093
1905	667,861	1,384,208	....	2,894,700	126,185	447,610	6,086	.048	28,420	107,570	2,215	.078
1906	775,776	1,391,667	....	3,493,643	148,045	560,350	7,731	.052	27,545	104,258	2,488	.090

Year.	Petroleum Products. (In Thousands of Units.)*											
	Illuminating Oil.				Lubricating Oil.				Residue, Etc. (*g*)			
	M Gals.	M Liters.	M Value.	Value per Gal.	M Gals.	M Liters.	M Value.	Value per Gal.	M Gals.	M Liters.	M Value.	Value per Gal.
1900...	739,163	3,104,593	$54,693	$0.074	71,211	269,540	$9,933	$0.139	19,750	74,760	$845	$0.042
1901...	827,479	3,131,399	53,491	.065	75,306	285,010	10,260	.136	27,596	104,463	1,255	.046
1902...	778,801	2,947,762	49,079	.063	82,201	311,163	10,872	.133	38,316	145,043	922	.024
1903...	691,837	2,618,603	51,356	.074	95,622	361,929	12,690	.133	9,753	36,916	282	.029
1904...	761,358	2,881,740	58,384	.077	89,738	339,469	12,389	.138	34,904	132,072	1,174	.034
1905...	881,450	3,336,288	54,901	.062	113,730	430,468	14,312	.126	70,728	267,705	2,128	.030
1906...	878,284	3,324,305	54,858	.063	151,269	572,553	18,690	.124	64,645	244,681	1,971	.030

Year.	Petroleum Products.* Paraffin.				Quicksilver.				Silver.	
	M Lb.	M Metric Tons.	M Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.	In Coin and Bullion (c)	In Ore (d)
1900	157,108	71.2	$8,186	$0.052	778,191	353	$425,812	$0.547	$65,705,909	$515,755
1901	151,694	68.8	7,960	.052	843,938	383	475,609	.563	55,526,975	111,383
1902	175,269	79.5	8,398	.048	1,013,434	459	575,099	.568	49,228,303	44,651
1903	204,120	92.6	9,596	.047	1,344,615	610	719,119	.535	4,0531,095	79,247
1904	174,582	79.2	8,273	.047	1,611,365	731	847,108	.526	49,975,370	159,875
1905	160,836	73.0	7,873	.049	1,009,446	458	497,470	.493	54,133,721	3,379,381
1906	173,504	72.9	8,463	.049	484,151	219	244,299	.505	57,012,104	3,944,987

Year.	Tin Mfres.	Zinc Ore.				Zinc Pigs, Bars, Plates and Sheet.			
		Long Tons.	Metric Tons.	Value.	Value per L. T.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	$ 467,354	37,555	38,158	$1,133,633	$30.19	44,802,577	20,322	$2,217,693	$0.050
1901	495,435	39,425	40,056	1,167,684	29.62	6,780,221	3,071	288,906	.043
1902	529,061	49,762	50,558	1,449,104	29.12	6,473,135	2,936	300,557	.046
1903	777,917	35,188	35,751	987,000	28.05	3,041,911	1,380	163,379	.053
1904	701,625	32,063	32,576	905,782	28.25	20,145,942	9,204	1,094,490	.053
1905	930,844	27,630	28,072	848,451	30.71	11,031,815	5,005	682,254	.062
1906	1,120,410	24,750	25,146	733,300	29.63	9,340,455	4,236	583,526	.062

Year.	Zinc Oxide.				Zinc Mfres.	Total (Except Ore).
	Lb.	Metric Tons.	Value.	Value per Lb.		
1900	11,391,666	5,167	$496,380	$.044	$99,288	$2,813,361
1901	9,122,283	4,138	393,259	.043	82,046	764,211
1902	10,716,364	4,861	433,722	.040	114,197	848,476
1903	14,429,885	6 544	578,215	.041	71,354	812,948
1904	16,313,826	7,399	628,494	.039	117,957	1,840,941
1905	22,559,625	10,236	810,203	.036	159,995	1,652,452
1906	31,156,616	14,129	1,149,297	.037	204,269	1,937,092

RE-EXPORTS OF FOREIGN PRODUCTS. (a)

Year.	Antimony.				Antimony Ore.			
	Lb.	Metric Tons.	Value.	Value per lb.	Short Tons.	Metric Tons.	Value.	Value per Sh. T
1900	23,520	10.7	$2,352	$0.100	*Nil.*	....	....	....
1901	*Nil.*	....	....	....	25	22.1	$1,536	$63.05
1902	37,184	16.9	2,710	.073	104	94.6	4,602	44.13
1903	79,917	36.0	4,478	.056	*Nil.*	....	....	....
1904	31,077	14.0	1,734	.056	214	194.0	10,775	50.35
1905	*Nil.*	....	....	....	*Nil.*	....	....	....
1906	24,892	11.2	4,939	.019	*Nil.*	....	....	....

Year.	Asphaltum, Crude.				Brass and Mfres.	Cement.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.		Bbl. (i)	Metric Tons.	Value.	Value per Bbl.
1900	629	639	$10,044	$15,98	$2,155	39,540	7,174	$63,880	$1.62
1901	2,209	2,244	18,078	8,19	813	43,691	7,927	72,761	1.67
1902	2,930	2,977	23,564	8,11	938	32,594	5,913	48,797	1.50
1903	1,605	1,631	13,894	8,66	7,576	25,362	4,601	32,156	1.27
1904	1,887	1,917	26,272	13,92	2,517	39,711	7,186	54,486	1.37
1905	1.081	1,098	18,190	16,83	1,228	31,874	5,782	40,583	1.27
1906	1,765	1,793	22,324	12,64	....	16,216	2,941	19,487	1.20

Year.	Chemicals.											
	Salts of Potassium. (f)				Chloride of Lime.				Nitrate of Sodium.			
	Lb.	Kg.	Value.	Value per Lb.	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900...	808,701	366,824	$43,524	$0.054	148,116	67,185	$1,987	$0.014	3,089	3,139	$112,550	$36.43
1901...	633,100	287,182	43,446	.068	13,916	6,312	312	.023	2,482	2,519	101,489	40.90
1902...	1,266,145	574,323	59,789	.048	198,794	90,172	2,997	.014	3,675	3,734	144,650	39.36
1903...	1,299,905	589,637	33,264	.026	836,411	379,696	7,609	.009	4,417	4,488	184,657	41.81
1904...	1,262,222	572,544	33,358	.027	1,434	650	13	.009	6,076	6,173	279,864	46.06
1905...	3,053,191	1,386,149	83,652	.027	100	102	3	.003	8,991	9,135	420,613	46.78
1906...	2,264,175	1,027,935	77,043	.034	*Nil.*	....	....	....	6,660	6,767	324,915	48.78

Year.	Chemicals. (*Continued.*)											
	Caustic Soda.				Soda Ash and Carbonate.				Sodium Salts, All Other.			
	Lb.	Kg.	Value.	Value per Lb.	Lb.	Kg.	Value.	Value per Lb.	Lb.	Kg.	Value.	Value per Lb.
1900......	1,139,954	517,080	$24,228	$0.023	78,017	35,388	$1,126	$0.014	270,307	122,610	$2,788	$0.009
1901......	1,001,940	452,482	21,511	.023	369,521	167,614	5,184	.014	133,400	60,510	3,398	.023
1902......	1,343,132	609,246	28,704	.023	62,653	28,419	931	.014	115,491	52,386	1,626	.014
1903......	1,116,354	506,378	23,227	.028	30,030	13,622	464	.015	42,540	19,294	437	.010
1904......	1,115,600	506,036	23,608	.021	40.351	18,303	593	.014	1,778,616	806,780	25,312	.014
1905......	1,087,772	493,848	22,728	.021	32.221	14,628	473	.015	16,748	7,604	177	.011
1906......	(l)	....	....	....	2,486	....	41	.016	1,032,372	468,180	21,624	.021

Year.	Clay or Earths.				Coal, Bituminous.				Copper.			
									Ore and Matte.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900........	78	79	$572	$7.34	6,740	6,848	$19,740	$2.93	964	979	$170,191	$176.53
1901........	80	81	825	10.34	3,796	4,403	10,627	2.45	9,891	10,050	1.406,648	142.19
1902........	123	125	1,284	10.43	7,559	7,680	22,153	2.93	14,446	14,657	2,229,912	154.57
1903........	88	89	621	7.06	88,468	89,883	453,613	5.13	5,750	5,232	852,726	165.58
1904........	210	214	2,466	11.74	7,250	7,366	21,910	3.02	*Nil.*	......	....	....
1905........	102	104	954	9.35	3,945	4,008	10,974	2.78	*Nil.*	......	....	....
1906........	140	142	1,291	9.22	2,541	2.582	13,062	5.14	71	72	29,791	419.59

Year.	Copper. (*Continued.*)					Earthen, Stone and China Ware	Fertilizers. Total Value.	Glass and Glassware.	Graphite.	
	Pigs, Bars, Ingots, Old and All Unmanufactured.				Mfres.					
	Lb.	Metric Tons.	Value.	Value per Lb.					Long Tons.	Value.
1900...	1,281,782	581	$212,264	$0.166	$21,032	$38,008	$32,102	$14,614	3	$115
1901...	12,888,083	5,846	2,145,468	.166	9,462	24,080	2,833	16,749	*Nil.*	....
1902...	11,629,877	5,275	1,604,522	.138	10,939	18,989	31,476	34,236	12	834
1903...	2.093.103	949	261,413	.125	13,027	19,411	3,281	19,116	63	4,223
1904...	1,088,672	494	140,695	.129	19,461	32,640	139,363	20,522	8	455
1905...	1,718,584	780	272,945	.159	12,621	30,455	8,984	34,552	5	91
1906...	1,567,782	711	309,605	.197	16,589	41,151	15,568	19,965	3	362

Year.	Iron and Steel.											
	Pig Iron.				Scrap.				Bar Iron.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900....	151	153	$6,579	$43.69	9,079	9,224	$131,241	$14.46	48	49	$2,447	$51.00
1901....	189	191	6,148	32.70	3,331	3,384	51,663	18.51	67	68	7,569	113.00
1902....	250	254	6,286	25.14	1,542	1,567	25,020	a 16.23	22	22	1,875	86.56
1903....	1,863	1,893	33,996	18.25	262	266	2,862	10.92	16	16	2,108	130.93
1904....	1,646	1,672	25,910	15.74	190	193	2,367	12.46	7	7	765	102.55
1905....	1,010	1,026	29,047	28.76	4,270	4,338	80,623	18.88	22	22	2,556	118.18
1906.. .	6,750	6,858	236,957	35.10	5,111	5,193	101,886	19.93	61	62	7,207	118.15

Year.	Iron and Steel. (*Continued.*)											
	Rails.				Steel, Ingots, Blooms, Etc.				Sheets, Plates, Rods, Wire.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900....	*Nil.*	....	....	....	2	2	$1,342	$559.33	209	213	11,599	$55.41
1901....	*Nil.*	....	....	....	2	2	1,059	504.30	190	193	17,272	90.81
1902....	297	302	$7,184	$24.20	106	108	6,774	64.09	236	240	14,221	60.26
1903....	739	751	17,560	23.76	60	61	5,316	88.01	55	56	5,532	100.04
1904 ..	96	98	2,305	24.00	40	41	6,208	154.81	108	110	6,482	60.01
1905....	31	31	1,132	36.52	86	87	15,570	181.05	161	164	8,019	49.81
1906....	Nil	....	....	....	196	199	14,104	71.96	318	323	27,631	86.89

Year.	Iron and Steel. (*Concluded.*)					Lead and Mfres.	Marble and Stone Mfres (*h*)	Metal Composition
	Tin and Terne Plates, Taggers Tin.				Mfres.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.				
1900..	464	470	$37,395	$80.60	$328,704	$3,843,881	$5,721	$79,218
1901..	118	120	8,519	72.20	149,771	4,190,525	17,063	35,438
1902..	98	100	7,471	76.24	242,225	3,553,144	11,210	108,575
1903..	2	2	184	118.71	399,147	2,917,957	7,389	54,490
1904..	81	82	5,306	65.86	518,564	2,880,907	21,055	42,730
1905..	26	26	3,014	115.92	544,321	2,441,166	13,653	92,227
1906..	0.4	0.4	28	70.00	200,489	2,307,345	20,966	85,955

Year.	Paints and Colors.	Salt.				Sulphur—Crude.			
		Lb.	Metric Tons.	Value	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900.....	$13,814	3,548,724	1,610	$3,907	$0.0011	590	599	$13,495	$22.89
1901.....	17,923	3,699,411	1,678	7,155	.0019	207	210	5,086	24.60
1902.....	14,217	2,310,759	1,048	4,544	.0020	1,253	1,273	28,024	22.37
1903.....	13,467	7,804,215	3,585	26,636	.0034	967	982	22,658	23.43
1904.....	11,888	2,089,234	948	2,814	.0013	2,493	2,533	58,887	23.62
1905.....	14,227	611,912	278	893	.0015	1,713	1,741	36,858	21.52
1906.....	12,775	1,462,413	743	1,129	.0008	403	409	8,475	21.03

Year.	Tin in Blocks, Pig and Granulated.				Zinc and Mfres.
	Long Tons.	Metric Tons.	Value.	Value per L. T.	
1900	495	503	$335,377	$677.96	$3,048
1901	939	954	562,350	598.89	1,641
1902	479	486	286,897	598.95	765
1903	512	520	317,805	620.47	2,362
1904	519	527	322,234	620.87	1,236
1905	557	567	375,763	674.62	1,831
1906	807	820	650,411	805.96	2,214

*For convenience in tabulating, the quantities of all the petroleum products and their gross values have been divided by 1,000.

(a) From Summary of Commerce and Finance of the United States.

(c) Total exports of coin and bullion; that is, includes both foreign and domestic.

(d) Only approximately correct. The Bureau of Statistics reports only the value of silver ores exported, but a much larger amount of silver leaves the country in copper matte, which is classified as copper ore, and no record is kept of its silver contents. The gold in copper matte exported is not included in the exports of gold given in the above table. These figures include ore of both domestic and foreign origin.

(e) Includes nickel oxide and nickel matte.

(f) Includes chlorate, chloride, nitrate and all other salts of potassium.

(g) Reported in barrels, but calculated to gallons, on a basis of 42 gallons to the barrel.

(h) Includes slate.

(i) Barrel of 400 lb.

(k) Does not include 5,125.103 tons of coal valued at $16,202,150, used for fuel on vessels for foreign trade.

(l) Included in all other salts of soda.

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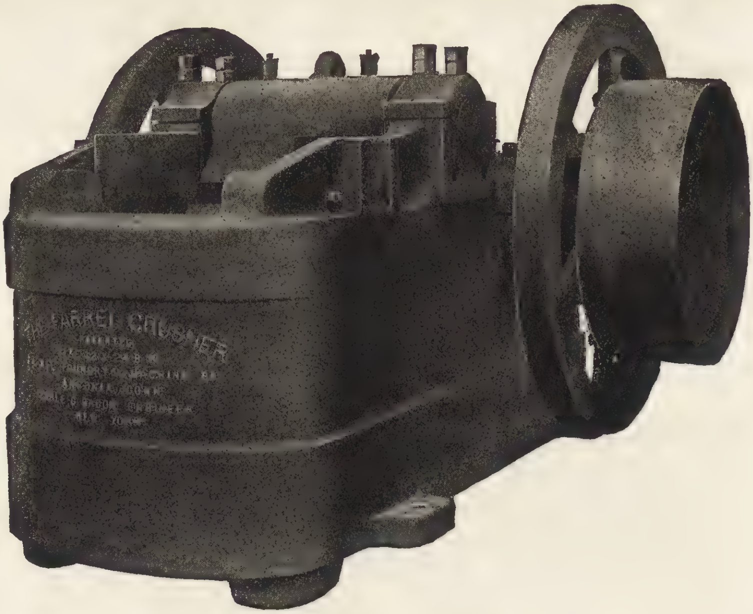

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